Using the Aegis underground drill and blast analyser to optimise drill patterns: A case study

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Abstract

Efficient drill-and-blast performance is pivotal to the profitability of deep-level mines, where high operating costs and declining grades demand maximum throughput. This study investigated persistent fragmentation problems at a longhole stoping operation that suffered from simultaneous oversized rock blockages in 300 mm × 300 mm grizzlies and excessive fines. Fifty-one muckpile images from five blasted stopes were analysed with Split-Desktop. Although the overall F_{80} was 287 mm, nominally suitable for the grizzly aperture, the Rosin-Rammler uniformity index of 0.80 revealed a highly inconsistent size distribution containing both boulders and fines. Existing drill patterns were then evaluated with the Aegis underground software, which showed negligible crack overlap at the toe owing to inadequate spacing and burden. Insufficient energy overlap between blastholes, particularly at the toe, was found to contribute to the coarse fragmentation, while excessive fines were attributed to the crushing and severely fractured zones of blastholes. Optimisation simulations demonstrated that an 89 mm blasthole diameter on a 2 m × 2 m pattern would overlap, aligning with Aegis requirements of balancing powder factor, break angle, and energy distribution. Implementing this revised pattern is expected to reduce secondary breaking, prevent ore-pass blockages, lower fines, and ultimately improve overall production efficiency. The methodology illustrates how integrated image analysis and modelling software can effectively diagnose and address fragmentation issues, particularly in underground longhole mining, where challenging blasting conditions require a balance between achieving optimal fragmentation and maintaining stope stability. This approach offers a cost-effective pathway to improving comminution performance in deep-level mines.

Keywords

slope failure, risk, geotechnical controls, drilling patterns, longhole stoping, ring blasting, fragmentation, optimisation, Rossin-Rammler, Aegis

Introduction

The mining of gold in the Witwatersrand Basin in South Africa began over a century ago. The shallow high-grade gold deposits are depleted and, as a result, mines now exploit lower grade ore at deep to ultra-deep depths (Neingo, Tholana, 2016), which necessitates the optimisation of all mining processes to derive maximum value and consequently high profits from the operation.

Improving the flow of ore from the stopes to the stockpile/crusher was identified as one area of interest to improve margins. According to Shelswell and Labrecque (2014), "a fundamental aspect of an efficient mining operation is the steady movement of material throughout the mine system; particularly the flow of ore from the upstream excavation point to the downstream processing or stockpile site(s)". Thus, a mine's efficiency and subsequent throughput can be achieved by attaining an optimal fragmentation size from drilling and blasting activities.

Multiple fragmentation optimisation studies, such as those by Afum and Temeng (2015), Cebrian et al. (2017), Duranovic et al. (2018), Marton and Cookes (2000), Ninepence et al. (2016), Preston et al. (2019), and Strelec et al. (2011), have highlighted the importance of achieving the correct fragmentation profile in all mining operations to match downstream activities to improve the overall efficiency and reduce operational costs.

The project discussed in this paper was conducted at an underground gold mine in South Africa. The mine uses ring blasting to extract gold from the Ventersdorp Contact Reef (VCR) and the Upper Elsburg Reef in the Witwatersrand Basin. Access to the stopes is through the main access drive (MAD), which leads the development advance. The stope access drives (SADs) break away from the MAD and extend up to the regional pillars at the limit of the orebody.

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To exploit the massive orebody and achieve production targets safely and economically, while controlling the behaviour of the rock mass, a series of primary and secondary longhole stopes are excavated in sequence using the checkerboard technique (Villaescusa, 2003). The technique entails the extraction of primary stopes while advancing, followed by backfilling. Once the backfill cures, stopes adjacent to the previously backfilled primary stopes (referred to as secondary stopes) are extracted on retreat and subsequently backfilled (Villaescusa, 2003). Thus, care is exercised in secondary stopes to prevent dilution of ore by over-drilling into the backfilled primary stopes. Duranovic et al. (2018) and Preston et al. (2020) highlighted challenges experienced in underground ring blasting as:

- Complex pattern designs because of complex orebody shapes.
- The concentration of energy distribution at the collar regions because of obliquely drilled blastholes.
- Inaccessibility and invisible open face.
- Requirement for perimeter control to prevent dilution.

The optimisation of fragmentation has a direct impact on the ore flow system efficiencies. When the fragmentation size is optimal for all infrastructure and transportation equipment, ore pass blockages are reduced, and ore flow efficiencies improve. Dislodging an ore pass is a dangerous activity, which, when avoided, can drastically improve safety and minimise stoppages. Avoiding blockages can improve the probability of achieving production targets and simultaneously reduce operating losses. The research discussed in this paper was thus aimed at enabling the mine to achieve a steady state production target of 254 kt/month.

Objectives and research method

The objectives of the research were to analyse the flow of ore from the stopes to the stockpile to determine bottlenecks and inefficiencies in the system; analyse the current blast design patterns and resulting fragmentation size using the Aegis drill and blast software; and conduct software simulations to determine the correct drill and blast patterns that would provide optimal fragmentation size distribution.

The baseline mapping of the flow of ore from the stopes to the stockpile was conducted to gain an understanding of the background of the project and the processes involved. Various sources of literature and data collection methods, such as a review of company internal documents and records were consulted to add to the knowledge and build a research case. The insights gathered from this exercise were used to identify the possible bottlenecks in the system.

As mentioned, ring blasting presents unique drilling and blasting challenges compared to surface blasting. Thus, peer-reviewed journal articles were surveyed to understand the factors and important considerations unique to designing ring blasts to achieve optimal fragmentation.

A Ricoh WG digital compact camera was used to capture 51 images of fragmented rock resulting from five separate blasts. The selection of stopes was guided by operational availability, focusing on those that had been recently blasted and were accessible during underground site visits. Although this approach introduces an element of convenience sampling, measures were implemented to minimise potential bias. To account for variability within each muckpile, images were taken at different loading intervals within each stope. Figure 1 shows an image of a loaded load haul dump (LHD) scoop with a spherical ball for fragmentation analysis.

Figure 1—An image of a loaded LHD scoop with a spherical ball added to the payload for the purpose of fragmentation analysis

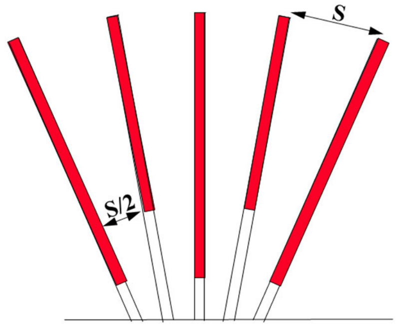


Figure 2—Ideal charging pattern for ring blasting (Cunningham, 1992)

A light source was used to illuminate the muckpile, considering the dark underground working conditions, to obtain clearer pictures. A soccer ball with a diameter of 15 cm was used as a scaling object, and the Split-Desktop software was used to analyse the images to gain an understanding of the fragmentation distribution size based on the work of Bobo (2009) and Kansake et al. (2016).

The Aegis drill and blast software were used to evaluate the impact of the drill and blast patterns used by the mine at the time. Furthermore, a simulation was conducted with the software for optimal drill and blast patterns to yield an optimal fragmentation distribution size.

Literature review

Ring blasting

Ring blasting is a complex technique for rock breaking due to its geometry and requirements for stope perimeter control, and blastholes are drilled radially from a central point to the limit of a block. This results in the convergence of blastholes at the collar while they are further apart at the toes. As a result, the distribution of explosive energy is often inconsistent. Without proper planning and design, higher powder factors will be experienced at the collar as opposed to the toe. As a result, finer fragmentation will emanate from the collar region while coarser fragmentation and possibly boulders will come from the toe region. Cunningham (1992) in Figure 2 suggested the AEL method of charging blastholes in an attempt to distribute the explosive energy for optimal fragmentation.

The method in Figure 2 is based on using various charge lengths at the collar of the blastholes to distribute explosives energy (Cunningham, 1992; Onederra, Chitombo, 2007).

Mechanism of rock breakage

Drill-and-blast is the primary rock-breaking technique in hard rock mines. Although rock breakage is mainly attributed to blasting with explosives, drilling also plays a role in rock breakage. According to Rossmanith et al. (1997), micro-cracks formed by drilling may extend radially and join preexisting cracks occupying a layer of about 2 mm around the blasthole. The creation and interaction of the cracks from adjacent blastholes result in rock fragmentation. Figure 3 illustrates the mechanism of rock breakage by explosives.

Following the detonation of an explosive, a transition from micro-cracking to macro-cracking and fracturing of the rock is observed and can be described using the zones of damage as illustrated in Figure 3. During the detonation of an explosive, a compressive shockwave and high-pressure gases are generated and at the same time, exert intense stresses on the surrounding rock mass while moving radially outward from the centre of the blasthole (Jimeno et al., 1995; Zolezzi, 1961). While the compressive shock wave induces tensile stresses in the tangential plane of the shock front and consequently creates new cracks, the high-pressure gases penetrate and expand new and preexisting cracks in the rock. The combination of the interaction of the compressive shock wave, the high-pressure gases within the rock, and the free face to allow for rock movement or heave, result in rock fragmentation. Further, when the compressive shockwave reaches a free face, it is reflected as a tensile shock wave with an associated shear failure, developing more cracks that result in rock fragmentation (Zolezzi, 1961).

The degree of fragmentation can be controlled by designing blast patterns that take into consideration the optimal interaction of controllable and uncontrollable factors. Uncontrollable rock parameters are characterised by two main geological properties of rock, namely, mechanical properties and physical properties while controllable parameters are divided into three groups, namely the geometric group, the physicochemical group, and the time group (Jimeno et al.,1995).

Given the complexity of ring blasting designs and all the factors that contribute to a successful blast, manual designs of blast patterns are often time consuming and tedious. Thus, a ring blast design computer software and good engineering judgement are necessities for an efficient mining operation (Cunningham, 1992; Chitombo, Onederra, 2007).

Underground drill and blast software

There are various underground drill and blast models embedded in software that enable the evaluation and design of ring blasts. The Aegis underground drill and blast software is one such software. This mechanistic model considers the mechanism of rock breakage while considering uncontrollable and controllable factors, and it uses mathematical models in its design and analysis.

The software uses four model components as discussed in Preston et al. (2019) to estimate optimal pattern designs that result in desired results such as optimal fragmentation and the minimisation of dilution. These models are the unit charge model that links together the rock properties, explosive properties, and the blast pattern model; the thermodynamic energy model used in the calculation of the available energy to break the rock at a specific charge diameter; the stress reflection model used to estimate the strength of the shock wave when it is reflected as a tensile wave at the free face; and the radial break model to determine the break radius of the blasthole. Preston et al. (2019) successfully optimised ring blasts pattern designs, perimeter control, and minimisation of dilution at the Eleonore mine in northern Quebec and a mine in northern Ontario using the Aegis software.

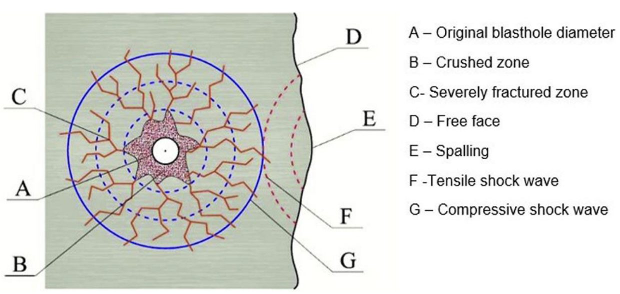


Figure 3—Mechanism of rock breakage by explosives (Strelec et al., 2014)

Results

Ore flow mapping

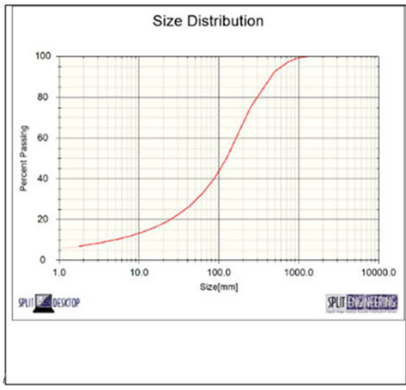
After blasting the rings in the stopes, LHDs load and dump the fragmented rock into mine trucks, which in turn tip the rock into the ore pass tips fitted with 300 mm by 300 mm aperture grizzlies. The track bound locomotives then tram the ore from ore passes underneath the tips to the main level station tip. The ore goes through a series of ore passes to the shaft bottom, where a jaw crusher crushes the rock down to −185 mm. A series of belt conveyors then feed the crushed rock into the skips via measuring flasks for hoisting to the surface. The skips, via the head gear bin, feed the rock into another series of belt conveyors for transportation to either the stockpile or semi-autogenous grinding (SAG) mill.

The analysis of the ore flow revealed that delays occurred mostly in the stopes and at the tips due to coarse and inconsistent fragmentation, which is unable to pass through the grizzlies. Sometimes secondary blasting in the stopes is required to reduce the size of the resulting boulders, and rock breakers at the tips are used to break boulders trammed by LHDs, which often results in production delays.

Fragmentation distribution size analysis

Achieving the correct fragmentation size has a major influence on the ore recovery, ore transportation efficiency, and crusher performance. Therefore, the size of the rock fragmented in the stopes must be below 300 mm, but at the same time not too fine to avoid gold losses in the stopes. An analysis of the fragmentation size distribution achieved by the mine based on the five blasts from five different stopes is shown in Figure 4.

The average F20, F50, F80, and maximum fragment size achieved were 24.64 mm, 124.35 mm, 287.48 mm, and 1 288.40 mm, respectively. This means that cumulatively, 20% of the fragmented rock was smaller than 24.64 mm, while 80% was smaller than 287.48 mm. The mean fragment size (F50) was 124.35



Mine	
% Passing	Size [mm]
F10	5.04
F20	24.64
F30	53.02
F40	87.33
F50	124.35
F60	165.53
F70	215.77
F80	287.48
F90	427.94
Top size (99.95%)	1 288.40
Ros-Ram Uniformity	0.80

Figure 4—Inferred fragmentation size distribution at the mine

mm. These values clearly illustrate the wide variation in the size distribution of the muckpile; from very fine particles to extremely coarse fragments. The presence of a significant proportion of fines (F20 = 24.64 mm) alongside boulders exceeding 1 200 mm in size points to a highly inconsistent fragmentation outcome. Such a distribution can negatively affect both ore transportation efficiency and gold recovery, reinforcing the need for a more uniform fragmentation profile through improved drill and blast design.

Although the F80 value of 287 mm suggests that 80% of the muckpile fragments were smaller than the 300 mm × 300 mm grizzly aperture, implying that the average fragmentation was acceptable, this concealed the inherent problem in the blasted muckpile. The muckpile contained fragments of up to 1 288 mm in size, well over four times that of the grizzly opening. Combined with a Rosin-Rammler uniformity index of 0.8, this indicates a fragmentation size distribution of low uniformity with substantial deviation between fines and coarse fragments (Onederra, 2005; Singh et al., 2016; Vesilind, 1980; Wimmer et al., 2015). The low uniformity index observed in this study explains the dual fragmentation problem: The processing plant received excessive fines in the ROM feed, while significant time was also spent on secondary blasting of large rocks in the stopes and on mechanical breaking at the grizzly tips. As a result, although the bulk of material may pass through the grizzly, the remaining oversized boulders cause frequent blockages and delays. This highlights a critical limitation of relying solely on the F80 value to assess fragmentation quality. Operationally disruptive outliers can still occur in a poorly controlled size distribution. To understand the cause of the presence of fines and coarse fragmentation size distribution, an analysis of the drill patterns was done.

Pattern analyses

Table 1 shows the blast design and explosive parameters that yielded the fragmentation distribution size shown in Figure 4

A 2 m burden between rings was maintained for all blasts with the spacing between toes planned to vary between 2 m and 2.4 m. The toe spacing is determined as the perpendicular distance from the end of a blasthole to the next adjacent blasthole in the same ring as stipulated by the Julius Kruttschnitt Mineral Research Centre (JKRMC) method of measuring toe spacing (Onederra, Chitombo, 2007; Williams, 2019).

The toe spacing in ring blasting is generally based on spacing to burden (S/B) values depicted in Table 2. Although the exact ratio of S/B is not critical in ring blasting, it is generally accepted that the ratio must be greater than 1 but not more than 2, as shown in Table 2. This is due to the understanding that in ring blasting, the correct drilling pattern is one that delivers the appropriate energy at the toes of the blastholes.

Rock strength and related mechanical properties were determined through laboratory testing conducted at the University of the Witwatersrand's Gold Fields Laboratory. Tests included uniaxial compressive strength (UCS), uniaxial tensile strength (UTS), density, primary and secondary wave velocities, Young's Modulus, and strain measurements, using core samples obtained from the mine. Additional geological and structural rock mass parameters such as joint plane orientation, joint spacing, and rock mass rating (RMR) were sourced from Joughin et al. (2006) and SRK Consulting (2006). Where necessary, standard empirical values from Onederra (2005) were used to supplement the dataset. The Holmberg-Persson attenuation constants (K and α) are incorporated into the software to predict the extent of blasthole fracturing. This prediction is based on the understanding that rock damage (or breakage) is related to the peak particle velocity (PPV) generated by blast-induced seismic waves (Onederra, 2005). These parameters are presented in Table 3.

Table 1

Blast design parameters used at the mine

Parameter	Value
Blasthole diameter	76 mm
Explosive	1.12 g/cm^3
Explosive density	1.12 g/cm^3
Burden	2 m
Toe spacing	2 m – 2.4 m
Intra-row timing	30 ms
Inter-row timing	10 ms

Table 2

Generally accepted S/B ratio for calculating the toe spacing in ring blasting (Onederra, Chitombo, 2007)

Spacing to burden ratio (S/B)	Authors
1.1 – 1.4 (for blasthole diameter 76 – 115 mm)	Julius Kruttschnitt Mineral Research Centre (1996,1999)
1.5 – 2.0	Rustan (1990)
1.4 – 2.0	Myers et al. (1990)
1.3 – 1.5	Cunningham (2005)

The Aegis drill and blast analyser was calibrated using these site specific parameters. To reflect the influence of natural fracturing, the rock strength was conservatively set to “medium-high strength,” and the attenuation rate was assigned an “average” value. With these settings, the software was used to evaluate the performance of the mine's existing drill patterns. An analysis of the blast patterns using the Aegis drill and blast analyser was conducted using the blast parameters in Table 1 and the rock parameters in Table 3 as input for the software. The blast patterns supplied by the mine for the research showed that the S/B ratio varied from 1 to 1.2. Crack extension analyses were also conducted for the 2 m × 2 m and 2.4 m × 2 m patterns, as presented in Figures 5(a) and 5(b), respectively. As both patterns produced similar break overlap results, the findings were consolidated and summarised in Table 4.

The crack extension analysis shows that there is no break radius and pattern overlap for all toe spacing and burden of blastholes in Figures 5(a) and 5(b). Further, the blast patterns depicted in Figures 5(a) and 5(b) show identical crack extension results for the different zones in the stated UTS range of −30% to +30%. The analysis of UTS range considers the variation in the rock strength and properties across the orebody. For example, the pattern extent radius is expected to reduce to 0.83 m when the tensile strength is 30% stronger than the static UTS but increase to 1.14 m when the UTS is 30% weaker than predicted/measured. Similarly, other extent radii of the two patterns in Figures 5(a) and 5(b) show the values of the −30% to +30% range.

The mean pattern extent radius of 0.95 m, as shown in Table

Table 3

Summary of rock mass characteristics and properties at the mine

Rock property	Value	Rock property	Value	Rock property	Value
Uniaxial compressive strength (MPa)	263.34	Mean in situ block size (X_{insitu}) (m)	>1.00 (Massive) can be greater than 20 m^3	K	500.00
Uniaxial tensile strength (MPa)	13.72	Rock quality designation index (RQD) (%)	82.70	α	0.90
Rock density (kg/m^3)	2 686	Joint plane spacing (JPS)	0.87 (Rating = 20)	$PPV_{Breakage}$ (mm/s)	4000 – 4500
Young's Modulus (GPa)	77.78	Joint plane orientation (JPO)	Dipping into the face (Rating = 40)	Primary-wave velocity (m/s)	5900
Strain (μ)	0.16	H-index	7	Secondary-wave velocity (m/s)	3750
Rock mass rating (RMR)	Faulted zones = 76.5 Unfaulted zones = 92.3	Rock mass description (RMD)	Massive (Rating = 50)		

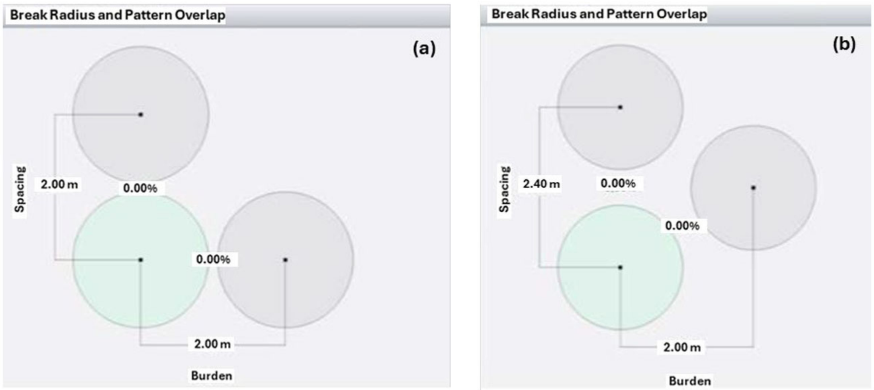


Figure 5—(a) Crack extension analysis for a 2 m toe spacing by 2 m burden patterns; (b) Crack extension analysis for a 2.4 m toe spacing by 2 m burden patterns

4, indicates that for both analysed patterns, the cracks generated by adjacent blastholes would result in minimal interaction. This marginal interaction suggests that there is insufficient overlap of the fracture zones to ensure complete breakage between blastholes. As a result, the rock between blastholes is likely to rely on natural failure mechanisms, which often lead to the formation of oversized boulders in the muckpile.

Radial break model

An analysis of both patterns using the radial break model is shown in Table 5. The Aegis software has pre-determined ranges within which the various parameters of a pattern, such as the burden and spacing break overlap, reflection point, break angle, etc., should fall to yield optimal results. The burden and spacing break overlap should fall between 2% and 12%, the reflection point between 23.42 MPa and 280.63 MPa for the rock properties' core samples analysed for this paper.

In Table 5, most parameters of the pattern fell within the required range except for the burden and spacing break overlap. Although this design may yield desired results, it is preferable to have all the parameters fall within the determined range for optimal results.

Stress reflection model

The second component of the Aegis analyser is the stress reflection model that illustrates the propagation and attenuation of the compressive shock wave through the rock mass, and the eventual reflection at the free face as a tensile wave. The Y-intercept line represents the blasthole wall and magnitude of the detonation stresses (Zone 1 of rock breakage), while the vertical faint purple line represents the free face of the blasthole. This model is highly dependent on the explosive and the rock attenuation properties. Figure 6 shows the stress reflection model for Figures 5(a) and 5(b).

As shown in Figure 6, the dynamic compressive shock wave generated during detonation reaches a peak magnitude of over 8000 MPa and attenuates as it propagates radially outward from the blasthole. By the time the compressive shock wave reaches the free face, its strength is reduced to around 153 MPa, however, it is over six times greater than the dynamic UTS of the rock. When the compressive shock wave reaches the free face, it is reflected back towards the blasthole as a tensile stress wave of significant magnitude. This reflection can induce multiple spalling events, leading to coarse fragmentation, a condition further exacerbated in zones where interaction between adjacent blastholes is minimal. According to Preston et al. (2019), to minimise spalling, and

Table 4
Calculated break values for the 2 m-by-2 m pattern and the 2.4 m-by-2 m pattern

	Calculated break values (unit charge length 1.82 m)					
Value	**Units**	**-30%**	**-15%**	**Mean**	**+15%**	**+30%**
Crush extent radius	m	0.19	0.17	0.16	0.15	0.14
Shatter extent radius	m	0.33	0.30	0.28	0.26	0.24
Minimum extent radius	m	0.51	0.45	0.43	0.40	0.37
Pattern extent radius	m	1.14	1.03	0.95	0.88	0.83
Maximum extent radius	m	1.76	1.60	1.47	1.37	1.29

Table 5
Radial break parameter analysis for the 2 m-by-2 m and 2.4 m-by-2 m patterns

Parameter	Range	Value	Parameter gauge	Results
Break overlap burden	2% – 12%	0.0%		✗
Break overlap spacing	2% – 12%	0.0%		✗
Reflection point	23.43 MPa – 280.63 MPa	157.40 MPa		✓
Ratio of radial break	45% – 75%	47.45%		✓
Break angle	80° – 90°	81.81°		✓
Powder factor mass	0.4 kg/tonne – 0.8 kg/tonne	0.59 kg/tonne		✓
Energy factor mass	1.52 MJ/tonne – 3.04 MJ/tonne	2.24 MJ/tonne		✓

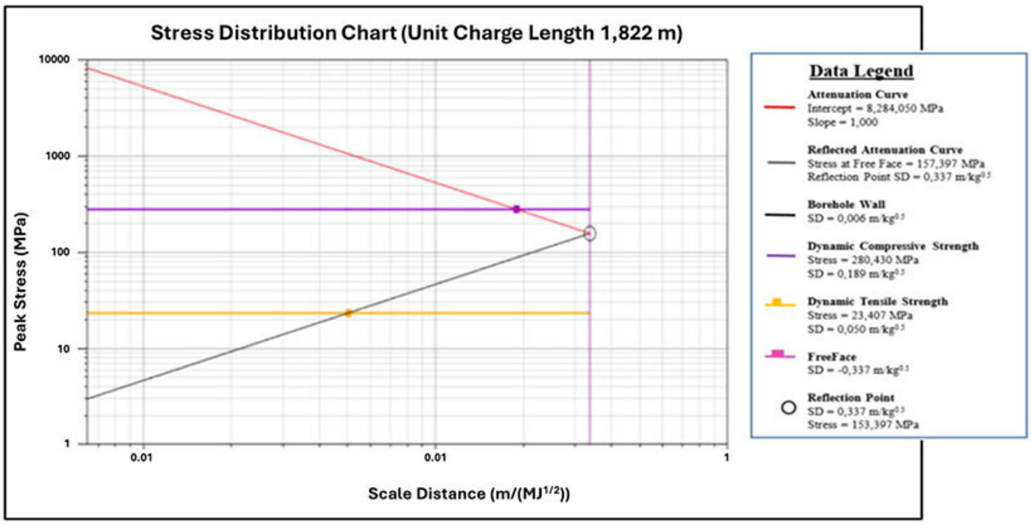


Figure 6—The stress reflection model for the 2 m-by-2 m and 2.4 m-by-2 m patterns

consequently achieve favourable fragmentation size distribution, the stress at the reflection point should ideally lie between the dynamic UCS and the UTS of the rock. As a result, the stress at the reflection point for this pattern is acceptable.

Simulation for optimal blast design

Multiple simulations of drill patterns were conducted using the Aegis analyser to identify optimal configurations that would yield improved fragmentation and enhance ore transportation. As the mine was trialling 89 mm blasthole diameters, simulations were carried out for both 76 mm and 89 mm holes. In addition to the blasthole diameter, the simulated variables included burden and spacing, as presented in Tables 6 and 7. Due to contractual obligations, the explosive type remained constant across all simulations. The timing configuration, comprising 10 ms intra-row and 30 ms inter-row delays, was also kept unchanged throughout the analysis. The overlap results for 76 mm blastholes achieved with various drill patterns are shown in Table 6.

The results in Table 6 were based on S/B ratios ranging from 1 to 2, in line with the recommended guidelines in Table 2. Among the 76 mm blasthole configurations, only the 1 m × 1 m pattern produced a burden and spacing overlap of 22.1%, suggesting strong crack interaction based on the radial break model. The analysis shown in Table 7 indicates that this overlap significantly exceeds the acceptable range of 2% to 12% prescribed by the Aegis software for optimal performance. An overlap of 22.1% would result in an excessively high powder factor and, consequently, a high energy concentration in the blast, increasing the likelihood of overbreak, excessive fines, and unnecessary energy wastage.

Furthermore, the extremely tight spacing in the 1 m × 1 m pattern introduces a high risk of blasthole convergence at the collars, which poses practical challenges during drilling and charging. Such a design would require highly accurate drilling practices and continuous supervision. Table 8 presents the overlap results for various drill patterns when an 89 mm blasthole diameter is used. The results indicate that overlap occurred primarily at an S/B ratio of 1. Among all the simulated configurations, only the 2 m × 2 m drill pattern achieved a burden and spacing overlap within the optimal range of 2% to 12% recommended by the Aegis software.

This pattern not only ensured sufficient crack interaction between adjacent blastholes but also maintained powder factor and energy factor values within the optimal range (see Table 9). This balance is critical in avoiding both underbreak and overbreak, while minimising the risk of generating excessive fines. Therefore, the 2 m × 2 m pattern is considered the most practical and effective design for achieving consistent fragmentation under the given site conditions.

At the time this study was conducted, the fragmentation model of the Aegis software had not been launched. However, there is high confidence that the fragmentation size distribution will be acceptable when the statistics are within the required range. Thus, a 2 m × 2 m pattern with a blasthole diameter of 89 mm is recommended. The mine currently uses 4G electronic delay detonators with 10 ms intra-row delay timing and 30 ms inter-row

Table 6

Pattern simulations to determine overlap with the 76 mm diameter blasthole

Φ (mm)	B (m)	S (m)	S/B	Overlap (%)
76	1	1	1	22.1
76	2	2	1	0
76	2	2.4	1.2	0
76	2	2.8	1.4	0
76	2	3.2	1.6	0
76	2	3.6	1.8	0
76	3	4	2.0	0
76	3	3	1	0

Table 8

Pattern simulations to determine overlap with 89 mm diameter blastholes

Φ (mm)	B (m)	S (m)	S/B	Overlap (%)
89	2	2	1	2.26
89	2	2.4	1.2	0
89	2	2.8	1.4	0
89	2	3.2	1.6	0
89	2	3.6	1.8	0
89	2	4	2.0	0

Table 7

Radial break parameter analysis for the 1 m-by-1 m patterns with 76 mm blasthole diameter

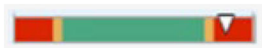
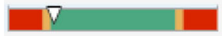
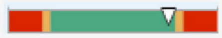
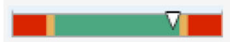
Parameters	Range	Value	Parameter gauge	Results
Break overlap burden	2% – 12%	22.1%		✗
Break overlap spacing	2% – 12%	22.1%		✗
Reflection point	23.43 MPa – 280.63 MPa	314,79 MPa		✗
Ratio of radial break	45% – 75%	94.89%		✗
Break angle	80° – 90°	85.57° – 90°		✓
Powder factor mass	0.4 kg/tonne – 0.8 kg/tonne	2.08 kg/tonne		✗
Energy factor mass	1.52 MJ/tonne – 3.04 MJ/tonne	7.91 MJ/tonne		✗

Table 9

Radial break parameter analysis for the 2 m-by-2 m patterns with 89 blasthole diameter

Statistics	Range	Value	Parameter gauge	Results
Break overlap burden	2% – 12%	2.3%		✓
Break overlap spacing	2% – 12%	2.3%		✓
Reflection point	23.43 MPa – 280.63 MPa	187.62 MPa		✓
Ratio of radial break	45% – 75%	56.31%		✓
Break angle	80° – 90°	82.38°		✓
Powder factor mass	0.4 kg/tonne – 0.8 kg/tonne	0.77 kg/tonne		✓
Energy factor mass	1.52 MJ/tonne – 3.04 MJ/tonne	3.00 MJ/tonne		✓

delay timing. Given the strength of the rock and the proposed drill pattern, it is further recommended that the mine maintains the timing plan as they are within the limits as determined by Langerfors and Kihlstrom (1963), and Grant (1990).

Discussion

The objective of this study was to diagnose the underlying causes of poor fragmentation in a deep-level underground gold mine and to recommend an improved drill and blast design that simultaneously mitigates oversized boulders and excessive fines. The discussion is organised around four themes: (i) the impact of fragmentation on the ore-flow circuit, (ii) diagnostic insights from the size-distribution analysis, (iii) mechanistic interpretation of the Aegis modelling results, and (iv) operational implications of the recommended 89 mm diameter blasthole with a 2 m × 2 m pattern.

At the time this study was undertaken, there were blockages in the underground ore transportation in the stopes where oversized material required secondary blasting, and at the ore-pass tips where large boulders had to be broken using mechanical hammers. The Split-Desktop image analysis yielded an F80 of 287 mm, acceptable for a 300 mm grizzly, yet the maximum fragment size reached 1288 mm, with the Rosin-Rammler uniformity index of above 0.8. Notably, 20% of the material measured less than 24.64 mm, highlighting the prevalence of excessive fines. This combination of very fine and extremely coarse fragments reflects a poorly controlled blast outcome. It is important, however, to note that fine material will always be present in the fragmentation size distribution due to the formation of the crushed zone and the severely fractured zone, as illustrated in Figure 3. The objective is to minimise these zones while ensuring sufficient energy transfer between blastholes to promote effective crack interaction.

The baseline blast design, using a blasthole diameter of 76 mm, employed a burden of 2 m and a toe spacing ranging between 2 and 2.4 m (S/B = 1–1.2). AEGIS crack-extension analysis indicated negligible crack interaction, with a mean pattern-extent radius of only 0.95 m from each blasthole, leaving zones of minimal energy that could result in coarse fragmentation. Meanwhile, energy concentration at the collars produced excessive fines. Thus, minimal toe interaction explains the observed phenomenon of the creation of fines and boulders within the muckpile. As a result, relying solely on F80 as an indicator of fragmentation quality concealed the operational consequences of these outliers. The simultaneous occurrence of fine ROM reported by the processing plant and oversized problems underground are two symptoms of the same underlying issue: inconsistent and suboptimal fragmentation distribution.

To overcome this problem, an exercise to optimise the blast pattern design was undertaken, with the explosives type being a constant. The 1 m × 1 m simulation of various blast patterns (combinations burden and spacing) using 76 mm blastholes achieved 22.1 % blasthole overlap at the toe, but all other combinations fell outside the recommended Aegis ranges, potentially resulting in the risk of an inflated powder factor and overbreak due to high blasting energy, potential collar convergence, and impractical drilling tolerances.

With respect to the 89 mm blasthole diameter, a 2 m × 2 m drill pattern satisfied every Aegis criterion simultaneously (all metrics fall in the green "acceptable" bands of the radial-break model), notably with the following technical aspects:

➤ A 2.26 % burden and spacing, which fell within the 2% – 12 % optimum range recommended by Aegis, would ensure controlled crack coalescence while avoiding over-break.

➤ The powder and energy factors, which fell within recommended Aegis limits, would prevent both under-break (coarse) and over-break (fines).

➤ The break angle and the stress at the reflection point, which fell within recommended Aegis limits (lies between dynamic UTS and UCS), would minimise spalling at the free face minimising coarse fragmentation.

Crucially, the larger blasthole diameter shortens the relative crack path, a benefit in the mine's high UCS rock mass. The tighter 2 m spacing (S/B = 1) is at the lower limit of JKMRC guidance but necessary to guarantee toe interaction.

Switching from 76 mm to 89 mm blastholes presents both practical and economic implications that must be carefully considered. From a practical standpoint, the proposed 89 mm, 2 m × 2 m drill pattern offers significantly improved fragmentation control, effectively reducing both oversized boulders that require secondary blasting and excessive fines that affect recovery. Improved fragmentation is expected to enhance underground ore transportation, resulting in increased mine throughput. Additionally, the larger diameter blasthole will reduce the total number of blastholes drilled per stope, thereby increasing blasthole drilling productivity and potentially improving drilling accuracy, and consequently, fragmentation size distribution. Economically, while 89 mm consumables may be more expensive than 76 mm consumables, the overall cost may be offset by savings from reduced secondary breakage, fewer ore-pass blockages, improved drilling productivity, and gains in plant throughput and energy efficiency. The drills currently used at the mine are compatible with the 89 mm blasthole diameter drill steels, therefore the additional investment is minimal. A phased implementation, supported by a cost-benefit analysis and pilot trials in selected stopes is recommended to manage risks and ensure a successful transition.

Conclusions

The optimisation of drill-and-blast designs is critical to achieving consistent fragmentation, reducing ore-handling bottlenecks, and enhancing overall mine efficiency, particularly in deep-level underground gold mines in South Africa, where cost pressures and low grades demand high throughput. This study uncovered that poor fragmentation, characterised by both coarse fragmentation and excessive fines, could lead to operational inefficiencies. At the mine in which this was conducted, a Split-Desktop image analysis of underground blasted muckpiles confirmed a low uniformity index of 0.8, indicating a wide-ranging fragmentation distribution, while Aegis simulations revealed a lack of energy overlap between blastholes, contributing to the significant presence of fines and large boulders in the muckpiles. The simulation of various drill patterns demonstrated that an 89 mm blasthole diameter with a 2 m × 2 m spacing and burden configuration meets all fragmentation criteria, including controlled crack interaction, optimal energy distribution, and reduced risk of both under- and over-break. This recommended design offers a practical solution to improving fragmentation, minimising secondary breakage, and supporting efficient underground ore transportation, which could ultimately increase mine throughput.

Limitations and future work

Future research should explore the impact of varying timing layouts, including electronic detonation schemes, to further optimise energy distribution and minimise undesirable fragmentation extremes. Additionally, newer Aegis versions incorporating in situ fragmentation modules should be used to validate predicted size distributions against actual muckpile sampling, further improving the predictive accuracy of blast performance models.

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CRediT author statement

MJN: Conceptualisation, methodology, investigation, formal analysis, writing – original draft and editing; EU: Supervision, writing – review and editing; PJKL: Writing – review and editing (technical review).

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