Effects of fragmentation on production at Peak mine Midlands, Zimbabwe

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Abstract

Over the years, the cost of production at Peak Mine Midlands, Zimbabwe, has been high due to oversized rock fragments. This led to increased mining costs far above what was planned by the mine. Initial investigations showed that valuable production hours were lost in trying to deal with the oversized rock fragments at the draw points. This study was conducted at Peak Mine Midlands, Zimbabwe, on 1210 level. The methods used to assess the effects of fragmentation on production at the mine included visual assessment, analysis of the drill and blasting costs of the production data, 2D image analysis, sieve analysis, time and motion studies, analysis of the explosive's consumption, mining blasting trials and post-mine blast experiments. The results show that the current ring hole designs used in the sublevel open stopes at Peak Mine do not give optimal particle size distribution after the primary blasts as the sieve analyses show that 50% of the sizes of materials at the draw points are larger than 1,000 mm. While 4 secondary blasts were budgeted per shift to reduce the sizes of the boulders to sizes that will pass through the grizzly's sieves, 7 secondary blasts were done per shift and this led to about 75% increase in the cost of explosives consumed by the mine.

The results also show that 35.96 minutes of production time were lost on each secondary blast conducted. Four trial blasts conducted on different ring hole patterns show that ring #1 which has a burden of 1.2 m instead of 1.8 m in the other ring holes led to a significant reduction in drilling and blasting costs from USD \$33.79 in trial #4 (the control case) to \$28.80 in trial #1. The sieve analyses also show that the optimal fragment sizes from primary blasts are in the range of X = +200 mm and X = +300 mm. So, the designed ring hole patterns in the sublevel open stopes should produce particles with sizes in the range of ≥ 150 mm and ≤ +350 mm from the primary blasts.

Keywords: Rock fragmentation, Secondary blasting, Mining cost, Particle size distribution, Sublevel open stoping.

Introduction

Over the years, there has been high cost of production at Peak Mine Midlands, Zimbabwe, due to oversized rock fragments at the draw points. This led to increased mining costs far above what was planned by the mine. Initial investigations showed that valuable production hours were lost in trying to deal with the oversized rock fragments at the draw points. This study was carried out at Peak Mine Midlands, Zimbabwe on 1210 level. The methods used to assess the effects of fragmentation on production at the mine included visual assessment, analysis of the drill and blasting costs of the production data, 2D image analysis, sieve analysis, time and motion studies, analysis of the explosive's consumption, mining blasting trials and post-mine blast experiments. Peak Mine Midlands, Zimbabwe, is located about 8 km south-east of the small town of Shurugwi in the Midlands province of Zimbabwe (Figure 1). Its Zimasco South dyke head offices are sited in the Shurugwi.

Peak mine is one of Zimbabwe's deepest underground mines and is currently operating at a depth of 1,260 m below the surface. It is a major producer of high-grade chrome which is a vital ingredient for Zimbabwe's steel making industry. The ore grade ranges from 40% to 55% Cr_2O_3 with a 2.0 to 3.3 chromium iron[10]. The run-of-mine ore from the mine is transported by rail to Kwekwe where it is smelted and the chromium metal is transported by rail to Beira, Mozambique where it is shipped to export markets. Chromite ore is sold to Zimasco at a price of US\$80/t to US\$120/t. The mine has a monthly production target of 4,000 tonnes which has not been achieved due to various problems with production.

Peak mine is located in the ultra-mafic complex on the left limb of the "Wolfshall Syncline" with both contacts north-south striking and steeply dipping about 70° to the west. The "Selukwe Greenstone" belt which hosts the ultramafic formation is bounded to the north by the Surprise Fault and the Great dyke to the south. The Selukwe greenstones are thrust faulted, folded and deeply eroded. The chromite bodies are hosted in the talc carbonates and silicified talc carbonates.

Generally, the chromite bodies have elongated pod forms which plunge steeply to the northwest. The main chromite bodies currently being extracted are the 21 and 4A bodies[11]. The host rock of the 21 body is predominantly talc carbonate while that of the 4A body is hosted predominantly in silicified talc carbonate.

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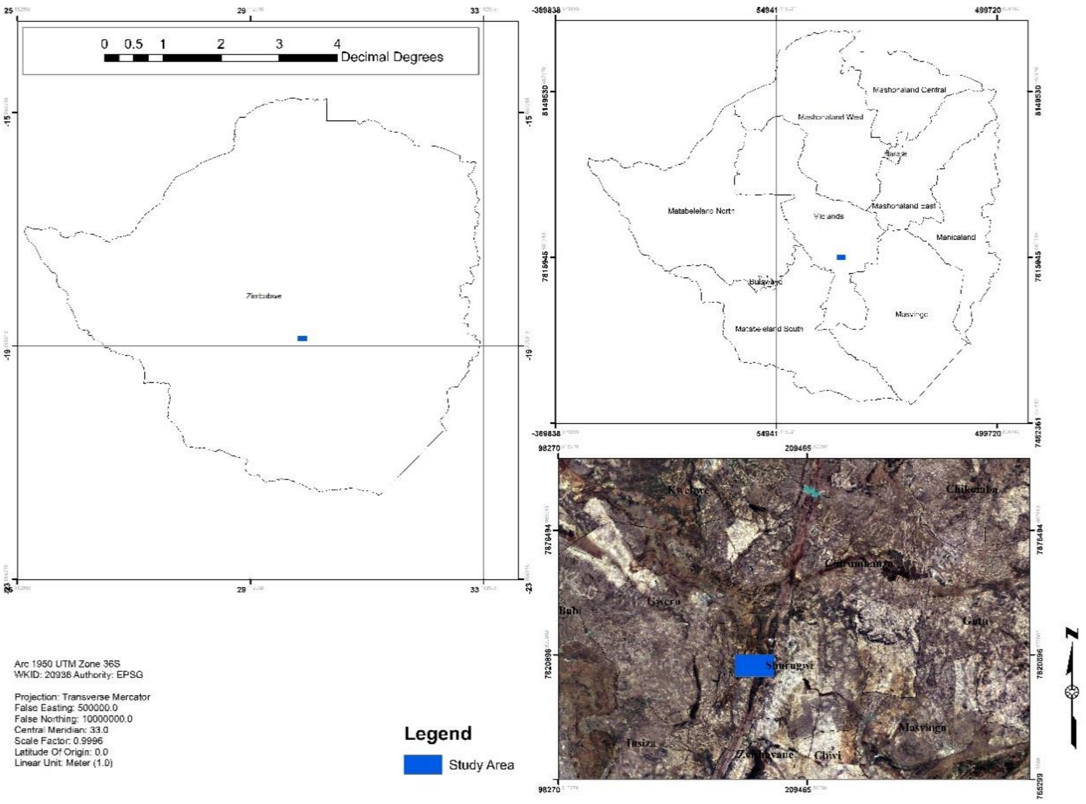


Figure 1: Location of the Study Area

Table 1
Properties of Rock Types at Mine[15]

Rock Type	UCS			No. of Test Results	No. of Zero Results	Mine Standard (UCS)
	Min	Max	Average			
Chromite	6.2	83.5	49.1	22	2	75.0
Talc carbonate	11.7	78.1	36.7	8	12	52
Silicified talc carbonate	54.5	256.1	105.7	19	0	191
Metadolerite	6.1	136.1	52.7	20	0	78
Argillite	8.9	82.2	58.9	10	0	49
Quartz porphyry	37.9	11.3	78.7	9	0	90

Rock tests

Six rock types, classified as chromite, talc carbonate, silicified talc carbonate, meta-dolerite, argillite and quartz porphyry, were obtained from the mine and uniaxial compressive strength (UCS) tests were conducted on them. The results are summarised in table 1.

The talc carbonate is generally classified as a weak rock as its strength ranges from 12 MPa to 78 MPa. The presence of jointing weakens the strength of the rock mass resulting in poor exaction stability. The silicified talc carbonate (STC) is considered as a competent rock as its strength ranges from 55 MPa to 256 MPa. The STC is generally stable and stability problems are only encountered where jointing creates blocky conditions. The exploration results show that the type of orebodies found at Peak Mine are regular and consist of both competent waste and ore rock. The orebodies are vertical - steep with uniform thicknesses and grades. Hence sublevel open stoping was considered as the best mining method due to its high productivity and low-cost of mining ranging from US$35/t to US$55/t. Shrinkage stoping method was mainly used in the upper levels of the mine where there was the need to extract the small pod-like deposits. Shrinkage stoping could be used as an alternative in mining the lower blocks of ore since it is more selective than sublevel open stoping method.

Good rock fragmentation is required for all the downstream mining activities in mines from materials handling to comminution processes as the cost of each activity depends on the degree of rock fragmentation achieved[2,14]. By definition rock fragmentation is an index used to monitor the efficiency of rock blasting[9]. Underground blasting operations account for about 60% of particle size reduction at Peak Mine while processing operations such as crushing, grinding and milling account for the remaining 40%[13]. The total cost of production of ore depends on the cost incurred during the primary and secondary breakage of the ore[5]. So special attention is needed to monitor the fragmentation from the primary blast using post blast analysis.

Mining of podiform chromite orebodies using longhole mining methods is a high production method but it is very sensitive to the price of chromite on the mineral market as a fall in market price has negative impact to the company's cash flow[6]. Unfortunately, peak mine has been experiencing high costs of mining due to poor primary breakage of its ore and it requires secondary blasting before the ore can be cost-effectively handled in the downstream mining and processing activities. According to Kanchibotla[7], an optimum design can only be achieved where the best fragmentation is obtained at minimum overall cost of mining. The cost of secondary blasting can be reduced significantly if due diligence is taken at the primary blasting stage to achieve optimum fragmentation.

Material and Methods

Mining method used at mine: The main mining method used at peak mine is sublevel open stoping. Figure 2 shows the fan, ring and parallel hole drilling patterns used in the sublevel open stopes. Every stope starts as a slot raise developed usually at the widest part of the orebody and in some cases, re-slots are also developed to ensure good recovery of the broken ore. The slot raises act as the free face for the initial blasts. Pillars of ore are left between adjacent levels and successive blocks.

Longhole drilling is done using pneumatic CH 123 drilling machines and the drilling is carried out from horizontal main levels which are spaced at vertical intervals of 60 m and three sublevels at vertical intervals of 15 m. After blasting, the material rills down to the draw points at the lower main levels or sublevels. Overhead loaders are then used to load the material into the 1.5-tonne wagons. The loaded ore is then hauled in the wagons by battery powered locomotives to the grizzly.

Methods used: This research was conducted to identify the factors that affect production at peak mine, to find the shortcomings in the current drilling patterns and the mining practices that will result in good fragmentation during primary blasting at the mine. Accordingly, the methods included visual assessment, analysis of the drill and blasting costs of the production data, 2D image analysis, sieve analysis, time and motion studies, analysis of the explosive's consumption, mining blasting trials and post-mine blast experiments.

Time and motion studies were conducted to determine how long it takes a loader with 0.26 m^3 dipper bucket to fill a 1.5-tonne wagon with ore at the draw points. It was also to assess how the size distribution of the rock fragments affects the loading time of the wagons. The fragments were classified into size ranges and the number of scoops taken to fill the wagons were recorded along with the time used in loading each wagon.

A post-blast photo analysis of the slot's crosscut was done using WipFrag.3 software. Images of the blasted ore were taken using a high-resolution camera and analysed to determine the particle size distribution. The assessment was conducted at strategic positions by viewing the stope from the top of the 1210 level grizzly. The percentage of the mass of blasted rock retained per load on top of the grizzly was noted. Graphs of the mass passing through the grizzly as a percentage the total load were plotted. A benchmark of 80% was set to assess the particle size distribution of the blasted material.

The main factors which affect the degree of fragmentation of a blast are the type of rock, burden to hole diameter ratio, spacing to burden ratio, stemming column length, stiffness ratio, explosives amount used per hole and type, initiation mode and powder factor[1,3,4,8]. The blasting parameters used in this investigation to determine their effects on the degree of fragmentation achieved included different burdens, different types of explosives (pentolite boosters) and the initiation mode (adjusting the timing sequence in various trial blasts) scenarios. Scenario 4 was considered as the control scenario in the trial blasts. The results of the trial blasts in ring numbers 1 to 4 are summarised under 4 scenarios in table 2.

Table 2
Mining Trials

Ring No.	Trial	Burden	Pentolite Booster	Timing	Control	Comments
R1	Scenario 1	✓	✕	✕	✕	Ring #1 has a burden of 1.2 m instead of 1.4 m while Ring Nos. 2 to 4 have burden of 1.8 m
R2	Scenario 2	✕	✓	✕	✕	Only Ring #2 was primed with pentolite booster + megamite primer
R3	Scenario 3	✕	✕	✓	✕	Different timing sequences used in Ring #3
R4	Scenario 4	✕	✕	✕	✓	Ring #4 was the control, which was used for comparison purposes.

Legend: ✓ - different conditions present ✕ - different conditions absent

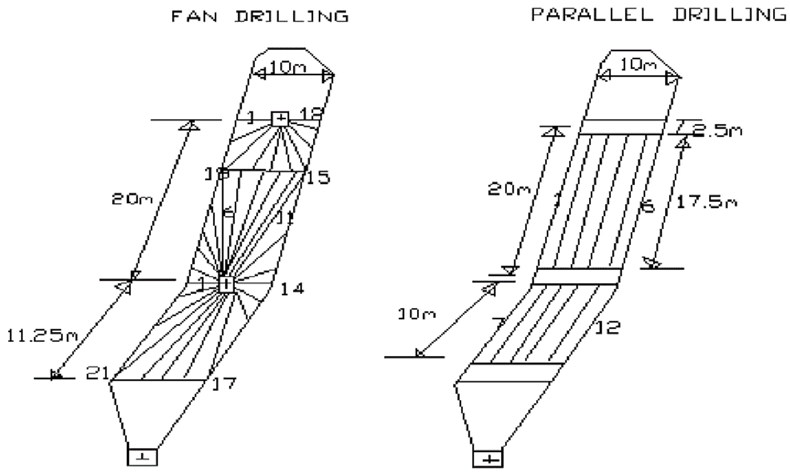


Figure 2: Drilling Patterns in Sublevel Stoping[16]

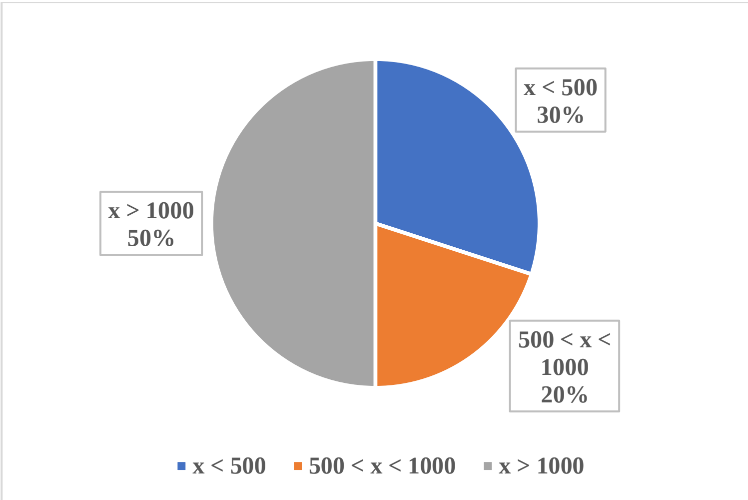


Figure 3: Distribution of Fragments at Draw Points at Peak Mine Using Sieve Analysis

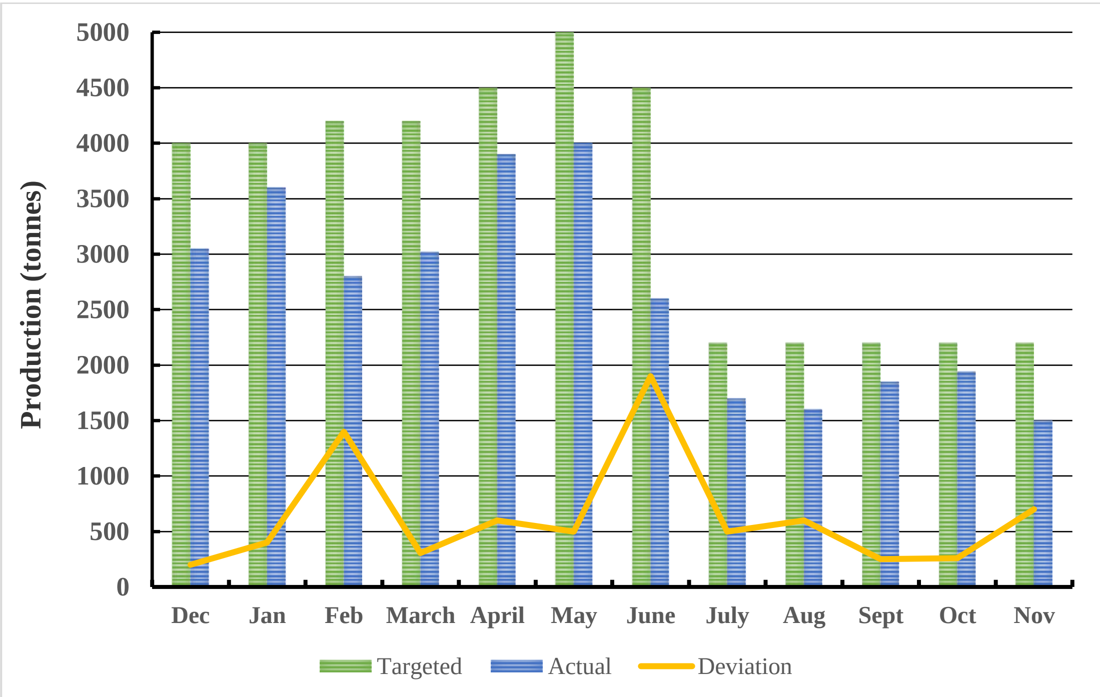


Figure 4: Summary of Production from 2017 to 2018[17]

Results and Discussion

Analysis of Distribution of Fragments at Draw Points: Figure 3 shows the distribution of fragments at the draw points at peak mine using sieve analysis. It shows that 50% of the particle sizes at the draw points are larger than 1,000 mm. The gape of the primary crusher at peak mine is 150 mm. The numerous boulders choke the draw points underground and the primary crusher at the surface. To increase the efficiency of all downstream mining processes (i.e. loading, hauling, hoisting, crushing, grinding and milling), an optimum fragmentation of the ore is required. So, secondary blasting is frequently required at the draw points to reduce the size of the boulders to manageable sizes for handling by the materials handling equipment underground. The numerous secondary blasting activities lead to the loss of valuable production time during the shifts as well as increased cost of explosives.

Analysis of Production Data: Figure 4 shows the production data from 2017 to 2018. The records show that the mine's actual production fell from 6,000 tonnes/month in the previous years to between 1,500 and 4,000 tonnes/month against targeted production ranging from 2,200 to 5,000 tonnes/month. It is noted that the mine never achieved its targeted production during the period of study. The time lost due to secondary blasting during production was probably the main reason behind the low production achieved.

Analysis of Drill and Blast Costs: The poor fragmentation from the primary blasts was mainly attributed to the current ring designs. Analysis of the direct and indirect costs of drilling and blasting was done to determine better blast pattern designs that could give better fragmentation during blasting. Various options of improving the rock fragmentation included varying the burden to hole diameter ratio, spacing to burden ratio, stemming column length, using more powerful explosives or initiation methods and varying the powder factor.

However, these options could lead to high explosives consumption with the resultant increased production costs. Figure 5 shows how the drilling and blasting costs vary with operator, fuel, maintenance, explosives and consumables when the number of holes drilled are increased. It can be seen that maintenance, explosives and operator costs affect the overall drilling and blasting cost more than the other parameters. This means that if the number of holes in the ring hole patterns are increased, it will result in longer operator hours, longer drill holes requiring more maintenance and higher consumption of explosives.

Figure 6 shows the current drilling and blasting costs in terms of operator, fuel, maintenance, explosives and consumables using the costs that are associated with the modified parameters in trials 1 and 4 in table 2. It shows that explosives cost is higher in the trial 4 than trial 1 while the costs are virtually the same with the rest of the parameters in Trials 1 and 4. Thus, ring #1 which has a burden of 1.2 m instead of 1.8 m in ring numbers 2 to 4 is preferred as it leads to lower explosives consumption.

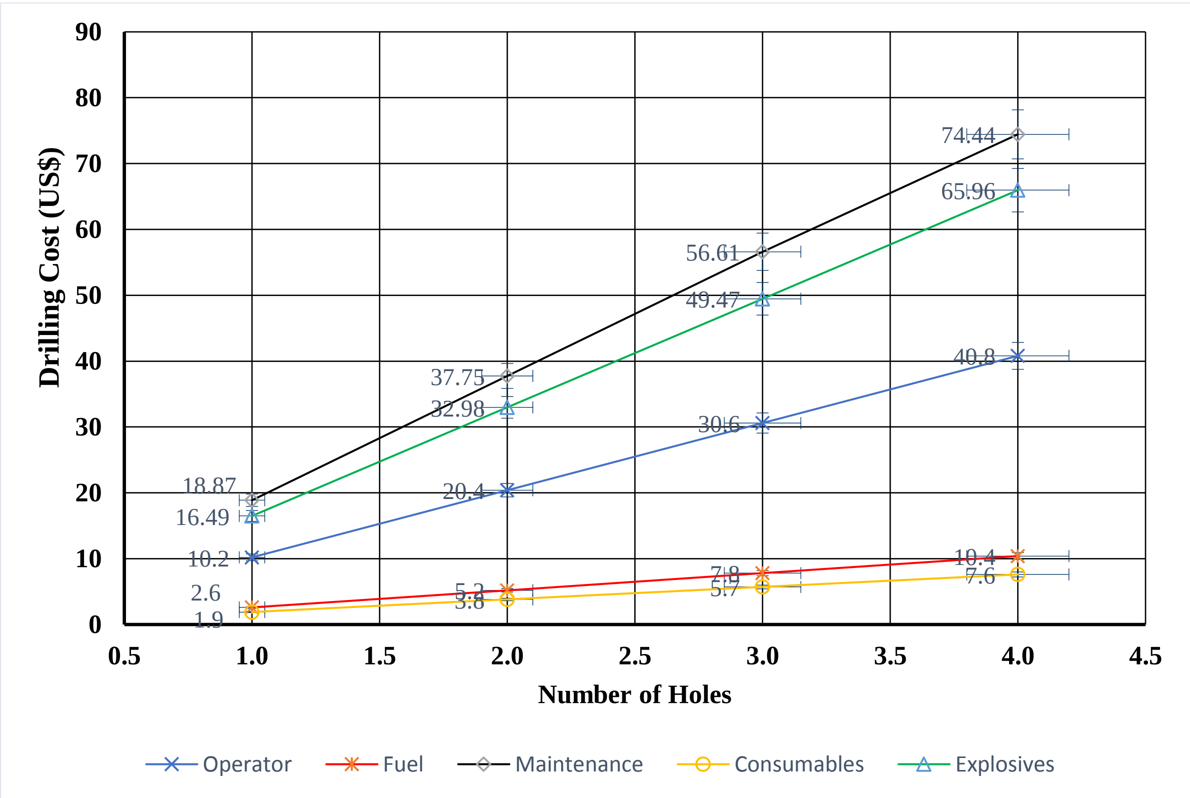


Figure 5: Cost of Drilling Cost per 20 m Hole

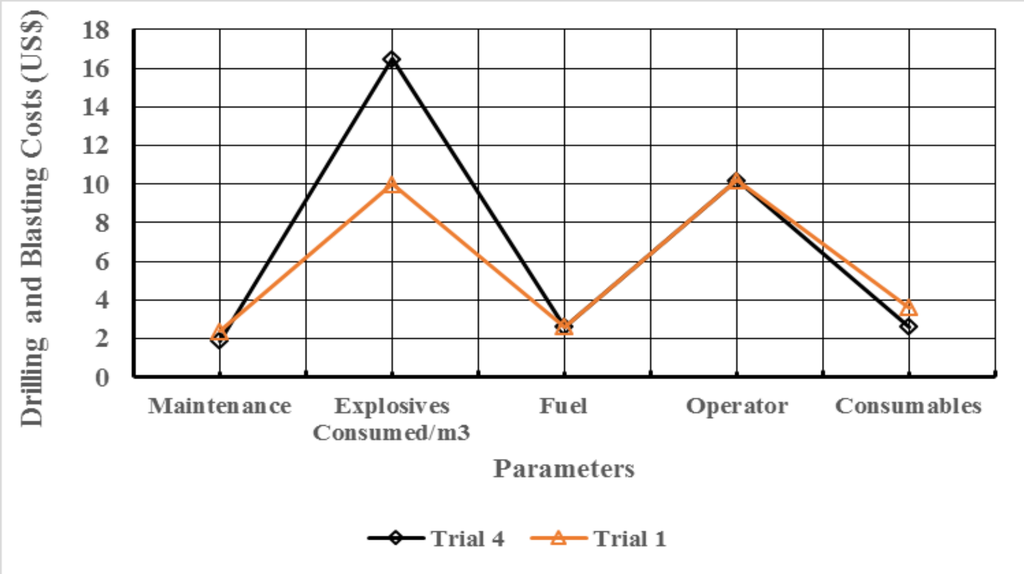


Figure 6: Drilling and Blasting Costs

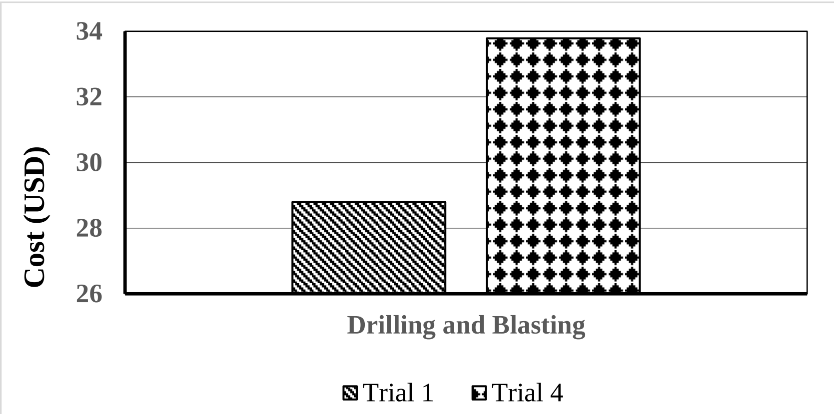


Figure 7: Total Drilling and Blasting Costs

Figure 7 shows the values of drilling and blasting costs obtained in trials 1 and 4. The results show a significant reduction in drilling and blasting costs from USD $33.79 in trial #4 (the control case) to $28.80 in trial #1.

Analysis of Time and Motion Studies: Time and motion studies were done to determine the time taken and the number of scoops required to load a 1.5-tonne wagon. Three readings were taken for each parameter and the average of the three readings was calculated. Figure 8 shows the particle size distribution versus the number of scoops and the time it takes to fill the 1.5 tonne wagons.

Figure 8 shows that the larger is the fragment sizes, the fewer are the number loader scoops required to fill the 1.5 tonne wagons. However, the time taken to load the wagons increases drastically when the rocks are poorly fragmented. For example, it takes an average of 120.50 s to load the 1.5-tonne wagons with X = 1,500 mm rock sizes and it takes about 41.20 s to load < 50 mm fragments (fines). However, the main objective in blasting is to reduce both the amounts of fines and number of boulders in the muck pile. So, it is necessary to find the degree of fragmentation that will give the optimum loading results (i.e. shorter loading times and higher fill factors). The optimal degree of fragmentation occurs around the point of intersection between the linear graphs for the number of scoops and time to load the wagons.

From figure 8, it can be concluded that the optimal fragment sizes are in the range of X = +200 mm and X = +300 mm. So, the designed ring hole patterns in the sublevel open stopes should produce particles with sizes in the range of ≥ 150 mm and ≤ +350 mm from the primary blasts. As much as possible, boulders with sizes ≥ +1,500 should be avoided as they choke the draw points and also take a very long time to load into the wagons.

Analysis of 2D Image Results: A post-blast photo analysis of the slot crosscut was done using WipFrag 3 software. These images are usually analysed to determine the particle size distribution from the blasts[12]. In this work, the data was plotted into histograms and cumulative graphs. The values for the mean fragmentation passing at D80, D50, D20 and the Sphericity value (SPH) values were generated automatically. The sphericity value (SPH) of 0.7 indicates that the average fragment width is 1/7 of its length. This is mainly because of the uneven distribution of energy during blasting. The images were processed and plotted in figure 9 from the first set of results while figure 10 is a plot of the particle size distribution from the blasts in the second set of results.

Figure 9 shows that the particle sizes passing at 80% were about 346 mm in size while in figure 10, about 80% of the particles that passed through the sieve, were about 1,077 mm. This means that the blast in figure 10 (second set) was poorer than that in figure 9 (i.e. the first set).

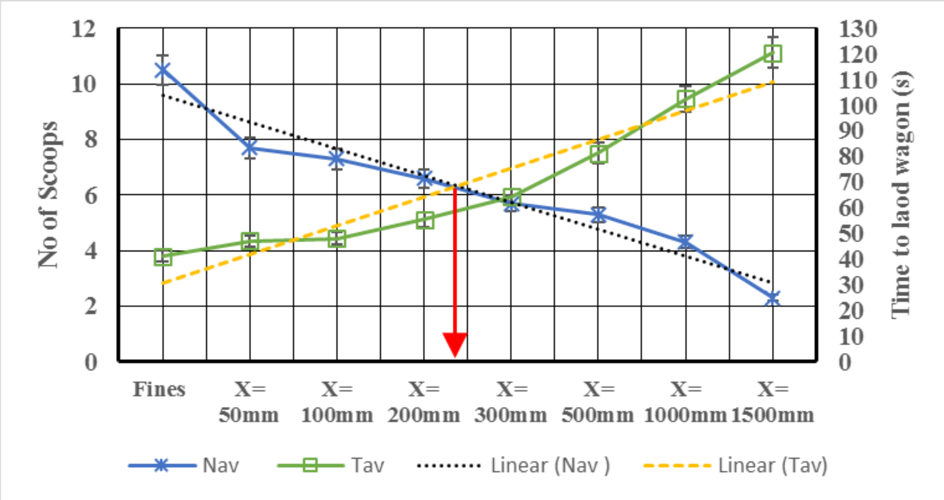


Figure 8: Variation of Time Scoops with Respect to Fragment Particle Size

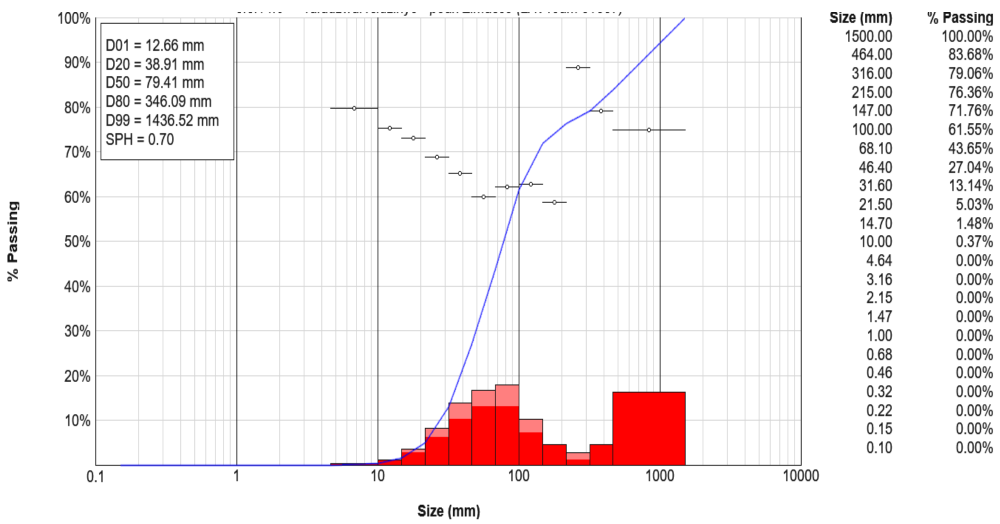


Figure 9: Log-Linear Results of Particle Size Distribution 1

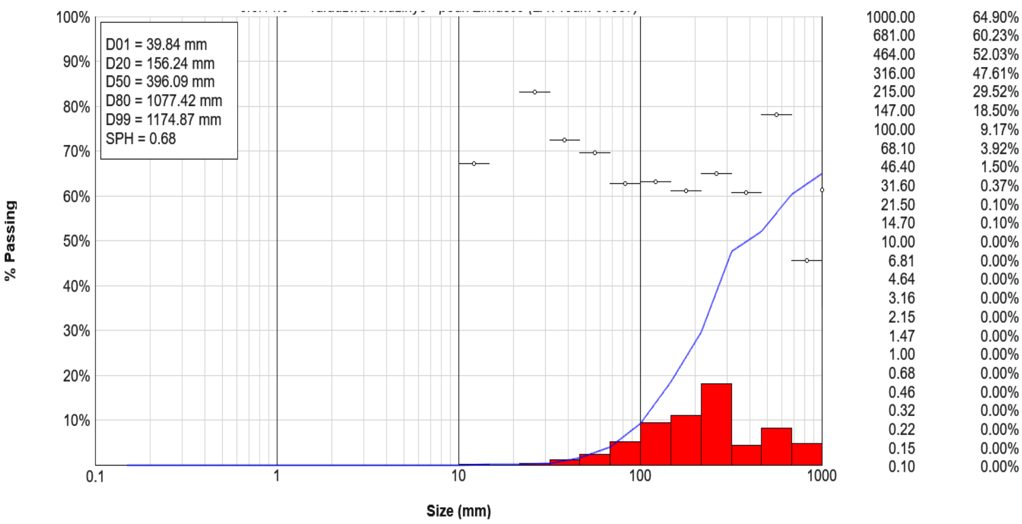


Figure 10: Log-Linear Results of Particle Size Distribution 2

Explosives Consumption: Data was collected on the number of 45 mm Megamite explosives that were used per blast in every shift of 8 hours. The mine had budgeted for a maximum of 4 secondary blasts to be done per shift. Figure 11 shows the number of secondary blasts per shift recorded during the period of the study.

Figure 11 shows that 4 secondary blasts were budgeted per shift, the actual number of secondary blasts per shift varied from 4 to 8 with a mode of 7 blasts per shift. Thus, there was about 75% increase in the cost of explosives at the mine.

Results of Lost Time due to Secondary Blasting: Time and motion studies were conducted to assess the time losses involved when the secondary blasts were done during the shifts. The final times were taken as the average of 6 readings. Figure 12 shows the composition of time lost during secondary blasting.

The results in figure 12 show that the total time lost during each secondary blast was 35.96 minutes (0.60 hr). About 49% (17.64 minutes) of the total time was lost due to the secondary blasting to allow for safe entry (re-entry period) for workers after each blast. Clearing the section before the blasts, charging of the holes and making the blasted areas safe constituted 14%, 23% and 14% of the total time lost respectively. Thus, the overall time lost during the 4 to 8 secondary blasts per shift ranged from 2.40 hr to 5.8 hr. This was clearly one of the main reasons why the mine was not able to meet its monthly production targets.

Sieve Analysis: In the sieve analysis, a grizzly with 800 mm spaces was used to analyse the size distribution of the fragments. Data on the amount of the fragments retained on the grizzly was collected at random from the 3 selected shifts and the results are as illustrated in figure 13. Series 1 of figure 13 shows the percentage of primary blasted ore passing through the grizzly while series 2 shows ore that was reblasted (secondary blasting) and 10 loads were picked for the analysis.

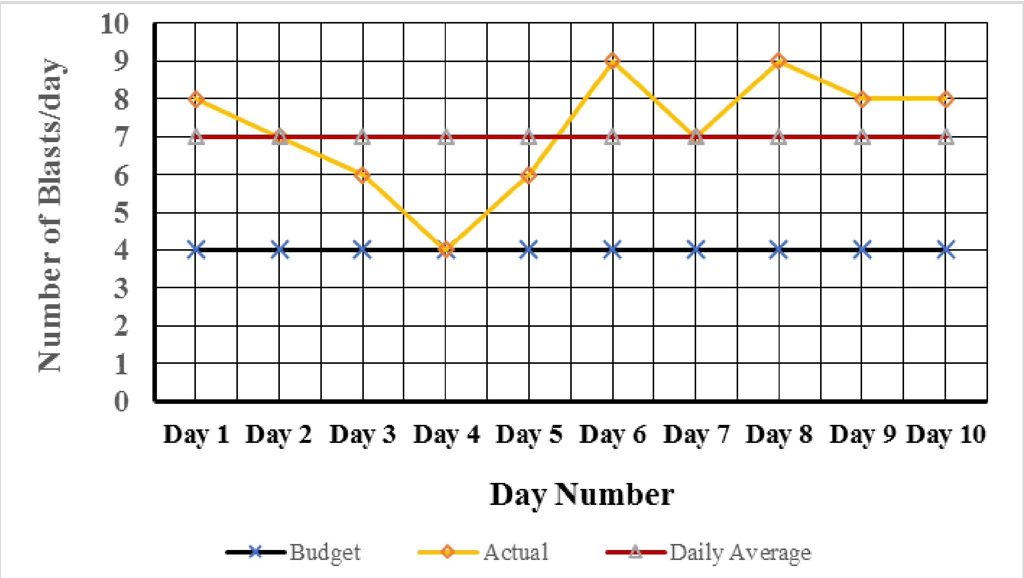


Figure 11: Number of Secondary Blasts per Shift

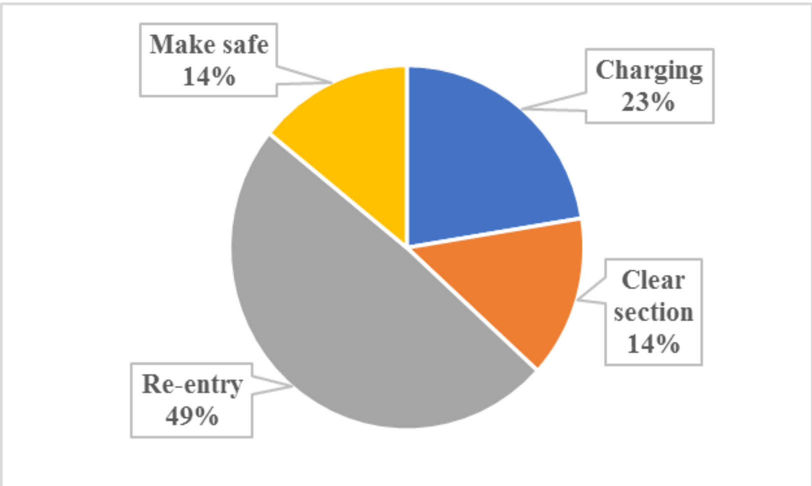


Figure 12: Total Time Loss during Secondary Blasting

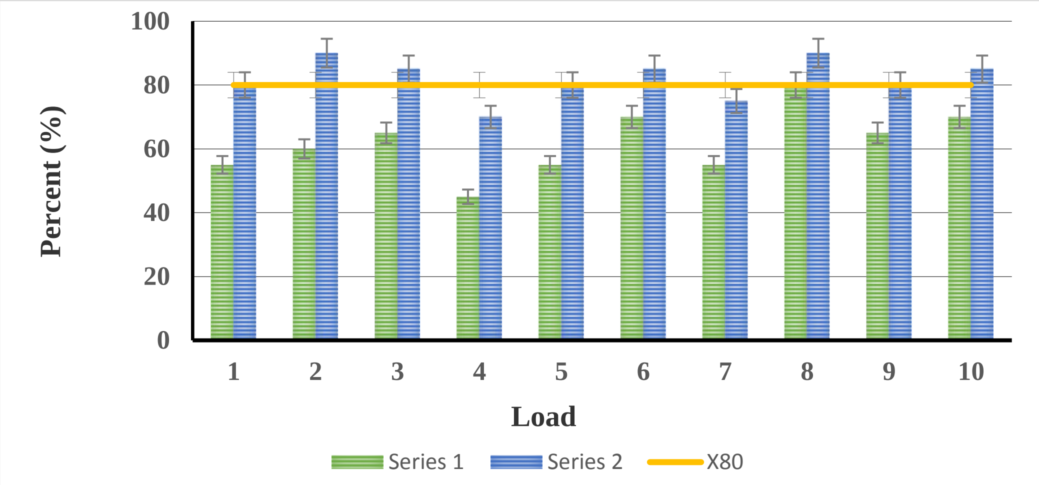
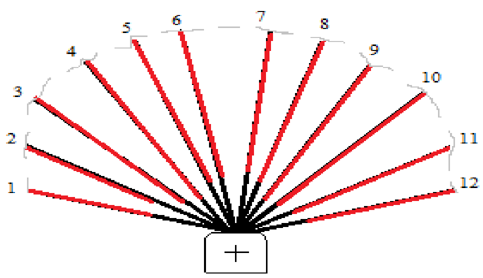


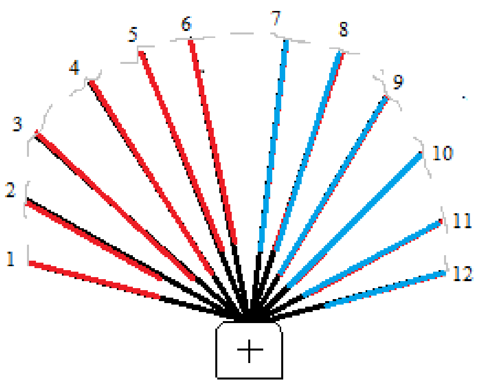
Figure 13: Sieve Analysis of Particle Size Distribution at Grizzly

Table 3
Ring 1 Design Parameters

Ring 1	Hole No.	Angle	Length (m)
	1	+16° W	9
	2	+32° W	11
	3	+52° W	15
	4	+66° W	18
	5	+78° W	18
	6	+86° W	20
	7	+86° E	20
	8	+78° E	19
	9	+66° E	17
	10	+52° E	17
	11	+32° E	16
	12	+16° E	16

Trial 1	Borehole Diameter	Explosive	Primer	Toe Burden	Ring Burden
	57 mm	ANFEX	45 mm Megamite	2.4 m	1.2 m

Table 4
Ring 3 Design Parameters

Ring 3	Hole No.	Angle	Length (m)	Delay (ms)
	1	+16° W	9	$T + T_1$
	2	+32°` W	11	$T + T_2$
	3	+52° W	15	$T + T_3$
	4	+66° W	18	$T + T_4$
	5	+78° W	18	$T + T_5$
	6	+86° W	20	$T + T_6$
	7	+86° E	20	$T + T_6$
	8	+78° E	18	$T + T_6$
	9	+66° E	17	$T + T_6$
	10	+52° E	15	$T + T_6$
	11	+32° E	12	$T + T_6$
	12	+16° E	12	$T + T_6$

Trial 3	Borehole diameter	Explosive	Primer	Toe Burden	Ring Burden
	57 mm	ANFEX	45 mm Megamite	2.4 m	1.8 m

The results in figure 13 show that the quality of the rock fragmentation at the draw points was very poor and is just about 10% of the loads of the samples meeting the benchmark level X80. However, after secondary blasting (Series 2), over 70% of the materials passed the benchmark level of X80. This shows that the current ring hole designs used in the sublevel open stopes at Peak Mine do not give optimal particle size distribution after the primary blasts and require to be reblasted before they can be efficiently handled by the materials handling equipment underground.

Blasting Trials: Four blasting trials were conducted with new ring hole designs in the sublevel open stopes. The details of the ring hole patterns are summarised in tables 3 and 4 while figure 14 shows the particle size distribution of the blasted materials from trials 1 to 4.

From figure 14, the degree of fragmentation from trial #3 gave the minimum percentage of ore passing through the grizzly. This shows that there was poor fragmentation in trial #3 blasts. The degree of fragmentation from trial #1 resulted in over 80% benchmark values passing through the grizzly's sieves. There was also an improvement of trial numbers 1 to 3 over the current blasting practice (Trial #4). However, it is also noted that more fines were produced in trial #1 accounting for the high values of the ore passing though the grizzly. There were many boulders in trial #3 resulting in less material going through the grizzly to the draw points.

Results of Experiments on Explosives Energy Distribution: A post-blast experiment to determine the explosives energy distribution was conducted and the results were recorded. Figure 15 is a 3D model of the results. The built model was then analysed to determine how the energy was distributed. For each set of X meters recorded, 3 readings were recorded each for 90°, 60° E and 60° W respectively. Trigonometric methods were used to calculate the values of the y-axis from the 600 readings that were obtained.

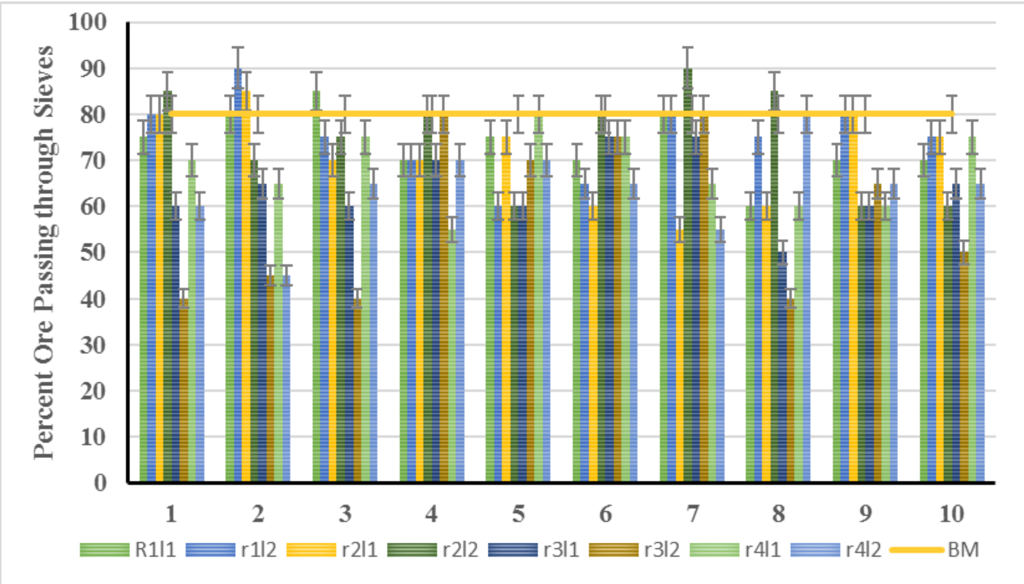


Figure 14: Particle Size Distribution of Four Mining Trials

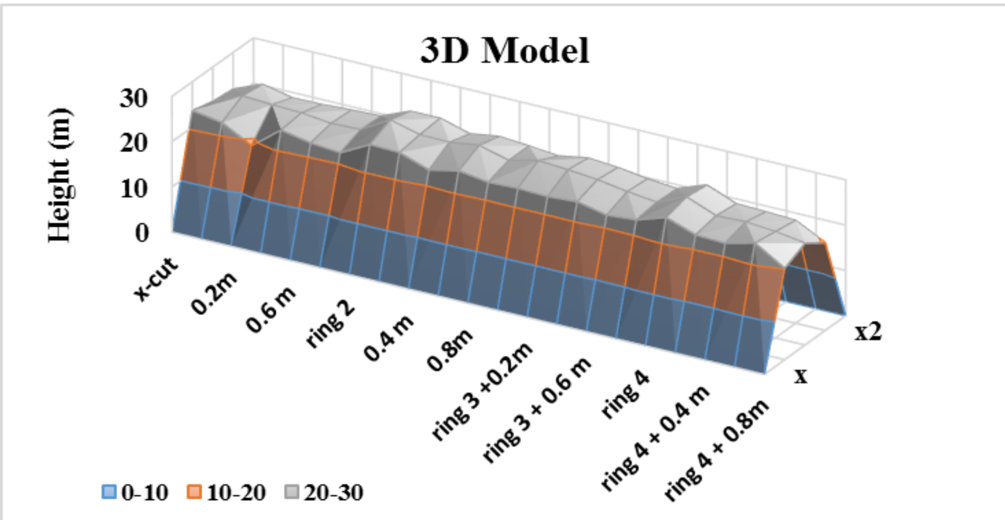


Figure 15: Post Blast 3D Model

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From figure 15, the blast energy is mainly concentrated in the regions close to the charge/blast rings and that the energy decreases gradually as the distance increases from the charge. Four blast conditions were set and each was analysed. Higher Distometer readings were recorded in the area between rings 2 and 3 showing that ring 2 generated more energy which resulted in over break of the hanging wall as depicted by the higher grey areas around ring 2 on the model.

Conclusion

From the analysis in this study, it is concluded that peak mine has not been able to achieve its production targets over the years due to the poor distribution of fragments from the primary blast as the sieve analyses show that 50% of the sizes of materials at the draw points are larger than 1,000 mm. The current ring hole designs used in the sublevel open stopes at the mine do not give optimal particle size distribution after the primary blasts. As a result, secondary blasting is required to reduce the sizes of the boulders to manageable sizes that will pass through the grizzly's sieves. While 4 blasts were budgeted per shift, the actual number of secondary blasts per shift varied from 4 to 8 (with a mode of 7 blasts per shift) and this led to about 75% increase in the cost of explosives consumed at the mine.

The numerous secondary blasting activities conducted during the shifts led to a total time loss of 35.96 minutes (0.60 hr) per blast; trial blasts conducted on different ring hole patterns show that ring #1 which has a burden of 1.2 m instead of 1.8 m in ring numbers 2 to 4 led to significant reduction in drilling and blasting costs from USD $33.79 in Trial #4 (the control case) to $28.80 in trial #1. The sieve analysis also shows that the optimal fragment sizes are in the range of X = +200 mm and X = +300 mm. So, the designed ring hole patterns in the sublevel open stopes should produce rock fragments with sizes ranging from ≥ 150 mm to ≤ +350 mm from the primary blasts. As much as possible, boulders with sizes ≥ +1,500 should be avoided as they choke the draw points and also take a very long time to load into the wagons.

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