

A Comparative Analysis of Rock Fragmentation using Blast Prediction Results*

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Abstract

This work presents prediction and optimisation of controllable parameters of drilling and blasting currently used at the Fobinso Pit of Perseus Mining Ghana Limited (PMGL). The mine faces challenges with blast particle sizes produced after primary blasting. The presence of boulders requires secondary fragmentation to further reduce the broken materials to the acceptable sizes, thereby increasing the cost of production. The mechanical properties of the rocks were determined using Protodyakonov Rock Strength Index. The drill and blast parameters were estimated using the Konya and Walter (1990), Principle of Proportionality, and *Instituto Geologo Minero de España* (IGME) model developed in 1987. The Modified Kuz-Ram model was used to predict and analyse the results of blasting based on the parameters such as spacing, burden, drillhole diameter, charge density, charge per hole, charge length, and powder factor. A comparative analysis of the predicted size distributions of the various models using diameters of 65 mm and 110 mm revealed no significant differences between the size fractions. The Principle of Proportionality produced the best technical and economic indices for blasting.

Keywords: Fragmentation, Drilling Parameters, Primary Blasting, Models, Secondary Blasting

1 Introduction

Perseus Mining Ghana Limited (PMGL) is a surface gold mine located at Ayamfuri in the Central Region of Ghana. The mine is currently operating four pits: Fobinso, Fetish, Esuajah North and Chirewewa. The mine has been facing challenges with fragmentation sizes where boulders are frequently produced after the primary blasting. The presence of lump sizes above 300 mm then requires secondary fragmentation to further reduce the broken materials to the required size where 80% of the fragmented materials fall below 300 mm before they are delivered to the processing plant for crushing. Delays and inefficiencies in the blasting activity greatly affect downstream operations such as loading, hauling and crushing. There is therefore the need to review the controllable factors to determine the optimal drill and blast parameters.

The following are some of the vital parameters which determine blast design (Mishra, 2009): pit geometry, explosive characteristics, burden distance, spacing between drillhole, ratio of spacing to burden, hole depth, diameter of drillholes, consideration of toe and depth of sub-grade drilling, blasting technique, powder factor, length of stemming column, and quality of stemming.

Issues generally discussed under explosive characteristics are type of explosive, type of booster, bulk strength, energy release per unit mass of explosive, detonation pressure, explosion pressure, ratio of decoupling, strength of explosive, time taken for explosive wave to travel to the free

face and back, volume of gases produced per unit mass of the explosive, velocity of detonation and velocity of explosion propagation. The burden is considered the most important and critical value in the design of a surface blast. There are two types of burden; drilled burden and short burden (Mensah, 2012). The drilled burden can be defined as the distance between the rows of holes and the nearest free face measured perpendicular to the row of holes. It is also the distance between two rows of holes. The short burden is defined as the distance between the holes that are detonating and the nearest free face developed during the blast. According to Bhandari (1997), with multiple row blast, the burden may not necessarily be given as the distance to the nearest free face because as the drillholes with the lowest delays detonate, they too create new free faces. As a result, the effective burden will depend on the selection of the delay pattern. Burden values below the optimum would result in an excessive forward throw, high air blast levels, and excessively fine fragmentation. Burden above optimum values would probably yield inadequate fragmentation with possible upward throw causing fly rocks. Excessive burden also causes over confinement of the drillholes, which results in high levels of ground vibrations per mass of explosives used (Hagan, 1983; Konya and Walter, 1990).

Spacing is the distance between adjacent drillholes, measured perpendicular to the burden which controls the mutual effect between holes (Bhandari, 1997). It is calculated as a function of the burden, drillhole depth, and relative primer location between adjacent charges; and also depends upon initiation time interval. According to Frimpong-

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Mensah (2016), smaller spacing distance causes crushing and cratering between holes, large blocks in the burden and toe problems. On the other hand, spacing above the optimum dimensions would result in loss of interaction between detonating charges, while spacing below the optimum values could cause partial cancellations of explosive forces and contribute to excessive vibration. When drillholes are too close, cracks from the closely spaced drillholes will link prematurely causing a shattered zone in the wall between holes. The premature linking of the radial cracks will form a fracture plane between holes through which gases can escape prematurely to the atmosphere causing air blast, flying rocks, and toes. If the drillholes are too far apart, fragmentation will be coarse, and toes, non-uniform and rough walls will result.

Spacing is a function of burden. For most situations, the spacing to burden ratio is between 1 and 2. In general, a ratio of 1:4 is used as an ideal geometric balance for breakage of massive material (Hagan, 1983; Mishra, 2009).

According to Bhandari (1997), the drillhole depth depends on the type of rock to be blasted. In a rock material of low strength which is readily loosened, shallow drillholes and smaller explosive charges are generally used. It is important that the burden and the drillhole be reasonably compatible. As a rule of thumb, for bench blasting the drillhole depth to burden ratio should be between 1.4 and 4.0. Hole depths less than 1.5 times the burden cause excessive air blast and fly rocks, coarse and uneven fragmentation (Hagan, 1983).

According to Konya and Walter (1990), sub-drilling is the depth to which drillhole will be drilled below the proposed grade to ensure that breakage will occur to the grade line. This is because drillholes usually do not break to full depth. The sub-drill depends on the burden, orientation of the drillhole relative to all open faces, and the geological structure conditions at its bottom (Hagan, 1983; Bhandari, 1997). If a shear plane or a parting exist at that location, sub-drilling may not be required. As a general rule, the sub-drill length should be equal to about one-third the burden distance for hard ground conditions (Bhandari, 1997). In multiple bench operations, excessive sub-drilling may cause undue fracturing and gouged pit floors, creating difficulties in collaring holes in the lower bench (Frimpong-Mensah, 2016). The required sub-drilling is also expressed regarding the blast drillhole diameter (Bhandari, 1997). If sub-drilling is required, it is important to limit it to only the minimum length required since excessive sub-drilling may cause floor damage in benching operations (Bhandari, 1997).

The drillhole diameter selected in combination with appropriate positioning of the holes, will give fragmentation suitable for loading, transportation and crushing (Bhandari, 1997). Additional factors to be considered in the determination of the drillhole diameter are the size of operation, the bench height, the type of explosive used and rock characteristics. Occasionally it may be necessary to use two different drillhole sizes within the same blast. According to Olofsson (1988), the borehole diameter dictates the charge diameter and controls the number of explosives in the borehole for any explosive type. It also affects the reaction velocity and thus the detonation pressure.

Powder factor is the ratio between the total weights of the explosives detonated in a blast divided by the amount of rock that is broken (Olofsson, 1988). It is an essential parameter in blast design and has a vital influence on the resultant fragmentation. According to Olofsson (1988), the powder factor is a function of the explosive type, rock density and the geology of the formation being blasted. Competent rock or formation with few or no pronounced planes of weakness require higher powder factor than a formation with numerous closely spaced structural weakness. Stemming is the filling of some part of the blasthole with inert materials such as debris from drilling sand, chipping, tailings from treatment plants and mills (Bhandari, 1997; Olofsson, 1988). Optimum stemming columns with coarse angular crushed rocks interlock themselves to confine high gas pressure in the blastholes for longer periods permitting the charges to perform to their maximum potential. Stemming below the optimum values can reduce fragmentation and loosening as energy is ejected prematurely to the atmosphere. Optimum stemming increases with a decrease in the effective strength of the rock. Columns below the optimum values cause airblast, cut-offs and overbreak problems (Hagan, 1983). According to Konya and Walter (1990), a stemming height of 0.7 times the burden is adequate to keep material from ejecting prematurely from the hole. Stemming height is proportional to the burden. The optimum size of stemming material would be material that has an average diameter of approximately 0.05 times the diameter of the drillhole.

Protodyakonov rock classification table (Table 1) using the sturdiness index, f values, based on which the rocks were divided into ten (10) categories is shown in Table 1. The Relative Rock Sturdiness Index (f), is suitable for an elaborate and a systemic study of standard machine productivity, and work efficiency for economic analysis of rock fragmentation. The f -index is proportional to the

limit of compressive strength (σ_c) of rock as expressed in Equation 1.

The first category refers to rocks with the highest compressive strength and tenth category refers to weak rocks. Protodyakonov used regular samples of 100kgf/cm³ in designing the table.

$$f = \frac{\sigma_c}{100} \quad (1)$$

However, field empirical tests have proven that calculation based on the Equation 2 gives a more accurate result.

$$f = \frac{\sigma_c}{300} + \sqrt{\frac{\sigma_c}{30}} \quad (2)$$

In reality, the UCS of some rocks are higher than 2000 kgf/cm³. However, precise results can be obtained when $f = 20$ is assumed for high strength rocks. Soft rocks with values below 200 kgf/cm³ should be adjusted to $f = 3$.

There is the need to determine the optimum drill and blast parameters to reduce the boulders. The objective of this paper is to do a comparative analysis of Konya and Walter (1990), IGME, and the Principle of Proportionality to predict the blast design parameters for the Fobinso pit of PGML to achieve mean fragment sizes that will help reduce boulders and produce uniform distribution after an initial blast.

Table 1 Protodyakonov's Classification of Rock Sturdiness and Powder Factor

Group	Sturdiness	Description of rock	f	Powder factor
I	Sturdiest	The hardest, toughest and most dense quartzites and basalts.	20	0.7-0.75
II	Very sturdy	Very sturdy granite rock, quartz porphyry, siliceous schist, weaker quartzites Sturdiest sandstone and limestone.	15	0.50-0.70
III	Sturdy	Granite (dense) and granite rock. Very sturdy sandstone and limestones. Quartz veins. Sturdy conglomerate, very sturdy iron ore.	10	0.45-0.60
IIIa	Sturdy	Limestones (sturdy), weaker granites Sturdy sandstones, marble, dolomites and pyrites.	8	
IV	Rather sturdy	Ordinary sandstones, iron ore.	6	0.45-0.60
IVa	Rather sturdy	Sandy schists, schistose sandstones.	5	
V	Moderate	Sturdy shale. Non-sturdy sandstones and limestones Soft conglomerates.	4	0.40-0.50
Va	Moderate	Various schists (non-sturdy). Dense marl.	3	
VI	Rather soft	Soft schists. Very soft limestones, chalk, rock salt, gypsum, Frozen soil, anthracite. Ordinary marl. Weathered schists, compressed shingle and gravel, rocky soil.	2	0.35-0.60
VIa	Rather soft	Detritus soil. Weathered schists, compressed shingle and detritus, sturdy bituminous coal, hardened clay.	1.5	
VII	Soft	Clay (dense), soft bituminous coal, sturdy alluvium, clayey soil.	1.0	0.35-0.45
VIIa	Soft	Soft sandy clay, loess gravel.	0.8	0.30-0.45
VIII	Earthy	Vegetable earth, peat, soft loam, damp soil.	0.6	0.35-0.40
IX	Dry loose rock	Sand, talus, soft gravel, piled-up earth, and substances extracted, coal.	0.5	
X	Flowing rock	Shifting sands, swampy soil, rare-fractioned loess, and other rare-fractioned soils.	0.3	

2 Resources and Methods Used

The following methods were adopted to fulfil the stated objective:

- i. Core specimens from many points were collected and carefully packed and sent to the Geotechnical Laboratory of PMGL for the analysis.
- ii. After the sample collection, they were prepared for laboratory testing. The collected samples were taken through analysis to know about the Uniaxial Compressive Strength (UCS) to identify the type of rock present using the Protodyakonov's classification of rock sturdiness.
- iii. The Determination of UCS was done with HZ-1009G 60T Computer Servo Compression Testing Machine with the following technical working conditions: Maximum Load Capacity: 1200kN; Platens Diameter: 80 mm; Platens Hardness \geq 58 HRC; Vertical Clearance: 167 mm; and Overall Dimensions: 249 diameter \times 333 mm heights and Weight: 30 kg.

The rock test was compliant with ASTM D 7012 for Rock Core Specimen Test Methods. Field visits for data collection on drill and blast parameters from the mine were conducted. For the calculation of drill parameters, several methods exist. However, for this work, Konya and Walter (1990), Principle of Proportionality, IGME and Fragmentation Size Prediction, using Modified Kuz-Ram, were selected because of their relative simplicity and effectiveness.

The blastability of rocks was determined using Protodyakonov Relative Rock Sturdiness Index, (f).

2.1 The Principle of Proportionality

The Principle of Proportionality as a blast design model depends essentially on the drillhole diameter and the rock strength index without taking into account other features like fillings, openings, spacing and orientation of discontinuities. For this reason, the Principle of Proportionality produces precise results if the correct drillhole diameter is selected.

The burden (B) was obtained using Equation 3 (Otaño 1998):

$$B = D \sqrt{\frac{8.2\Delta}{mq}} \quad (3)$$

where D is the diameter of drillhole (dcm), Δ is the charge density (dcm³), m is the coefficient of

approximation of charges, taken between 1.1 and 1.5, q is the powder factor, kg/m³.

Table 2 shows the mathematical expressions and the values of the rest of the controllable parameters by the principle of proportionality.

2.2 The Konya Method and Walter Method

The Konya and Walter (1990) Model parameters are burden, spacing, stemming and subdrill. The burden (B) was obtained using Equation 4 as follows:

$$B = 0.012 \times \left[\frac{2 \times \rho_e}{\rho_r} + 1.5 \right] \times D \quad (4)$$

where ρ_e is the explosive density (g/cm³), ρ_r is the rock density (g/cm³) and D is the drillhole diameter (mm).

The Konya and Walter Method uses bedding and other qualitative characteristics that may bring about unpredictable errors. The parameters using Konya and Walter (1990) model is shown in Table 3.

2.3 IGME Method

Like Konya and Walters (1990), the fundamental drill and blast parameters estimated with the IGME method are burden, spacing and subdrill. The method determines the controllable parameters on the basis of the UCS (MPa) of the rock and the proposed diameter of drillhole diameter (D) is expressed in Table 4.

2.4 Modified Kuz-Ram Model

The modified form of the Kuz-Ram model with some modification to the Kuznetsov (1973) equation and the uniformity index was used. The Rosin-Rammler function is maintained as in the original Kuz-Ram model. Cunningham (2005) adjusted the Kuz-Ram model. The major changes to the model however, were developed as a result of the introduction of electronic delay detonators (EDs), since these have transformed fragmentation. The new equation, Equation 5 set includes changes in the uniformity and mean fragment size equations as follows:

$$R_x = \exp \left[-0.693 \left(\frac{x}{x_m} \right)^n \right] \quad (5)$$

Table 2 Drill and Blast Parameters for Principle of Proportionality

Parameter	Symbol	Formula
Burden, m	B	Equation (3)
Spacing, m	S	mB
Hole Depth, m	H_d	$H + 0.3B$
Sub-drill, m	J	$0.3B$
Stemming, m	T	$20D$
Hole Diameter, mm	D	Deduced
Bench Height, m	H	Field Data
Charge Length, m	L_c	$(H + 0.3D) - 18D$
Charge Density, g/cc	Δ	1.2
Charge Per Hole, kg	Q	$0.7 \times q \times S \times B \times H$
Powder Factor, kg/m ³	q	deduced
Spacing to Burden Ratio	S_B	S / B
Stiffness Ratio	S_R	H / B
Bench width (m)	B_w	Field data
Number of rows	N_R	$\frac{B_w}{S} + 1$
Distance between rows (m)	D_R	$B(1 - 0.1H)$

Table 3 Drill and Blast Parameters for Konya and Walter (1990)

Designed Parameters	Symbol	Formula
Burden (m)	B	Formula
Spacing (m)	S	$S = [H + (7 \times B)] / 8$
Hole Depth (m)	H_d	$H_d = H + J$
Sub-drill (m)	J	$J = 0.3 \times B$
Stemming (m)	T	$T = 0.7 \times B$
Hole Diameter (mm)	D	Deduced
Bench Height (m)	H	Field data
Charge Length (m)	L_c	$L_c = H_d - T$
Charge Density (g/cm ³)	Δ	1.2
Charge Per Hole (kg)	Q	$Q = \pi \times D^2 / 4000 \times L_c \times \rho_e$
Powder Factor (kg/m ³)	q	$PF = \frac{Q}{B \times S \times H_d}$
Spacing Burden Ratio	S_B	$S_B = \frac{\text{Spacing}}{\text{Burden}}$
Bench Width (m)	B_w	Field data
Stiffness ratio	S_R	$S_R = \frac{\text{Bench height}}{\text{Burden}}$
Number of rows	N_R	$\frac{B_w}{S} + 1$
Distance Between rows	D_R	$B(1 - 0.1H)$

Table 4 Relationship between Drill and Blast Parameters and Diameter (IGME, 1987)

Parameter	Weak <70	Medium 70-120	Hard 120-180	Very Hard >180
Burden (mm)	39D	37D	35D	33D
Spacing (mm)	51D	47D	43D	38D
Stemming (mm)	35D	34D	32D	30D
Sub-drill (mm)	10D	11D	12D	12D

where R_x is the mass fraction retained on screen opening x , x_m is the mean size and n is the uniformity index.

The mean size, x_m is given in Equation 6 as:

$$x_m = AA_T K^{-0.8} Q^{1/6} \left(\frac{115}{RWS} \right)^{19/30} C(A) \quad (6)$$

where A is rock factor; K is powder factor (kg/m^3); Q is mass of explosive in hole (kg), RWS is weight strength relative to ANFO, 115 is RWS of TNT; $C(A)$ is rock correction factor; and A_T is timing factor.

The uniformity index (n) is also given in Equation 7 as:

$$n = n_s \times \sqrt{\left(2 - \frac{30B}{d} \right)} \times \sqrt{\frac{1+S/B}{2}} \times \left(1 - \frac{W}{B} \right) \times \left(\frac{L}{H} \right)^{0.3} \times C(n) \times \left(\frac{A}{6} \right)^{0.3} \times g \quad (7)$$

where B is burden (m), S is spacing (m), d is hole diameter (mm), W is standard deviation of drilling precision (m), L is charge length (m), H is bench height (m), n_s is timing scatter factor; $C(n)$ is

correction factor for known uniformity, A is rock hardness factor, g is geometry factor (Mireku-Gyimah and Sarpong, 2018) uses Modified Kuz-Ram Model to estimate and compare the fragmentation of two blast designs to determine the optimum sizes and cost.

3 Results and Discussion

The Principle of Proportionality and IGME are essentially based on the diameter of drill and the strength of the rocks without considering other important aspects like infill material, discontinuities, joint spacing, and joint orientation. However, they are considered to be more precise than Konya provided the right diameter is selected because they have limited variables in their calculation. The simplest of the three is IGME, model which is based solely on the diameter of drill. The existing drill and blast parameters of PMGL are found in Table 5. The spacing to burden and the stiffness ratios are between 1 and 2. There is a good geometric balance for PGML pit at Fobinso.

From Table 6, the fresh granite is relatively stronger than the fresh phyllite and fresh greywacke rocks. All the rock types can be classified as strong and require an efficient drill and blast design.

Table 5 Designed Drill and Blast Parameters from PMGL at Fobinso Pit

Designed Parameters	Symbol	Values
Burden (m)	B	3.1
Spacing (m)	S	3.5
Hole Depth (m)	H _d	6.0
Sub-drill (m)	J	1.0
Stemming (m)	T	2.6
Hole Diameter (mm)	D	115.0
Bench Height (m)	H	5.0
Charge Length (m)	L _c	3.4
Charge Density (g/c^3)	Δ	1.2
Charge Per Hole (kg)	Q	38.7
Powder Factor (kg/m^3)	q	0.59
Spacing to Burden Ratio	S _B	1.12
Stiffness Ratio	S _R	1.61

Table 6 Intact Rock Strength Tests from UCS Testing

Rock Types	Fresh Granite	Fresh Greywacke	Fresh Phyllite
No. of samples	25	12	15
Max (MPa)	330	232	200
Min (MPa)	86	48	46
Range (MPa)	243	185	154
Mean (MPa)	180	126	126
Std. Dev (MPa)	62.09	72.48	49.01

The mean UCS of rock samples taken from each point obtained using Equation 8 is as follows:

$$f = \frac{R}{300} + \sqrt{\frac{R}{30}} \quad (8)$$

where R is the uniaxial compressive strength (kgf/cm^3), f is the resistance factor ($.1 \text{ kgf/cm}^3 = 0.0981 \text{ MPa}$).

From Table 4, granite has the highest mean compressive strength at the Fobinso pit; and substituting this value, results in:

$$f = \frac{1834.86}{300} + \sqrt{\frac{1834.86}{30}}$$

$$f = 13.94$$

$$f \cong 14$$

With reference to Protodyakonov’s classification of rock sturdiness, $f = 14$ confirms that the rock is granite, it falls in category II, it is very sturdy and requires fragmentation by blasting.

3.1 Estimation of the Controllable Factors using Principle of Proportionality Parameters

The first step is to relate the Rock Sturdy Index with the productivity of the pit; because, the rocks are of high resistance with $f = 14$ and production is mainly dependent on the conditions of the rock properties. The expected productivity, according to Table 7, is $60 \text{ m}^3/\text{h}$ and the corresponding diameter is 65 mm for an open pit mine (Otaño, 2013). The hourly production of ore of the pit is 60 m^3 and the rock strength is high, the diameter can be estimated as 65 mm according to Table 7 for open pits. The current drill diameter of 115 mm for Fobinso Pit is higher than the proposed 65 mm .

The designed parameters using 65 mm for Konya and Walters, Principle of Proportionality and IGME are shown in Table 8 as KW-65, P-65 and IGME-65 respectively. Similar notations for 115 mm are shown in Table 9.

Table 7 Parameters for the Selection of Diameter of Drill

Average Hourly Productivity (m^3/hr)		Diameter (mm)
Soft to Medium Strength	Hard to Ultra-hard	
190	60	65
250	110	89
550	270	150

(Source: IGME, 1987)

However, when the diameter is reduced, the number of drillholes increases, causing the cost of drilling, priming and initiation increase. Stemming and connecting up operations are time consuming and labour intensive. Among all the three prediction models for 65 mm diameter, the principle of proportionality gave the least number of drill holes per row. However, IGME model gave the least charge per hole with the same powder factor as the Principle of Proportionality.

The determined blast parameters shown in Tables 8 and 9 were used to predict the fragmentation size distribution using the Modified Kuz-Ram model of 25 ms delay. Fig. 1 shows the predicted fragmentation size distribution for Konya and Walter (1990), Principle of Proportionality and IGME parameters using the Modified Kuz-Ram.

The comparison of the prediction models using 115 mm with PMGL actual parameters are shown in Table 9. Larger diameters above the optimum can lead to poor fragmentation in highly fractures rocks with long spaced discontinuities. However, an increase in diameter is accompanied by an increased in charge diameter. Higher diameters increase explosion energy factor and higher detonation velocity. A stable and high detonation velocity produces good fragmentation (Hagan, 1983). A comparative analysis of the drill parameters in Table 9 reveal that the Principle of Proportionality gave the least drillholes of six per row. This means that Principle of Proportionality can help reduce drilling cost as compared to the drilling parameters currently in use in PGML. However, the charge per hole of PGML of 38.70 kg is lower than the 45.55 kg for the Principle of Proportionality design.

Table 8 Comparison of Principle of Proportionality, Konya and Walter and PMGL Actual Parameters Using 65 mm

Designed Parameters	Symbol	KW-65	P-65 mm	IGME-65
Burden (m)	B	1.86	2.36	2.15
Spacing (m)	S	2.26	2.59	2.47
Hole Depth (m)	H _d	5.56	5.71	5.64
Sub-drill (m)	J	0.56	0.71	0.78
Stemming (m)	T	1.30	1.30	1.95
Bench Height (m)	H	5.0	5	5
Charge Length (m)	L _c	4.25	3.78	4.47
Charge Density (g/cm ³)	Δ	1.20	1.20	1.20
Charge Per Hole (kg)	Q	16.93	14.55	12.98
Powder Factor (kg/m ³)	q	0.72	0.68	0.68
Spacing to Burden Ratio	S _B	1.21	1.1	1.15
Stiffness Ratio	S _R	2.68	2.12	2.33
Bench width (m)	B _w	25	25	25
Number of rows	N _R	12	11	12
Distance b/n rows (m)	D _R	0.93	1.16	1.10

Table 9 Comparison of Principle of Proportionality, Konya and Walter and PMGL Actual Parameters Using 115 mm

Designed Parameters	Symbol	KW-115mm	P-115 mm	IGME-115	PGML
Burden (m)	B	3.30	4.17	3.80	3.1
Spacing (m)	S	3.51	4.59	4.37	3.5
Hole Depth (m)	H _d	5.99	6.25	6.14	6.0
Sub-drill (m)	J	0.99	1.25	1.38	1.0
Stemming (m)	T	2.31	2.07	3.45	2.6
Bench Height (m)	H	5.0	5.0	5	5
Charge Length (m)	L _c	3.68	4.18	4.07	3.4
Charge Density (g/cm ³)	Δ	1.20	1.20	1.20	1.20
Charge Per Hole (kg)	Q	45.86	45.55	3947	38.70
Powder Factor (kg/m ³)	q	0.66	0.68	0.68	0.59
Spacing to Burden Ratio	S _B	1.06	1.10	1.15	1.12
Stiffness Ratio	S _R	1.52	1.20	1.32	1.61
Bench width (m)	B _w	25	25	25	25
Number of rows	N _R	8.0	6	8.0	8
Distance b/n rows (m)	D _R	2.78	2.09	1.90	1.55

3.1 Fragmentation Prediction Using Modified Kuz-Ram Model

All the prediction models produced similar size distribution patterns with more than 90% of the predicted sizes above the optimum size of 300 mm. The predicted fragment size using IGME with drillhole diameter 65 mm had the least predicted percent oversize of about 0.2% with about 97.9% percent in range of 300 mm. An analysis of the predicted size distributions curve in Fig 1 using 115 mm also revealed that (IGME model) had the least percent oversize of about 1.6 % with about 96.6% within the optimum size range of 300 mm. This can help us to deduce that IGME produced the best predicted fragmentation results for 65 and 115

mm for all the predicted models. About 93.2% of the predicted sizes of PGML were within the range of 300 mm. This explains that the high percentage of boulders produced in the mine cannot be attributed to the current blast design but maybe due to poor operational practices by the drill and blast crew.

Ho: $\mu_1 = \mu_2 = \mu_3 = \mu_4 = \mu_5 = \mu_6 = \mu_7$ Null hypothesis (mean values have no significant difference)

H1: $\mu_1 \neq \mu_2 \neq \mu_3 \neq \mu_4 \neq \mu_5 \neq \mu_6 \neq \mu_7$ Alternate hypothesis (mean values have significant difference)

The null hypothesis is valid when all the sample means are equal and can be considered as a part of a larger set. On the other hand, the alternate hypothesis is valid when at least one of the sample means is different from the rest of the sample means.

From the results in Tables 10 and 11 and, it can be seen that the F-value (0.031664) is lesser than the

F-critical value (2.157914) for the level of significant selected (0.05). Also, the p-value obtained (0.999862) from the analysis is greater than the selected p-value (0.05). There is therefore not enough evidence to reject the null hypothesis and conclude that, the mean of the predicted values obtained using each prediction model has no significant difference for material size from 0.00 to 1100.

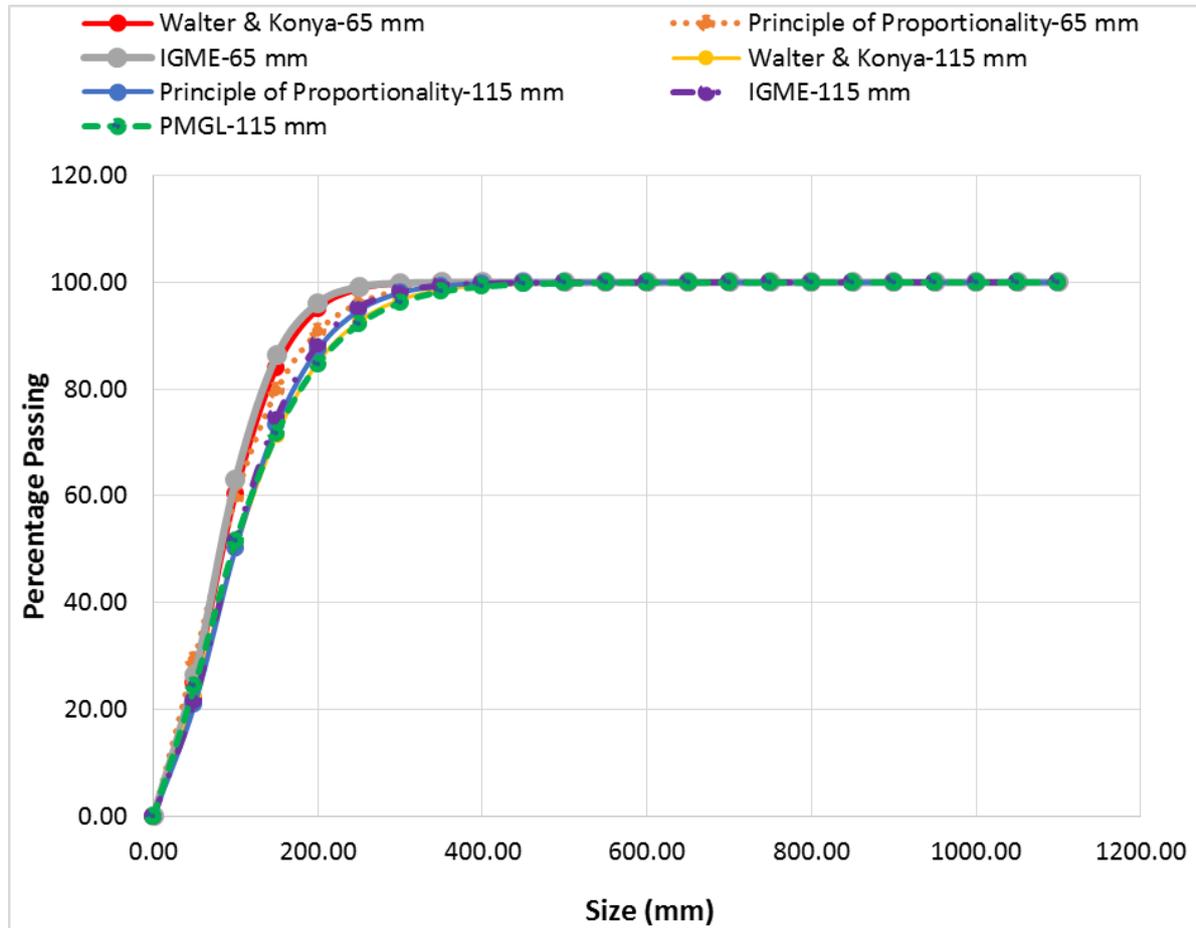


Fig.1 Predicted Size Distributions Curve

Table 10 Anova: Single Factor for 65 mm and 115mm

<i>Groups</i>	<i>Count</i>	<i>Sum</i>	<i>Average</i>	<i>Variance</i>
Konya & Walter-65 mm	23	2016.276	87.66417	731.5854
P.Prop.-65 mm	23	2023.639	87.9843	741.5903
IGME, -65 mm	23	2029.055	88.2198	732.1177
Konya & Walter-115 mm	23	2017.59	87.72129	714.5535
P.Prop.-115 mm	23	2062.393	89.66925	684.3943
IGME, 1987-115 mm	23	2053.324	89.27497	658.9134
PMGL	23	2070.881	90.03831	668.3651

Table 11 Source of Variation

<i>Source of Variation</i>	<i>SS</i>	<i>df</i>	<i>MS</i>	<i>F</i>	<i>P-value</i>	<i>F crit</i>
Between Groups	133.846	6	22.30767	0.031664	0.999862	2.157914
Within Groups	108493.4	154	704.5028			
Total	108627.3	160				

4 Conclusions

The mean of the predicted values obtained using each prediction model has no significant difference for material size from 0.00 mm to 1100 mm. The Principle of Proportionality model produced the least drilling cost. The prohibitive levels of boulders produced by the mine cannot be attributed to the existing blast design parameters. The boulders produced may be explained by inappropriate operational practices. All the predicted models produced fragment sizes of more than 93 % within the range of the optimum size of 300 mm for the ore.

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