Optimal Design Blast Parameters for an Effective Rock Fragmentation*

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Abstract

Achieving the required fragment sizes with the maximum size (p100 value) being less than 1000 mm after blasting is a major challenge in the Newmont Ahafo South mine., Ghana. Blasting usually results in an excessive proportion of boulders which negatively affects productivity by increasing the time taken for loading, hauling, and crushing. It also increases the cost of operation because of secondary blasting, and the excess fines also result in the loss of gold. The objective of the study is to review the current blast design procedures adopted by Ahafo South Mines in order to identify the causes of the boulders and to optimise the design parameters, if necessary, in order to obtain the required fragmentation for the mine. Quality assurance and quality control were done on the existing drilling and blasting procedures to identify the deficiencies, and optimised (modified) drilling and blasting parameters were obtained through a simulation using the Kuz-Ram model. Digital images from the blast shots were taken and analysed using the Orica Shotplus software, and the average result of the two blocks was compared to results from the Kuz-Ram model. Because of the relative ease of predictability of the Kuz-Ram model, it was used to predict the blast design parameters that would yield optimal fragmentation without any boulder. The image analysis showed an average variation of 16.4% of the expected fragmentation, which aided in obtaining an expected maximum size of 603.4 mm, other than 517 mm from the prediction model. Thus, the Kuz-Ram model was used to predict spacing and burden of 4.2 x 3.5 m (with a powder factor of 1 kg/m^3) as the optimal values for the blast design.

Keywords: Optimisation, Drilling, Blasting, Fragmentation, Kuz-Ram, Model

1 Introduction

The valuation of fragmentation remains an important discussion in the mining operation as it is the first step towards mineral recovery (Kazem and Bahareh, 2006). A variety of modelling approaches ranging from empirical to rigorous numerical models have been used to predict rock fragmentation with industrial explosives. Some of the most common are as follows: the Kuz-Ram model, the Julius Kruttschnitt Mineral Research Centre (JKMRC) Model, the Kuznetsov-Cunningham-Ouchterlon (KCO) Model, Larsson's equation and Swedish Detonic Research Foundation (SveDefo formula) (Adebola *et al.*, 2016). The KJMRC model is composed of the crushed zone model (CZM) and the two-component model. Both models are an extension of the Kuz Ram model and are used to overcome the underestimate of fines (Ouchterlonya and Sanchidrián, 2019)

The methodology of the use of Kuz-Ram and the discrepancies that exist between predicted results and the actual results generated by design for similarities and correlation are examined.

The Kuz-Ram model for the prediction of rock was first discussed in (Cumingham, 1983). Since then, there has been significant improvement in the model, and it probably has surpassed in performance by more complex fragmentation models. The Kuz-Ram's model is a simple model that gives a reasonable approximation of rock fragmentation results in a three-parameter fragment size distribution model.

Agyei and Owusu-Tweneboah (2019) estimated drill and blast parameters using the Konya and Walter (1990) Principle of Proportionality and *Instituto Geologo Minero de España* (IGME) methods. However, the comparative prediction of the fragmentation size distribution for diameters from 65 mm to 110 mm using the Modified Kuz-Ram model did not yield a statistically significant difference in the results for the three methods. Mireku-Gyimah and Boateng (2018) employed the Kuz-Ram model to predict the fragmentation sizes and feasible performance of two alternative blast designs, obtaining a hybrid optimised value for burden, spacing, bench height and powder factor for Kofi C Pit of Endeavour in Mali. The modern approach to optimisation is called 'Mine-to-Mill', provided by Julius Kruttschnitt Mineral Research Centre in 1998 (Beyglou, 2012). Mine-to-Mill, in short, is an approach that identifies the leverage that blast results have on different downstream processes and then optimises the blast design to achieve the outcomes that maximise the overall profitability rather than individual operations (Sharma, 2015). Mine-to-Mill concept goes further to explain that blasting should be designed in a way

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that satisfies the overall requirements of the comminution process and haulage.

In this paper, the Kuz-Ram model is used to predict and analyse the results of blasting to achieve the optimal design of blast parameters for an effective rock fragmentation for the Newmont Ahafo mine.

1.1 Orica Shotplus Size Distribution Analysis

Shotplus software by Orica is a professional software used for blast design, post-blast fragmentation analysis and analysis of comprehensive blast issues. Unlike Split Desktop Software which is user dependent and hence might not be reliable to some extent, Shotplus gives a practical prediction.

Ahafo South mine has, however, been encountering many challenges since its initiative to achieve zero boulders. Despite the efforts to improve the drilling and blasting practices at the pit, the results of blasting and degree of fragmentation have not yielded the expected results. The presence of boulders increases equipment wear and tear and requires re-drilling and secondary blasting, which increases the operational cost. The objective of this paper is to optimise the blast design parameters to produce suitable size distribution of rock that can lead to efficient loading, hauling and crushing.

2 Resources and Methods Used

2.1 The Kuz-Ram Model

In recent years, empirical methods for predicting the fragmentation from a given structural geology, rock type, explosive, and blast pattern have become better and more useful, and many of these models have been developed. Yet the Empirical model used for the prediction of the fragmentation for this study is the Kuz-Ram model.

The basic strength of the model lies in its simplicity in terms of the ease of gathering input data and in its direct linkage between blast design parameters and rock fragmentation (Cumingham, 2005).

The Kuz-Ram model consists of three equations; namely, (Cumingham, 2005):

i. Kuznetsov’s equation
ii. Rossin-Ramler’s equation and
iii. Cumingham’s Uniformity index

Kuznetsov’s equation, which predicts the average fragmentation, is given in Equation (1) as;

$$X_{50} = A * \frac{Q^{\frac{1}{6}}}{K^{0.8}} * \left[\frac{115}{RWS}\right]^{\frac{19}{30}} \qquad (1)$$

The Rossin-Ramlers equation is given in Equation 2:

$$R_X = e^{-\left(\frac{X}{X_C}\right)^n} \qquad (2)$$

Cumingham’s Uniformity index is given in Equation 3:

$$n = \left[2.2 - 0.014\left(\frac{B}{D}\right)\right]\left[0.5\left(1+\frac{S}{B}\right)\right]^{0.5}\left(1-\frac{W}{S}\right)\left(\frac{L}{H}\right) \qquad (3)$$

where A is the Rock factor; Q is the Mass of the explosive, kg; K is the powder factor, kg/m^3; RWS is the Relative Weight Strength; R is the weight fraction of fragments larger than X; n is the uniformity exponent; X_c is the characteristic size, X is the fragment size; B is the burden, m; S is the spacing, m; D is the diameter, mm; W is the standard deviation of drilling accuracy; L is the length of the drilled hole, m; H is bench height, m.

Kanchibotla *et al*. (1998) argued that the Kuz-Ram model underestimates the distribution of fines. This deficiency of the model is overcome by introducing a second uniformity index to describe the fine distribution below the mean size. In the case of the finer fractions, it is hypothesised that they are produced by pulverising the explosive in a blasthole.

2.2 Data Collection

This research focused on the Subika pit, where blasting activities were concentrated. The pit was divided into three phases; PH1, PH2, and PH3, which the design grouped as one. The blast parameters were set according to the nature of the material to be mined and the rock formation to be blasted. The parameters used by Newmont Ahafo South Mine are shown in Table .1

The parameters in Table 1 were put to constant review depending on the conditions of the rock and the grade of the ore on the bench.

The specific charge of the explosive was 26.7 g/cm^3. The mechanical stripping method was used in mining the oxide, where the excavators were used to rip the area without a blast. Drill and blast operations were, however, done when hard material was reached. This research took an interest in the fresh materials of Subika pit phase 3. For the purpose of this study, the two shots are named; SK-1144-412P and SK-1144-416P

Table 1 Burden and Spacing

	Waste		Ore	
Weathered	Burden (m)	Spacing (m)	Burden (m)	Spacing (m)
Transition	4.5	5.0	4.2	4.7
Fresh	4	4.5	4	4.5

The analysis of the fragmentation, that is, the volume of material that requires secondary breakage or blasting, has their data collected and analysed. The procedures for each activity were carefully followed, and the data was collected under them.

The drilling and blasting procedures currently adopted were reviewed in order to identify the possible causes of boulders and optimise the blast parameters where necessary to recommend a low-cost solution to the mine.

To confirm if the Kuz-Ram model could predict the optimised blast design for the different benches of interest, digital images for blast shots for SK-1144-412P and SK-1144-416P (Subika pit) were respectively analysed with the Orica Shotplus Software. The result from Shotplus Software and the Kuz-Ram model were compared to ascertain the predictability and reproducibility of the model.

The surveyors use different Global Position Systems (GPS) with spray paint to mark out the blasthole locations, and the drillers drill according to the pattern with an allowable standard deviation of ± 0.01 m. This, however, is very difficult to achieve since there are always variations in the blasthole locations, burdens and spacing owing to the nature of the field.

The drilling procedure currently adopted at Newmont Ahafo South Mines was thoroughly assessed, and some of the parameters are presented in Table 2.

Before charging is done, the surveyors assess the pattern and declare it ready for charging; the holes are checked/dipped for depth deviation using a depth scanner equipped with DIPPlus™ blasthole depth scanning system software so as to prevent any under-drilling or over-drilling. The reversed cone is placed on any under-drilled blast holes to indicate it requires re-drilling, whilst drill cuttings are used to fill the over-drilled blast holes to the planned depth.

Table 2 Ahafo South Mine Blast Design

Parameters	**Value**
Hole Diameter (mm)	165
Burden (m)	4.0
Spacing (m)	4.5
Stemming (m)	3.5
Bench Height (m)	8.0
Hole Length (m)	9.2
Hole inclination (°)	90
Sub-drilling (m)	1.2

Blocked holes are either left out or charged partially, depending on the depth of the blockage. High explosives and detonators are transported from the *Magazine* to the blasting area by specialised vehicles, and the Mobile Manufacturing Units (MMUs) generally convey emulsion. These accessories include cartridges of pentolite (called booster, 150 g and 400 g), capped fuse, NONEL detonators, detonating cords, in-hole and surface delays and safety fuse.

A primer is prepared by inserting a 500 ms Unidet in-hole NONEL detonator into a cartridge of 500 g pentolite (booster). Two methods of charge loading are used at Ahafo South. They are column and deck loading. Column loading is the most frequently used method. The primer is lowered to the bottom of the hole, and the emulsion is poured onto the primer charge to the required depth. This is done by using a pipe connected to an MMU, which is equipped to monitor the density of the emulsion so as to avoid over-loading or under-loading. The charged holes are stemmed using chippings to allow better retention of gases during the blast.

The SHOTPLUS, version 5, is the blasting software used to design the blast to ensure that not more than one hole is fired at a time (single hole firing) to reduce ground vibration. The designed sheet is sent to the field for the surface connections. Surface millisecond (ms) connectors used are 9, 17, 25, 42, 65 and 100 ms. The centre lifting technique is used when there is no free face for a particular blast or to prevent ore dilution.

2.3 Data Collection

A Quality Assurance and Quality Control (QAQC) method is put in place to ensure compliance with the parameters designed in the various plans: drilling plan, loading plan and connection plan. This procedure allowed data to be taken on each step of the blast that would be used to determine the problems to improve on the current parameters in order to reduce boulders.

It has been found by Alireza and Hosseini (2017) that errors are associated with measurements of fragmentation by using image analysis and can be summarised as follows:

i. Sampling error;

ii. Optical distortions; and

iii. Manual corrections of delineation

2.4 Sampling Error

Image acquisition of blasted muck for size distribution analysis is the most critical phase of the analysis. To minimise, if not totally eradicate, sampling errors, the following parameters were critically considered:

i. The location of the image – both randomised, and methodological approaches were used for this investigation. Therefore, the entire block was photographed during loading.
ii. Image angle from the surface of the muck pile – efforts were made to achieve images which were perpendicular to the camera lens

3.3 Optical Distortions

All necessary measures were taken to avoid tilt errors (Lyana *et al.*, 2016). A telescopic camera lens was used in acquiring the images; hence optical distortions were duly overcome. The photos were also taken from an approximately fixed angle.

3 Results and Discussion

3.1 Data Analysis of the Blast Design to Achieve the Mine's Target

The result in Fig. 1 shows that the drilled hole deviation is not the sole factor in the mine's inability to succeed in its initiative but the designed plan. The rest of the study focused on finding the best optimal solution at a lower cost, and since the drilled hole deviation was not the only cause, there was a need to calculate the impact level. The average fragmentation of the two blast shots was evaluated and compared with the average fragmentation according to the design plan in order to calculate the impact level, which was to be factored into the simulation to obtain the optimal result for the Mine.

Blast design for SK-1144-412P and SK-1144-416P were obtained from the Short-Term Drill and Blast Department, and the average fragmentation was computed using the Orica Shotplus Software. The result is shown in Fig. 1. The result in Fig. 1 shows that the blast design currently adopted by the Mine could not achieve better size distribution with the maximum size less than the expected 1000 mm. This necessitated reviewing the Mine's blast design and optimising it to achieve good fragmentation with a maximum size of less than 1000 mm.

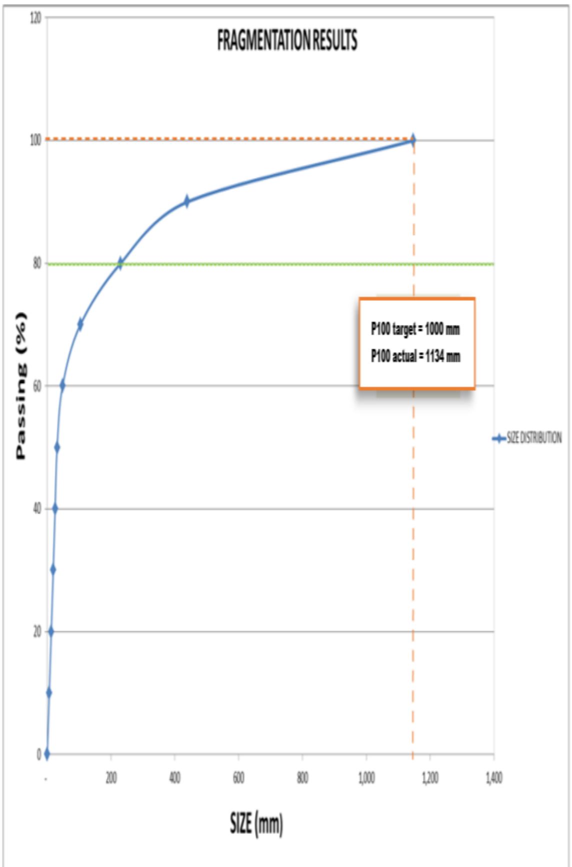

Fig. 1 Designed Fragmentation for SK-1144-412P & SK-1144-416P

3.2 The Collar Deviation Impact level

The post-qualitative assessment methods of the fragmentation for every blast can be done using various methods. In this study, two assessment methods, namely, visual assessment and digital image assessment, were used. However, much attention was given to the digital image assessment using the Orica Shotplus version 5 software. The application of digital image processing is preferred over screening because it is very fast and efficient (Maerz *et al.*, 1996)

A total of twelve pictures of the blasted muckpile from SK-1144-412P and SK-1144-416P were taken with a digital camera and were processed using Shotplus Software. A Sample of pictures taken for the calculation of the size distribution is shown in Figs. 2 and 3, respectively. The pictures analysed were representative areas of the blasted muck, similar to Singh *et al.* (2019) digital image analysis to calculate the mean fragment size and the boulder percentage in a muck pile.

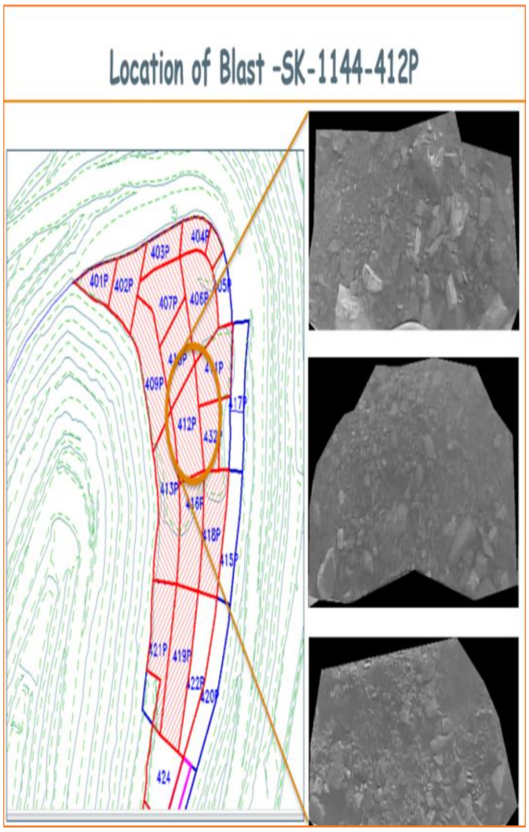

Fig. 2 Images Taken for Fragmentation Analysis for SK-1144-412P

The fragmentation obtained from an identical image is not the same for two different users when using Split-Desktop for analysis because the software is highly user-dependent. This implies that fragmentation results obtained from the same image of muckpile taken for analysis at the Laboratory will differ on two different desktops. Since the fragmentation for this study was done in conjunction with Orica Mining Services' Technical advisor, a more practical and reliable software, the Orica Shotplus, was chosen for this research.

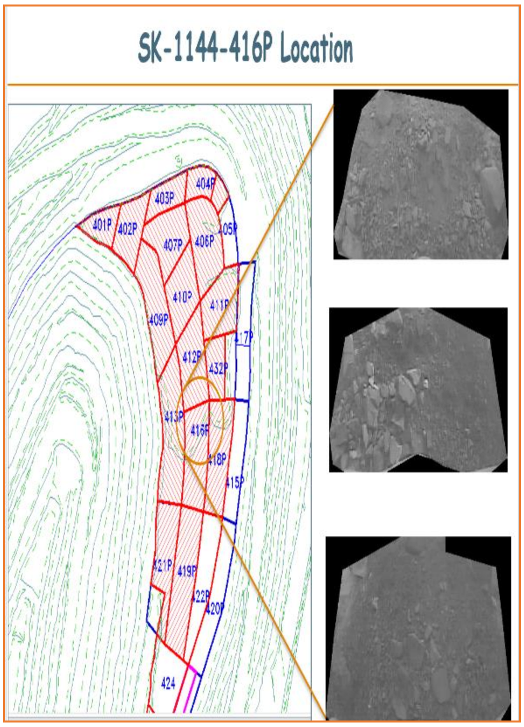

Fig. 3 Images Taken for Fragmentation Analysis for SK-1144-416P

3.3 Computation of Size Distribution

A cumulative percent passing was plotted against the particle size. The cumulative percent passing curves for the current blast of SK-1144-412P and SK-1144-416P are shown in Fig. 4 and 5, with their average size distribution shown in Fig. 6.

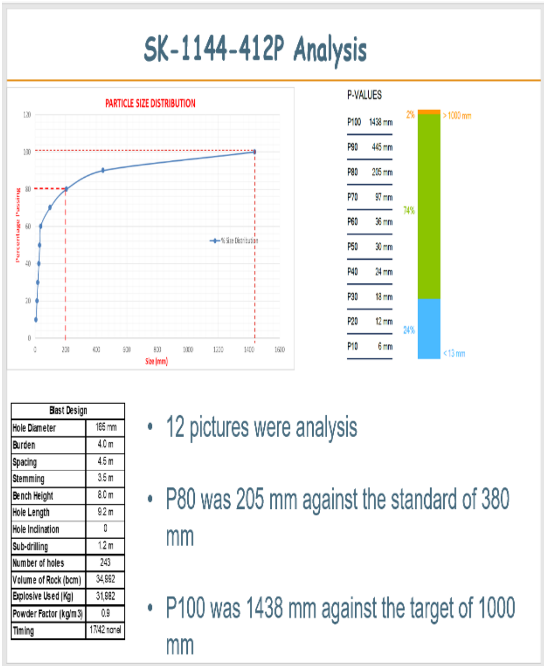

Fig. 4 Size Distribution for the Designed SK-1144-412P

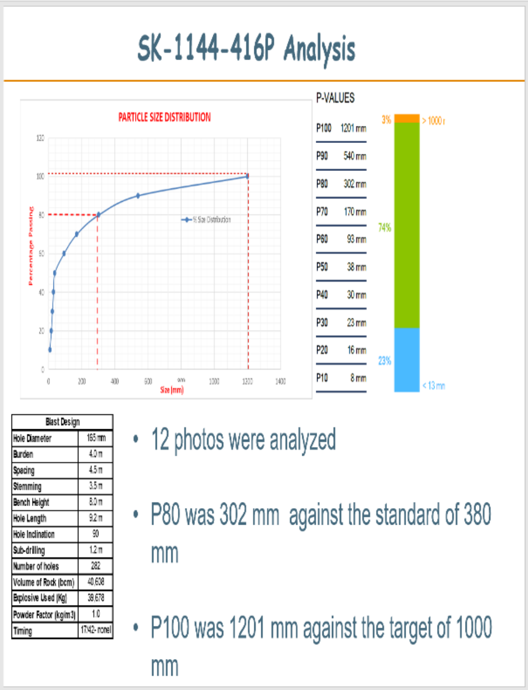

Fig. 5 Size Distribution for the Designed SK-1144-412P

The average fragmentation for the two blast sots is presented in Fig. 6.

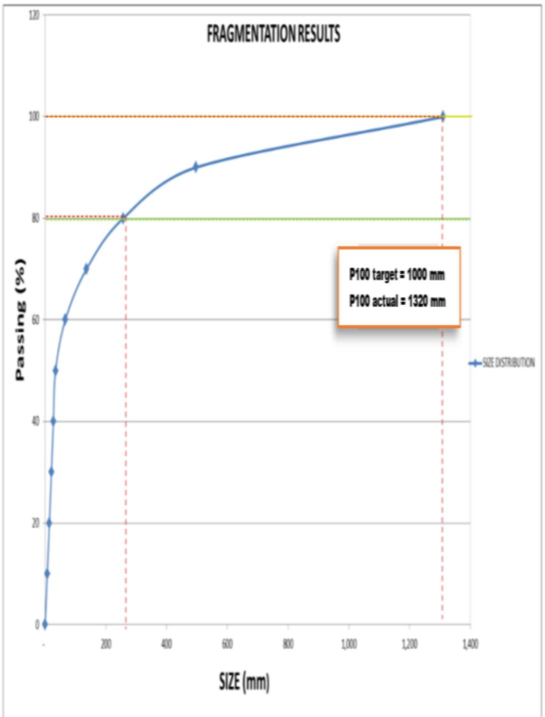

Fig. 6 Actual Fragmentation for the SK-1144-412P & SK-1144-416P

Comparing Fig. 1 and Fig. 6, the blasthole deviation caused the expected p100 value to increase by 16%, as seen in the results of Equation (4):

p100=AF-DF/DFx100 (4)

$$P100 = \frac{Af}{Df}x100$$

where Af is the actual fragmentation sizes, Df is the designed fragmentation size

$$P100 = \frac{1320 - 1134}{1134}x100 = 16\%$$

3.4 The Optimisation

The failure of the designed plan to achieve the targeted fragmentation with a p100 value of less than 1000 mm informed the study to modify some blast design parameters. The study investigated the controllable factors, which (Hustrulid, 1999) classified as follows:

i. Geometric: diameter, charge length, burden and spacing;

ii. Physicochemical or pertaining to explosives: types of explosives, strength, energy and priming systems; and

iii. Time: delay timing and initiation sequence.

This study gave particular attention to the burden and spacing of the geometric parameters whilst keeping all the other blast design parameters used by Ahafo Mine constant, and the objective of the study was accomplished by simulating the spacing and burden parameters using the Kuz-Ram model. The set of parameters (spacing and burden) that produced better fragmentation with a maximum size (p100 value) of less than 1000 mm was selected. The parameters are summarised in Table 3.

The Kuz-Ram model overestimated the size distribution and could not give the maximum size value (X_{max}), the Rosin-Rammler equation in the Kuz-Ram model was supplemented and corrected with the SveDefo formula and the Kuznetsov-Cunningham-Ouchterlon KCO model in order to present a more practical and reliable prediction of size distribution to the management of Ahafo South Mine and also to remove two of Kuz-Ram model's drawbacks:

i. The poor predictive capacity in the fines range; and

ii. the upper limit cut-off to block sizes

The predicted size distributions using the Kuz-Ram and the KCO models are presented in Fig. 7.

Table 3 Optimised blast design for Ahafo South Mine

Parameters	Value
Hole Diameter (mm)	165
Burden (m)	3.5
Spacing (m)	4.4
Stemming (m)	3.5
Bench Height (m)	8
Hole Length (m)	9.2
Hole inclination (°)	90
Sub-drilling (m)	1.2
Volume of Rock (m^3)	40,608
Explosive Mass\(kg)	39678
Powder Factor (kg/m^3)	1
Timing	17/42 – NONEL

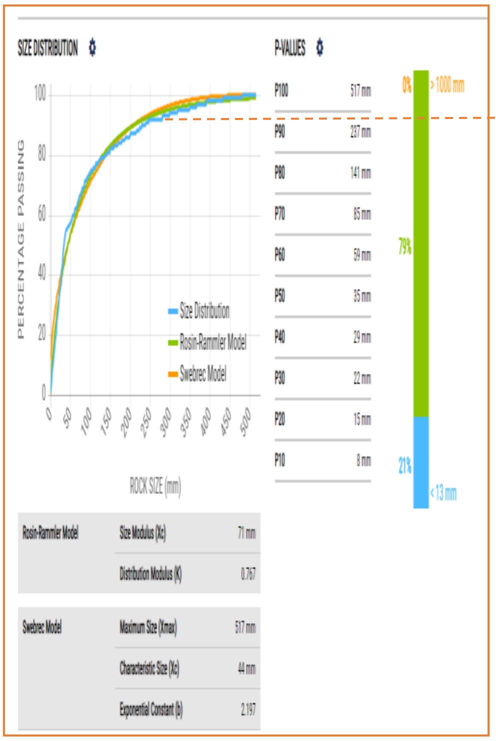

Fig. 7 Size Distribution for the Proposed Design

According to the technical requirement of the Ahafo South Mine, the absolute value for the burden or the spacing distance is considered acceptable when it is within the range of 3 m to 5 m; using this range as a reference, the spacing and burden were simultaneously varied from 3.0 m to 5.0 m, and the optimum blast design to achieve desire size distribution and also the maximum size is less than 1000 mm was observed. This was achieved by closing the current spacing and burden distance from 4.5 m by 4.0 m to 4.2 by 3.5 m.

The results show that the proposed design can achieve the size distribution and the zero-boulder target at the Ahafo South Mine. The maximum size, X_{max} = 517 mm, represents a P100 value less than the Mine's target of 1000 mm.

For high accuracy, the deviation (maximum permissible percentage error is 20%) was computed and factored into the obtained result to give a more practical expected design fragmentation for the Subika pit, as expressed in Equation (5).

$$E = PE*MS \qquad (5)$$

where *E* is the expected size

PE is the Percent Error

*MS** is the maximum expected size

$$\frac{20}{100} * 517 = 103.4$$

The proposed design with an expected p100 value of 517 mm will give a practical p100 value of not more than 517+103.4 = 620.4 mm.

The initial spacing and burden of 4.5 m by 4.0 m gave a spacing-to-burden ratio of 1.13 compared to the proposed 4.2 m by 3.5 with spacing to burden ratio of 1.2. When the spacing-to-burden ratio was increased from 1.13 to 1.20, the mean fragment sizes were found to be reduced, and the maximum target size was achieved. The relation can be explained as increasing spacing-to-burden ratio to a certain extent created thin ledges of rock mass which finally broke into smaller fragments.

However, it must be stated that increasing the spacing-to-burden ratio further resulted in coarser fragmentation which may result from hampered propagation of radial cracks.

In ascertaining the uniformity of the particle size, the Uniformity Coefficient C_u was used, calculated using Equation (6).

$$C_u = P60 / P10; \qquad (6)$$

Table 4 Uniformity Coefficient Analysis

Uniformity Coefficient	Description
$C_u < 5$ mm	Very Uniform
C_u = 5-15 mm	Medium uniformity
$C_u > 15$ mm	Non-Uniform

Cu =P60/P10

where;

P60 is a blasted product with 60% passing particle size in mm

P10 is a blasted product 10% passing particle size in mm

From the size analysis, P60 = 59 mm and P10 = 8 mm

Hence, $C_u = \frac{59}{8} = 7.4mm$

This shows that the size distribution is evenly distributed across a broad spectrum and possesses the desired sizes.

The coefficient of Gradation or Coefficient of Curvature C_g (Measure of the shape of the particle size curve) is calculated in Equation (7) as follows:

$$C_g = \frac{(p30)^2}{(p60 * p10)} \tag{7}$$

where;

P30 is a blasted product 30% passing particle size in mm

P10 is a blasted product 10% passing particle size in mm

P60 is a blasted product with 60% passing particle size in mm

C_g from 1 to 3 shows the distribution is well graded or desired sizes

Hence, C_g = 1.03

This shows that the distribution is well-graded.

4 Conclusions

From the fieldwork, data collection, analysis and discussions, the following conclusions have been made:

i. The optimised blast design to produce a maximum size of less than 1000 mm and also achieve suitable size distribution of rock that can lead to efficient loading, hauling and crushing was found by decreasing the spacing to burden distance from 4.5 m x 4.0 m to 4.2 x 3.5 m and in turn, increased the spacing to burden ration from 1.13 to 1.20

ii. The collar deviation was caused mainly by the presence of water in the pit because the JigSaw software, used by the Mine's Drilling machines, is sensitive to water.

iii. The blast design used by the Mine cannot help them achieve their target of zero boulders (*i.e.* maximum size value of the design plan exceeds 1000 mm)

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