Comparison of the existing and calculated blast design parameters for the rock mass conditions at Bamesso-Latet rock quarry

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ABSTRACT

Blasting is the most cost effective methodology to break rock for mining engineering applications. A good production blast will break only the rock that is needed to be removed, leaving the host rock with minimal damage. Accurate measurement of blast, fragmentation is important in mining and quarrying operations, in monitoring and optimizing their design. Currently, there are several methods available to predict damage due to blasting. The accuracy of many of these methods is questionable, and in most cases, the methodologies over predict the results. This paper presents a practical method (i.e. Langefors's method) that we shall use in Bamesso-Latet rock quarry to do a comparison between the existing and calculated blast design parameters. The proposed method allows to assess the rock damage from blasting. It shows great potential as a practical aid to control and get a good quality of the fragmented material in Bamesso-Latet rock quarry.

INTRODUCTION

Rock damage due to blasting plays an important role in the industrial process in which it is desired in the most efficient and controlled manner. Thus, the elaboration of an effective method (blast design) for rock breakage with explosive is of considerable interest for mining industries. Hence, successful blasting operations can lead to the most appropriate distribution of rock fragment with a minimum production cost (Ash RL, 1963 and Coulombez, 2007).

According to published literature, in blasting operations the use of explosive materials for rock breakage is the main objective to reach a result reflecting the positive influence on the continuation of technical processes (loading, transport and crushing), and to reduce consequently the combined cost of these processes. Blast design and type of explosives are only the variables, the technical processes of loading, transport and crushing depend enormously on the fragmentation obtained after blasting (Nefis M., 2010; Med et al., 2016).

In this context, the rock mass to be discussed is located in Bamesso-Late (Fig.1) which is an opencast site composed of different rock units (granite and gneiss) which can have a significant effect on quarrying operation. The blast design used in the quarry showed insufficiencies which we tried to improve by introducing the Langefors's method. It takes into account at the same time the nature of the rock and the characteristics of explosives adequate to calculate all the parameters of the blasting plan.

GEOLOGICAL SETTING

As a part of Cameroon Volcanic Line (CVL) which has been active over 52 Ma (Millions of years), the surfaces the Mbouda subdivision are composed of volcanic, plutonic and metamorphic rocks like basalts, granites, migmatites and gneiss (Le Marechal A., 1975).

Volcanic rock units are the most represented and extent all along the CVL. These rock units are Cretaceous to present, constituted principally of basalts, trachyte phonolites.

Plutonic and Metamorphic rock units are well represented of age 1800 and 2500 Ma. They are essentially made up of migmatites, gneiss and granites.

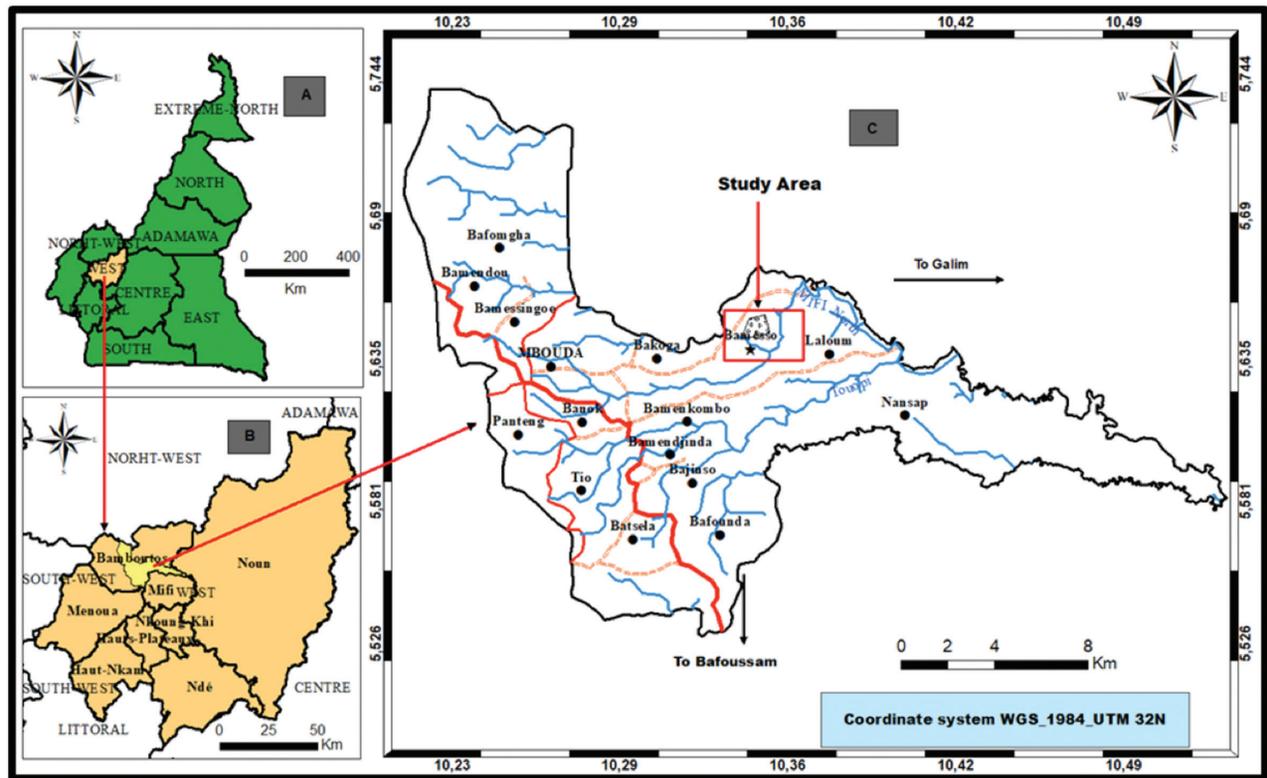The geological map (Fig. 2) illustrates the different rock formation of the Mbouda subdivision.

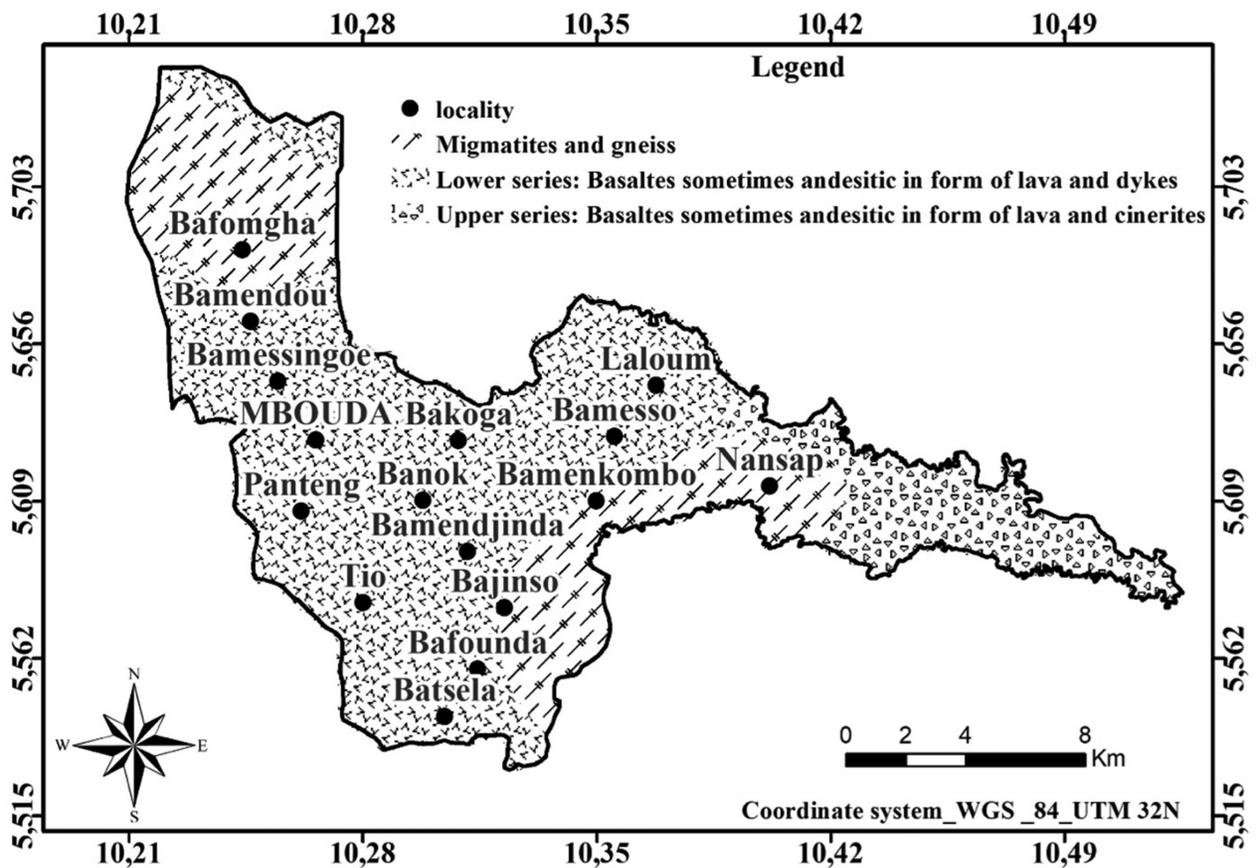**Fig. 1: Location Map of the study area (A: Cameroon Map; B: West Region Map; C: Mbouda Subdivision).**

Fig. 2: Geological map of rock formations (extracted from West and Adamaoua region geological map produce by Le Marechal A., 1975).

METHODOLOGY

Langefors's method was used for blast design calculations. This method uses a semi-empirical formula that permits to calculate the theoretical value of burden (Bth) from five parameters and a constant. This formula is applicable in bench blasting parallel to the free face.

$$B_{th} = 1.08 \, x \sqrt{\frac{S \, X \, Lf}{C \, X \, R \, X \, E/B}}$$

According to Langefors (1963), the following parameters from the above mentioned formula are:

The weight strength (S) or Energy coefficient

This coefficient corresponds to the bottom charge energy or concentration (i.e., shear charge 0.6Sc+ column charge 0.7SB). If the explosives of the two loads are different, and in the general case, it is advisable to calculate an S average balanced according to their distribution.

Linear Charge (Lf)

It is the quantity of explosive per linear meter of hole.

For explosives delivered in bulk, the quantity is calculated by multiplying the volume of one meter hole by the density (d) of the product.

$$Lf_{Bulk} = \pi \times \emptyset^2 \times d$$

For products delivered in cartridges, one calculates the number of cartridges or fraction of cartridge which occupies one meter length of holes. One applies to it a different packing coefficient (K_t) according to the nature of the explosive and one multiplies the result by the unit weight of the cartridges.

$$Lf_{Cartridge} = \pi \times \emptyset^2 \times d \times K_t$$

Packing coefficient (K_t) to apply

- 1,06 to 1,08 Explosive with low consistency (gel, emulsion);
- 1.04 to 1.06 Explosive with average consistency (Dynamite);
- 1.02 to 1.04 Explosive of hard consistency (Explosive powders or nitrated).

NB: The bench of the face to be cut down increases like the square root of the product S × Lf.

As for the energy coefficient (S), the use of explosives of different nature in the bottom charge requires a weighting of the linear charge in order to obtain an average Linear charge (Lf).

Linear charge average =(Lf$_{Cartridge}$×0.6)+(Lf$_{Bulk}$×0.7)/1.3

Inclination Coefficient (C_{in})

During the blasting process, the shock wave of compression is reflected in traction on the free surface. It induces a secondary fracturing which is at the origin of the fragmentation of the rocks. Its effectiveness is proportional to the importance of free surface offered.

At equal bench, the open space surface varies according to the inclination of the blasting face. It increases with the inclination.

The inclination coefficient (C_{in}) is function of the angle made by face makes with the vertical form. It is in this case equal to 1.

For α = 0° Cin = 1

For angles between 10° to 30° the Cin values is:

For α = 10° C_{in} = 0.95, For α = 20° C_{in} = 0.90, For α = 30° C_{in} = 0.85

For intermediate values, one interpolates linearly.

For α =12° C_{in} = 0.94

Resistance to pulling (R)

It takes into account the shear strength of the rock. In the case of a homogeneous terrain, the coefficient of resistance to pulling is of:

0.35 For elastic rocks, 0.40 for average rocks, 0.45 for plastic rocks.

We corrects this value according to the state of fracturing of the rock (Langefors, 1963).

State of fracturing: For an average rock (R= 0.4) if the state of fracturing is very weak, one has a physically homogeneous rock massif. The R coefficient is close to 0.35.Inversely if the rock massif is highly fractured, the coefficient tends towards 0.45.

NB: There is less loss of gas energy and/or shock in a homogeneous rock massif. A low resistance to pulling allows equal load, to increase the bench (Langefors, 1963).

The Pattern or Stiffness Ratio (E/B)

E is the spacing between holes and B the burden. These values are expressed in meters and centimetres generally rounded to 5 cm after calculation. This report influences the granulometry of the products:

For obtaining aggregates one recommends 1 < E/B < 1.3;

For the production of riprap one advises 0.8 < E /B < 1.

A stiffness ratio that is too low damages the average particle size. A high ratio induces a bad cutting of the blasting face (Langefors, 1963).

The Langefors's Constant

Following many tests of validation of the theoretical

formula, Langefors determined a coefficient which is equal to 1.08.

The value *Bth* thus obtained is a theoretical value, which must be corrected according to several parameters which depend on the site conditions. We take into account:

Implantation defaults:

Implantation defaults are fixed values independent of the height of the bench. Depending on the type of errors, the value can be taken as 30 cm for estimation, 10 cm for tamping bar, 5cm for decametre and 1 cm for theodolite.

Errors of positioning the drilling machine

The error due to the attack of the hole is of the order 0.5 to 1 multiplied by the diameter of the drilling bit. It depends on the mode of location and the nature of the terrain at the point of attack.

Drilling deviations

This depends on the type of drilling machine used (bottom hole or out hole), to the natural fracturing of the rock massif. The error is 0.5% × height of the bench.

Average errors generated according to mode of adjustment of the drilling angle or inclination is proportional to the depth of drilling**.** The mode of adjustment with respect to scale is 0.1% for optic, 1% for decimeter and 2% for wire. These values are averages and must be adjusted according to specific conditions of the operating site: quality of the material, nature of the field, care brought by personnel during blast hole implantation and drilling, etc.

The error is multiply by the bench height. The real spacing (E) is $E = 1.25 \times B_R$.

It is admitted that all the preceding errors (ε) cumulate in an unfavorable way and that there is no compensation. In this case, the reel or corrected burden value (BR) is equivalent to**:**

$B_R = B_{th} - \sum \varepsilon$

Charge calculations

For charge calculations we need to calculate the charge length and the weight of explosives for each charge defined previously. Calculations are done by using the true burden value (B_R) and the different linear charge calculated. It is represented by the following equations:

For bottom charge (Q_B)

$Q_{shear\ charge} = 0.6 \times$ True burden (B_R) × Linear charge (L_f)

Linear charge (L_f) = $\pi \times$ Radius ($\emptyset^2$) × Density (d) × Packing coefficient (K_f)

Q column charge for the bottom hole = 0.7 × True burden (B_R) × Linear charge (L_f)

For Column charge (QC):

Explosive in bulk: Q_{Column} = Charge length (CL)× Filing coefficient (K_{re}) × Linear charge (L_f)

Charge length (CL) = (Hole length (HL) + Sub-drilling (S_f))-(Length of bottom charge (Q_BL) + Stemming length (SL))

Cartridge explosive: $Q_{Column} =$

$$\frac{\text{Charge length (CL)} \times \text{Filing coefficient (Kre)} \times \text{Linear charge (Lf)}}{\text{Weight of cartridge (}\rho\text{)}}$$

Powder Factor (PF)

Powder factor for a single borehole is calculated as:

PF (kg/m^3) = Quantity of explosive per hole / Volume of rock to blast

The charge column is calculated according to filling coefficient (Kre). It is the relationship between the overall lengths occupied by the explosive added on the overall length of the charge (explosive and inter-mediate stemming).

Filling coefficients according to the type of explosives can be taken as 0.38 - 0.45 for dynamites, 0.40 - 0.60 for ammonium nitrate, 0.50 - 0.70 for emulsions and Gels and 0.60 - 1.00 for ammonium nitrate and fuel oil (ANFO).

RESULTS

Bench blasting calculations

In the quarry, two types of explosives were used for calculations: Emulstar 6000 (Titanobel type, diameter 70 mm, density 1.28 g/cm^3, strength of 0.98 MJ/kg) for bottom charge and ANFO (Maxime type, density of 850 kg/m^3, strength of 0.77 MJ/kg) for column charge.

During drilling operations the following parameters were used:

Diameter of drilling bit ($\emptyset$):   89 mm
Bench height (BH):   10 m

Inclination of blast hole (Cin): 10°

Sub-drilling (Sf): 0.3 B (B: burden)

Type of drilling: Bottom hole

Mode of implantation: Theodolite

Mode of adjustment of drilling angle: Optic

Rock type: Granite & Gneiss

Table 1 present the resulting values obtained from calculations done by Langefors's formula.

Table 1: Blasting parameters calculated according to Langefors's method.

As per Langefors	Values
(Strength energy (S) (Mj/kg	0.867
(Average linear charge (Lf) (kg/m	5.30
(Inclination (C_{in}	0.95
(Resistance to pulling (R	0.40
(Spacing (E/B	1.25
Langefors constant	1.08

From the above results obtained in Table 1, the theoretical burden is calculated as follows:

$$Bth = 1.08 \, x \sqrt{\frac{SXLf}{CXRXE/B}} = 3.36 \text{ m}$$

The following corrections are applied to the calculated theoretical burden:

Corrections applied

Implantation Error = 0.01 m

Mode of adjustment of drilling angle = 0.1% ≈ 0.001

Position of drilling machine= 0.75×0.089 =0.06675 m

Drilling deviation = 0.005×10= 0.05 m

Corrected burden (B_R):

$B_R = B_{th} = \sum \varepsilon = 3.36 - (0.013.23 = (0.05 + 0.06675 + 0.001 + $ m

Real spacing (E) = 1.25 × 3.23 = 4.04 m and stiffness ratio = 4.04 × 3.23 m

Final stemming length (SL) = B_R = 3.23 m

Explosive charges are calculated as shown below and followed by an evaluation of the powder factor.

For bottom charge (Q_B)

Q shear charge = 0.6×True burden (BR)×Linear charge (Lf) for Emulstar 6000

= 0.6 × 3.23 × 5.32 = 10.31 kg

Number of cartridge required:

2.08 kg ⟶ 1 cartridge

10.31 kg ⟶ 10.31× 1/ 2.08 = 4.96 ≈ 5 cartridge

Length of bottom charge ($Q_B L$):

1 cartridge ⟶ 0.43 m

5 cartridge ⟶ 5 × 0.43 /1 = 2.15 m

For Column charge (QC)

Charge length (CL) = (Hole length (HL) + Sub-drilling (S_f)) - (Length of bottom charge ($Q_B L$) + Stemming length (SL))

= (10 + 0.97) – (2.15 + 3.23) = 5.59 m

Q Column = Charge length (CL) × Filing coefficient (Kre) × Linear charge (Lf) for ANFO

= 5.59 × 1 × 5.29 = 29.57 kg

Table 2: Comparison between existing and calculated blasting data.

Parameters	Existing Data	Data calculated by Langefors's method
Type of Explosive use	Emulstar 3000 and ANFO	Emulstar 6000 and ANFO
(Diameter of explosives (mm)/ Weight (g	mm / 1560 g 60	mm / 2080 g 70
(Hole depth (m	10.5	10
(Sub drilling (m	0.84	0.97
(Burden (m	2.813	3.23
(Pattern ratio (m	3.52	4.04
(Stemming length (m	2.813	3.23
(Quantity of Emulstar per hole (kg	6.36	10.31
(Quantity of ANFO per hole (kg	39.35	29.57
(Total quantity of explosives per hole (kg	45.71	39.88
(Volume of rock blasted per hole (m³	103.96	130.5
(Powder factor (kg/ m³	0.44	0.31

Powder factor (PF):

PF (kg/m³) = Quantity of explosive per hole / Volume of rock to blast

$$PF \ (kg/m^3) = \frac{10.31 + 29.57}{4.04 \times 3.23 \times 10} = 0.31 \ kg/m^3$$

Comparison of blasting parameters

Table 2 gives a comparison between existing blasting data and those calculated by the Langefors's method using Emulstar 6000 for 10 m bench height followed by a blast design as seen in Figure 3.

Economic evaluation of explosives

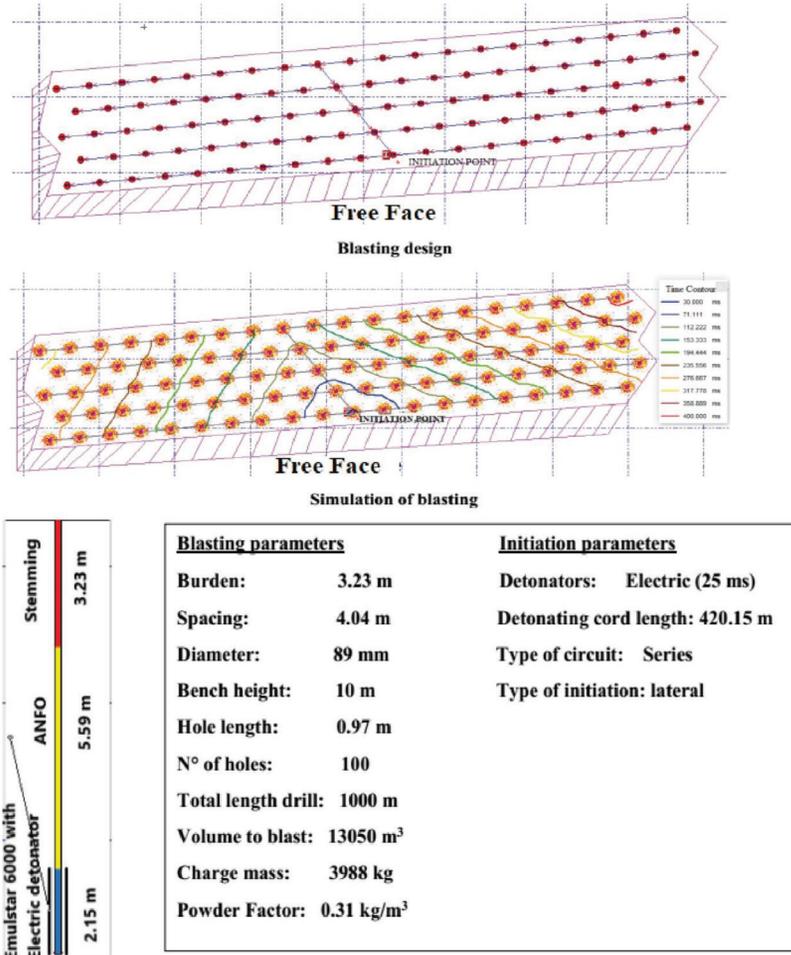Table 3 presents a brief account on the economic evaluation of explosives.

Fig. 3: Blasting data and Simulation of blasting.

Table 3: Comparison on explosive evaluation between exiting and calculated budget.

	Existing Data	Calculated data
(Bench height (m	10	10
(Number of holes (N	120	100
(Total volume of blasted rock (m³	12475	13050
(Total mass of foot charge (kg) ($M_{fc} = N \times Q_{Emulstar}$	763.2	1031
(Total mass of column charge (Kg) ($M_{cc} = N \times Q_{ANFO}$	4722	2957
Total amount of foot charge FCFA	3510720	4742600
Total amount of column charge FCFA	3966480	2483880
Electric detonators /Unit FCFA	34300	29400
Detonating cords / m FCFA	184498	176463
Total amount of explosives FCFA	7695998	7432343
Difference in price of the 2 explosive FCFA	263655	

DISCUSSION

The results obtained during calculations using Langefors's method allows to compare between the exiting and calculated blast designs.

According to previous blasting calculations mention above (3.36 m), the burden calculated is appropriate for blasting which must be indeed corrected from errors such as drilling deviations, position of drilling machine, implantation and inclination of blast holes. The resulting burden after corrections is 3.23 m. As a result, the real spacing is $1.25 \times 3.23 = 4.04$ m and the stiffness ratio 4.04×3.23 m.

The calculated sub-drilling is $0.3 \times 3.23 = 0.97$ m compared to the less value (0.84 m) used in the quarry (see table 4). The existing stemming length (2.81 m) is less than the calculated stemming length (3.23m). An inadequate stemming causes premature venting of explosion gases which may create flyrock and air overpressure while reducing the effectiveness of the blasting (Langefors, 1963).

About 45.71 kg of explosives per hole was used by the company and less volume of rock was blasted per hole (103.96 m^3) compared to 39.88 kg of explosives and 130.5 m^3 of blasted rock per hole is obtained using Langefors's method (see Table 4).

The specific consumption of explosives used in the quarry is 0.44 kg/m^3 which is slightly higher compared to calculated value of 0.31 kg/m^3. This means that the loading plan is optimal that is, the height of the explosive charge as calculated by the Langefors's method will result to a good fragmentation hence reducing the size of blocks.

CONCLUSIONS

A comparison of the existing and calculated blast design parameters shows promising potentials for application of Langefors method for rock breakage. However, proper considerations of geomechanical parameters of the rock mass that strongly influence the performance of blasts is the future research challenge in the application of the Langefors method.

AUTHOR'S CONTRIBUTIONS

S. Ousmanou contributed in providing field data, choosing an adequate blast method to use, calculating the blasting data being provided and establishing a blast design. N. Blaise contributed in providing the calculated blast data of the quarry (which is confidential). F. Eric contributed in drawing the location and geological map of the study area.

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