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Analysis and Optimization of Blasting Practices at the Sangaredi Mine

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Abstract

Generally, the Mos hardness of bauxite is 2.5 to 3.5. According to the specific conditions of the Sangaredi bauxite deposit, that is, the rock hardness coefficient is between 3 and 6, and there is a clayey zone in the bauxite mining area, it is necessary to carry out blasting work before mining. This article mainly analyzes and optimizes the blasting practice of the Sangaredi open pit bauxite mine. The subject was finally extended to the reduction of nuisances due to blasting, that is to say essentially due to vibrations: this presupposed a study around the vibration data available, and the proposal of methods to limit these vibrations. The bottom conditions of the Sangaredi bauxite ore zone are studied. Bauxite reserves are very rich and the market is huge. The analysis of mine blasting practices, mainly the types of explosives, the selection and analysis of explosives and blasting equipment and choosing the mode of longitudinal blasting, the link of the blasting site of the mining area, further study the optimization of mining blasting practice, the choice of drilling and process to determine reasonable blasting parameters, which improving the drilling method. The experimental results show that this optimization scheme improves efficiency blasting of the bauxite mine but also promotes an increase in production.

Keywords

Open Pit Mine, Blasting Parameters, Blasting Effect, Bauxite

1. Introduction

The harmonious development of science and technology inevitably involves the extraction and development of useful minerals, the basis of technological progress in the contemporary world. Mining activities stimulate the development of backward regions; they improve the professional and technical qualifications of nationals; and for some countries they constitute a nucleus for economic development (creation, multiplication of jobs). The Guinean territory is one of the rare areas of the globe to be endowed in sufficient quantity and in appreciable quality with numerous and diverse natural resources: plant, animal and mineral.

For the development of useful minerals, there is a need for a process which is the very first activity which is blasting, either by mechanical blasting, hydraulic blasting or blasting by explosives. The Guinea Bauxite Company (CBG) which is a metallurgical bauxite production company created in 1963, following an agreement between the Guinean Government and the American company Harvey Aluminium Company (CBG) extracts bauxite by blasting which consists of in tearing the rock from the virgin massif and fragmenting it using the energy of the explosion of the explosive charges placed in the cylindrical holes of small diameter called a blast hole. Taking into account the specific conditions of the bauxite deposits of Sangaredi, in particular the presence of clay pockets in the form of clusters and the hardness coefficient of the rocks according to the scale of Professor Protodiakonov ($f = 3$ - 6), it requires blasting works which are very important processes during the exploitation of hard and semi-hard ores, they aim at the fragmentation of the rock in order to facilitate its loading, its transport, and its treatment mechanical. The need for uninterrupted work on the machines on the one hand and on the other hand the compaction of the slaughtered mass remains the key to a good organization of shooting work. This is why this thesis has the title "Analysis and optimization of blasting practices in the Sangaredi mine". Blasting is the simplest and most widely used technique in quarries and massive rock mines today. It makes it possible to fragment large volumes of rock for the recovery and processing of the material blown. Being the first step in the process, blasting is a key part of the chain: it is the first part of the industrial process, and in particular the first part of the particle size reduction chain. Slaughter therefore plays an important role in the exploitation of minerals: the control of the shooting will make it possible to guarantee safety during the operation, to limit the costs generated by the operation itself, to reduce the nuisances and meet regulatory constraints, and finally obtain a grain size suitable for the installations and what you want to do with the product.

At present, the successful excavation of rock masses requires appropriate blasting design of the drilling pattern, quantity and type of explosives and initiation sequence (Dowd & Onur, 1993; Lerchs & Grossmann, 1965). It is important to determine the burden of cut holes, which is key in the overall blasting procedure. The basic principles behind calculating patterns and charges for a four-section cut (known as the Swedish method) were first developed by Langefors and Kihlström (Zhao & Kim, 1992).

The method was later updated by Holmberg and then simplified by Olofsson (Anon, 2000). These modified methods are applied to the calculation of cut blasting design for shallow rock masses. However, an optimized method has not been reported for determining the cut hole burden in a deep rock mass. In LS-DYNA, three main damage models are used to simulate the damage evolution of rock mass under blasting load (Anon, 2004; Abrand et al., 2014). In contrast to the HJC and JH series models, the RHT model considers strength characteristics in the three-dimension stress space, along with deformation and failure under high confining pressure. It can better reflect rock mechanical performance under different confining pressures and high strain rates. When the D&B methods applied, the excavated deep rock masses are under dynamic-static coupling loading. At such a moment, strain rate, confining pressure, strain hardening, and damage softening have a significant influence on the mechanical performance of the rock masses.

These factors are comprehensively considered in the RHT model to investigate the optimized method and determine the burden of cut holes applicable to deep rock masses, the RHT model in LS-DYNA is chosen (Angeline, 1998).

To apply the RHT model to the evolution of rock damage evolution under blast loading, the rock mechanical parameters are first determined based on existing experimental results (Caccetta & Hill, 1987; Whittle, 1990). The model parameters and model rationality are verified by comparing the model results to existing blasting test and simulation results (Anon, 2016a). To overcome the difficulties encountered during cut blasting in rock masses, the determined RHT model parameters are applied to a simulation of the damage distribution around a blasthole under hydro-static pressure and various lateral pressure coefficients. Through the simulation, the causes of difficulties encountered during cut blasting are analyzed, and the cut blasting design is optimized to overcome these difficulties. This study provides both a method for determining RHT model parameters in LSDYNA and a theoretical basis and reference for addressing excavation difficulties related to cut blasting in deep rock masses.

Effectiveness of hard rock blasting are measured with two basic indices that are oversize generation and blast-hole productivity, cost per ton of rock blasted is also another index that measures the effectiveness of blasting and are dependent on rock mass and blast design parameters such as hole diameter, burden, spacing among others. Pointed out that the determinant parameters differ from one mine to the other and some of the blast design parameters could be regulated to deliver the desired blasting effectiveness. The individual influence of the determinant parameters on blasting effectiveness has been studied by several authors, but their cumulative influence on the same is yet to be formulated. However, the huge statistical data generated from the well organizes and documented large scale hard rock surface mines operating variable conditions worldwide constitutes the only readily available resource which could be used for the analysis and regression models of indices that determine effectiveness of blasting of the rock blasted fit on uncontrollable and controllable blasting parameters.

Efficiency of blasting operation in underground and surface mines determine to a large extent utilization of equipment, productivity and economics. Proper fragmentation of blasted rocks improves the efficiency of downstream operations, viz. loading, transport and crushing to desired sizes. An optimal blast not only results in proper fragmentation but also reduces undesirable effects like ground vibration, fly rock and formation of toe in quarry benches (Anon, 2016b; Ataei, 2016). The drilling and blasting is the first unit operations in the mining process and has a major impact on the performance and cost of subsequent unit operations. An increase in the degree of fragmentation will give the loading equipment a higher rate of productivity. This will result in lower costs per ton or cubic yard moved. The effect of wear and tear will also decrease, giving lower operating cost per hour. Under similar conditions of haul, lift, size and type of truck, and haul road condition, truck production per hour will increase with greater degree of fragmentation due to faster shovel or loader loading rates and a decrease the crusher. There will be a consequent decrease in cycle time.

Guinea has the world's largest reserves of bauxite, the ore used for the production of aluminum. With high alumina content, Guinean bauxite reserves are estimated at more than 7,400,000,000 tons, of which 2,300,000,000 tons are located in the Boké region (See **Figure 1**).

Guinea has the world's largest reserves of high-quality bauxite—Financial Times Highly competitive, Guinean production now supplies the main world markets. The reserves are mainly located in the following g regions: Lower Guinea (Ataei & Osanloo, 2003; Siskind, 1999).

Mining is based on a depleting resource and the prices of commodities and costs are forecasted using different techniques. These two are the major challenges in analysis and optimization processes. Commodity prices have become very volatile and extremely unpredictable. The depleting nature of the resource and the existence of unknown geological structures exacerbate the problems to planning and optimization. The techniques described herein attempt to develop ways of fully exploiting the resources. Each optimization area has got its own challenges and the techniques used also have their own

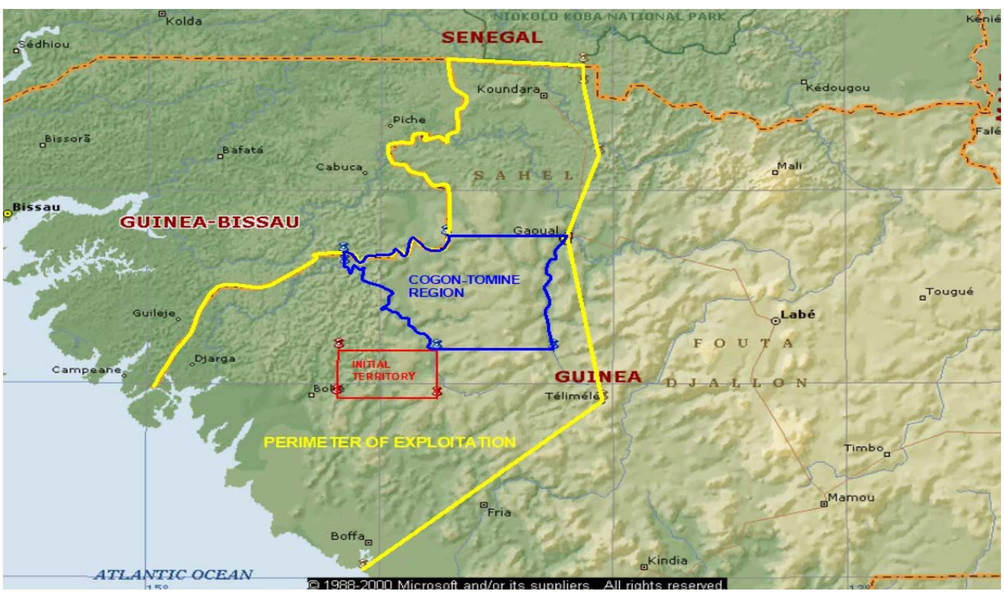

Figure 1. Geographic map of the mining concession (CBG).

limitations. Environmental considerations are becoming extremely important in modern mining. There is an increase in stakeholder requirements and this increase is coupled with uncertainty of the requirements. Inclusion of these factors and obtaining a balance is very difficult, and failure to include them only leads to decreased profits. Additionally, the technique available assumes a number of parameters which in their nature may be dynamic. This prompts planning and optimization practitioners to force real world problems to fit into the developed techniques. The solution obtained thereafter does not represent the actual real world solution. Although heuristic techniques have tried to combine other techniques in order to solve different problems, there exist difficulties in creating one technique that can address all the challenges. The use of particle swarm optimization requires the application of other techniques as it converges prematurely giving sub-optimum solutions (Langefors & Kihlstrom, 1963). Techniques that are not heuristic create formulations that are too large and as such they are costly to solve and difficult to formulate requiring a significant amount of time to solve (Kahriman, Ozkan, Sul, & Demirci, 2001). Smit and Lane (Fleurisson, 2001) identified additional challenges to the field of planning and optimization. These challenges include: The difficulty in connecting labor profiles and production profiles presenting complications in developing the operating cost models; the continuous changes in mining layouts which shows that there is no sticking to planned layouts. The newly implemented layouts are not optimized which affects optimization in the specific area and the whole value chain; The relaying of planned objectives across all levels of work has a disconnection which results in planners optimizing against the strategic vision of the company as they locally try to optimize within their departments; The time allocated to conduct optimization process is very little considering the number of alternatives and scenarios that must be run in order to get an optimized solution and The response to continuously changing business environment is very poor especially to changing exchange rates, commodity prices and costs. This suggests inefficiency of optimized plans. The failure of most mines to meet expected targets shows that there is a gap in planning and optimization. Ataei & Osanloo (2000) also identifies that failure to optimize cut-off grade is a result of not using different cut-off grades, which are dynamic in nature. Lack of optimization and poor design can have adverse effects such as: inability to achieve maximum ore recovery and maximum profit or premature closure of the result of any (Kahriman, 2000).

2. Research Methods and Technology

This research study methodology comprises three parts. These include,

1) The pre-field study,

2) The field study and,

3) Mining software (whittle programming Pty Ltd and Gemcom Surpac).

Pre-field study involves desk studies such as technical books, international mining journals, internet research, program software, and all methods and materials that are involved in creating a block modeling, doing an analysis and optimization of blasting open pit design and scheduling.

Field study involves data collection, field test, laboratory test and analysis. Field survey is basically performed to examine the site of interest prior to major investigations.

Data collection is aimed at systematic collection of input parameters which are necessary for the study.

Post field study involves the analyses of the pre-field study and field study to comprehensively determine the realistic solution(s) for the study in question.

The work involved in analysis and optimization of blasting is so laborious that it is almost impossible to do it manually. Fortunately, there are computer software such as Data mine, Mine map, mine shed, Surpac and Whittle that assist mining engineers to do the work. In this study, Surpac and Whittle software will be used simply because of their availability, they are also widely accepted in mining industry. Both software are menu driven and combine as a powerful tool for open pit analysis and optimization of blasting (Figure 2).

3. Analysis of Blasting Practices on the Mine

Taking into account the special conditions of the Sangarédi bauxite deposit, especially according to the proportions of Prof. PROTODIAKONOV ($f = 3 - 6$), blasting is required. Hard and semi-hard ores are mined to break up the rock for loading, transport and mechanical handling. For this, the following conditions must be met:

1) A sufficient size of fragmentation and a minimum amount of large block and dust;

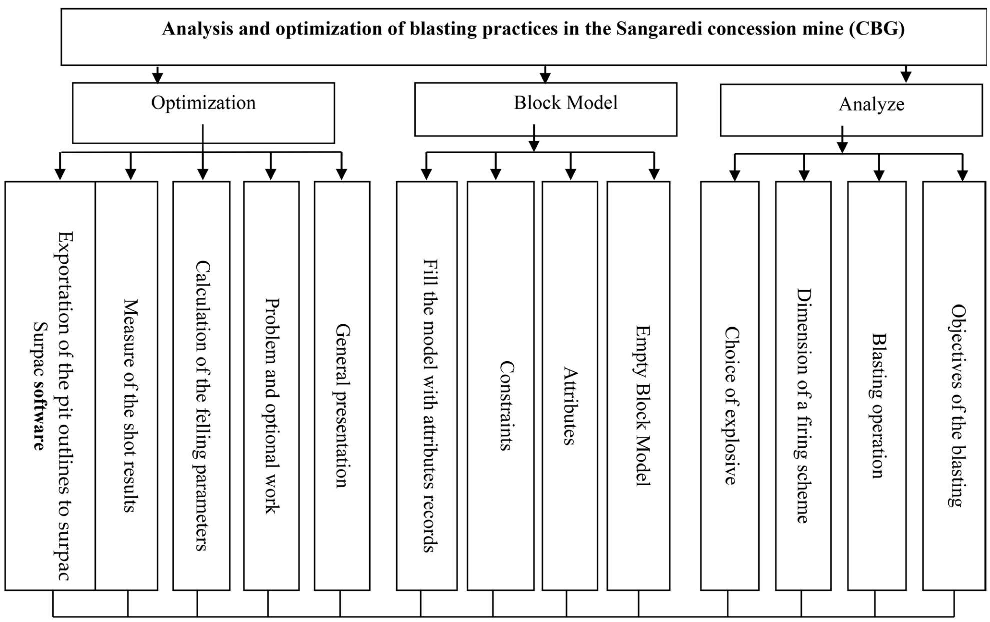

Figure 2. Technical road map.

2) Sufficient safety for staff and work equipment;

3) A high economic yield and uninterrupted work of the quarry machines;

4) A site with a sufficient felled mass.

3.1. Types of Explosives

The selection of explosives must consider the following factors:

The mechanical and physical properties of the rock;

The expected degree of fragmentation;

The hydro-geological conditions of the deposit;

- Operational safety ("sensitivity");
- Purchase price
- Climatic conditions

These various explosions are carried out by the most common mine explosives are: Anfo, Heavy Anfo, Emulsions, Low density and Packaged.

3.2. Analysis and Choice of Explosives

Drill and blast mining is a common method used to break up "beds" of rock, to send small pieces of stone containing ore to the processing plant, where the valuable ore is then separated from barren rock. As the name suggests, holes are drilled in a section of rock—above or below the ground—and explosives are placed in the drill holes.

In the use of explosives in mining operations, the volume of explosives required varies greatly between mine sites depending on the mining method used as well as the type of rock and its hardness.

Figure 3 shows us not only the resistance of the different explosives in water, their shock/heave energy but also the purchase cost.

Taking into account the hardness of the rocks, the cost and the energy; Anfo is better placed for the fragmentation of rocks from the Sangaredi mine.

But as the mine is in a zone of two seasons (rainy and dry) to avoid anomalies it will be preferable to choose another explosive which can resist in water, ensures the fragmentation of the rocks and means costs on those the engineers have chosen the Emulsion.

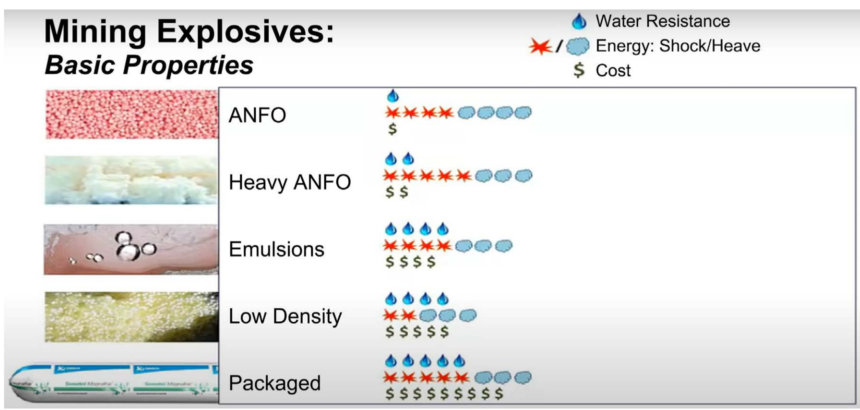

Figure 3. Mining explosives.

Considering the hardness from bauxite to sangaredi, according to Prof. protodiakonov's scale, from 3 - 6', explosives are required for blast. The two explosives used in CBG are Anfo and Emulsion.

ANFO is a mixture of ammonium nitrate (94%) and gasoline (6%). The density is 0.9 and the detonation speed is 400 m/s. It is easy to dissolve in water. Its cargo loaded in the pit uses an ANFO truck ("UMF mobile manufacturing unit") with a capacity of 12t, either in bulk ("dry condition") or in line ("wet condition" and fractured rock).

Its advantages and disadvantages lie in its high efficiency and low cost in arid areas, while its main disadvantage is its sensitivity to water.

ANFO (or AN/FO, for ammonium nitrate/fuel oil) is a widely used bulk industrial explosive. Its name is commonly pronounced as "ANN-foe".

It consists of 94% porous prilled ammonium nitrate (NH_4NO_3) (AN), which acts as the oxidizing agent and absorbent for the fuel, and 6% number 2 fuel oil (FO). (Show the **Table 1** for the technical parameters and **Figure 4** for the anfo composition)

ANFO has found wide use in coal mining, quarrying, metal mining, and civil construction in applications where its low cost and ease of use may outweigh the benefits of other explosives, such as water resistance, oxygen balance, higher

Table 1. Anfo technical characteristics.

PARAMETERS	UNIT	VALUES
Density of the explosive	kg/dm^3	0.9 - 1
Moisture resistance	Hours	8
Explosive energy	kcal/kJ	(800 - 940)
working capacity	cm^3	(320 - 330)
Breakage	Mm	(15 - 20)
State of use	-	in grains
Detonation speed	m/s	4000
Power coefficient	-	1.13

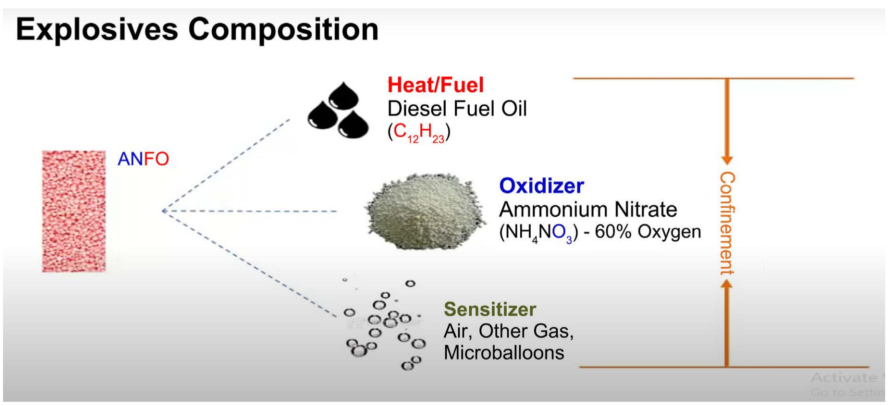

Figure 4. Anfo composition.

detonation velocity, or performance in small-diameter columns. ANFO is also widely used in avalanche hazard mitigation.

3.3. Analysis of the Longitudinal Combustion Scheme

At present, mining companies use a variety of programs and shooting accessories. The success of long jump depends on the correct choice of jump shot plan and shooting accessories. The blasting scheme shall ensure good rock fragmentation and provide convenient results for loading and processing.

Filling is done with a four meter long rod (blast hole connector); after loading and filling the blast holes, the different holes in one row should be connected in a sequence of 17 ms, 25 ms by numbered surface joints (See Figure 5). For drill meshes, it depends on the mechanical and physical properties of the rock and the depth of the holes. The Sangaredi mine uses the following grids: (6 × 5) m; (6 × 5.5) m; (7 × 6) m.

4. Optimization of Blasting Practices on the Sangaredi Mine

4.1. Annual Production Planned on the Sangaredi Mine

Planning is a practical means of designing an ideal future and mobilizing people to achieve it. In the mining environment, it is to plan the blocks that can be mined in time and space, and specify the means to achieve the goal.

In this Table 2 we see the instability of production and especially at the level of 2019 and 2020 records the lowest production of the last six years. we see that the CBG has provided a lot of effort in 2021 compared to the last 5 years with a production over 20 million tons per year, on this, the article will allow us to

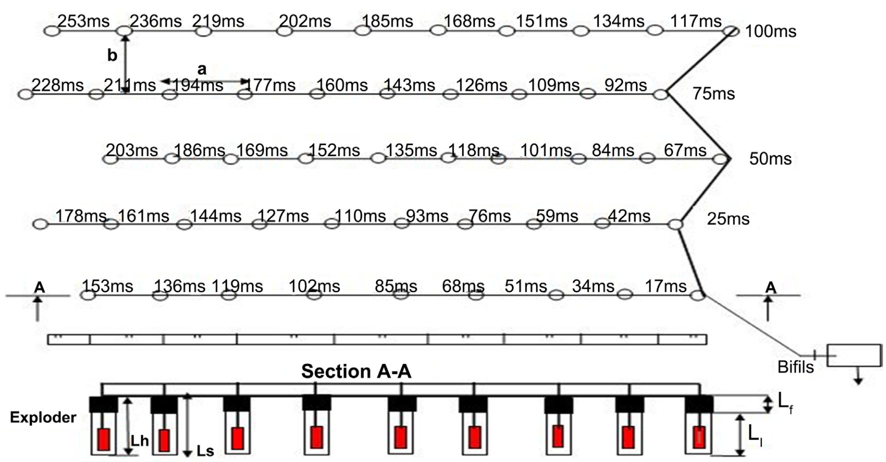

Figure 5. Illustration of the parameters constituting a firing diagram.

Table 2. Annual production 2016-2021.

ANNUAL PRODUCTION 2016-2021 in tons						
Years	2016	2017	2018	2019	2020	2021
Production	17,022,186.68	17,716,200.20	17,454,431.60	14,568,858	15,504,607.10	**20,132,000**

examine the data for a progressive production, that is to say to have a production always greater than or equal to that of 2021.

4.2. Blasting Parameters

This method is crucial in this study, because it can not only observe and understand the drilling diameter, but also the drilling diameter must be suitable for the nature of the rock and the height of the rock mass.

In particular, it is used to determine the input/output law of the mechanism. Therefore, it is necessary to introduce a geometric dimension which can reflect the relative position between different parts.

The drilling and removal of this trench will be carried out by Nitrokemine, just as it did at the entrance to Boundou-waade and N'dangara. Therefore, there is no need to purchase equipment. There are two types of drill bits available.

Guide slope: it depends on the type of transportation and is determined as follows:

$$i_p = t_g\,;\ \ i_p = \frac{Hg}{Lt}\,;\ ‰$$

The ore is transported by truck. Trucks transport bauxite cleaner to N'dangara's trap. The allowable slope is $i_p = 80‰$.

Length of trench: Since the depth is large, in order to provide a small slope, we do not require the length, but we determine the length according to the following expression:

$$Lt = \frac{Hg}{i_p} \times 1000 = \frac{7}{80} \times 1000 = 87.5\ \text{m} \Rightarrow La = 87.5\ \text{m}$$

1) The turning radius of these trucks is 12.5 meters. We don't consider the circular design because we don't want a lot of excrement. In order to speed up the progress of the project, let's use the bottom of the bag design. The rotation angle of the excavator is between 60˚ and 90˚ and the cycle time is short. The truck driver stands opposite the shovel and has good visibility, which means that in this particular case, the excavator is more cost-effective.

So, we're going to choose a pattern from the bag $B = 37$ m.

In this case, the appropriate type of program is the butt.

$$b = 2p + \frac{L_p}{2} + d + \frac{L_c}{2}$$

$L_c = 13.3$ m is the width of the truck.

$L_p =$ It's the length of the shovel.

$d =$ It's the steering diameter.

$p =$ It's a safe distance.

$$b = 2 \times 3 + \frac{6.28}{2} + 25 + \frac{5.21}{2} = 36.7 \Rightarrow b = 36.75\ \text{m}$$

2) Cross section: Since the profile of the trench is four rings, its calculation is as follows:

$$Sta = \frac{Hg\left(2b + 2Hg\cot\mathrm{g}\,\beta\right)}{2} = Hg\left(b + Hg\cot\mathrm{g}\,\beta\right)$$

$$Sta = 7\left(37 + 7\mathrm{cotg}80\right) = 267.64\ \mathrm{m}^2 \Rightarrow Sta = 267.64\ \mathrm{m}^2$$

Hole diameter: The procedure for hole diameter depends mainly on the type of mining and geology of the site. Each configuration has a clear program.

In choosing our options, we have looked at the options generally adopted to select the right diameter that best suits our situation. This diameter is theoretically calculated to compare with the diameter used in the field.

The working group shall decide on the basis of the following formula:

$$D_{opt} = \frac{\left(Hg \times \mathrm{cotg}\,a + C\right)\sqrt{y}}{30\left(3 - m\right)};\mathrm{m}$$

Hg—gradient height; $Hg = 7$ m.

a—l slope of active gradient; $\alpha = 60^\circ$.

c—The safety distance between the upper stop of gradient and the axis of the first hole line; $c = 3$ m.

y—ore density; $y = 2\ \mathrm{t/m^3}$.

m—hole alignment coefficient; $m = 0.8 - 1.2$ ($-m = 1$).

$$D_{opt} = \frac{\left(7 \times \mathrm{cotg}\,60^\circ + 3\right)\sqrt{y}}{30\left(3 - 1\right)} = 0.1654 \approx 0.165\ \mathrm{m}\,,\quad D_{opt} = 165\ \mathrm{mm}\,.$$

The standard diameter is 171 mm, and I choose the best diameter.

$D_{opt} = 171$ mm, equivalent to the number in the field.

In CBG, considering the physical and mechanical properties of ores in different deposits (3 - 6), the drilling method is rotary, which is discharged by blowing compressed air.

At present, there are two rigs on the track (DM45E et DM30E) in sangaredi mine, and the drill pipe length is equal to 11 meters.

Determine the thrust required for drilling: This thrust depends on (3) three conditions:

- Fixed conditions of drill bit;
- Rock drilling conditions;
- Rock ultimate thrust condition.

Based on this condition, the thrust is determined using the following formula:

$$Pat = pat \times DT;\ \mathrm{N/cm}$$

Dt—drill bit with diameter of cm;

pat—specific axial thrust;

for cutting bit, this specific axial thrust change is

(1000 - 2000) N/cm.

$Pat = (1000 - 2000) \times 17.1 = (17100 - 34200)$

$Pat = (17.1 - 34.2)$ KN.

According to drilling conditions: Based on this condition, the axial thrust is determined using the following formula:

$$Pat = (0.6 - 0.7) \times f \times Dt \times 1000;\ \text{N}$$

f—The hardness of the rock; $f = 3$ - 6, $f = 6$.

From where: $Pat = (0.6 - 0.7) \times 6 \times 17.1 \times 10^3$; N

$Pat = (61.5 - 71.8) \times 10^3$; N

$Pat = (61.5 - 71.8)$; KN

According to the pause time of the detector:

This thrust is achieved according to the technical characteristics of the detector.

For drilling machines DM45E: $Pas = 227$ kn.

For drilling machines T4BH: $Pas = 168$ kn.

We emphasize that in order to make the detector operate normally, it is necessary to: $Pas > Pat$. For drilling machines DM45E.

$Pat < Pat$; 34.2 KN < 71.8 KN

$Pat < Pat$; 227 KN > 34.2 KN

Therefore, drilling is unreasonable because the conditions provided allow drilling with maximum thrust and it is the tools that limit the drilling efficiency. The detector is rational at Sangaredi mine, the borehole diameter is 171 mm, which is consistent with that of the nitro detector.

Bl_2 is 50 m long the second block of the depth trench access 4 to 7 m.

4.3. Technical Parameters

4.3.1. Drilling Rig Operation Calculation

Excavation speed

The excavation speed of a trench depends on the profile and efficiency of the device for excavating the trench. His statement is as follows:

$$V_C = \frac{Q_P}{Sta};\ \text{m}^3/\text{p}$$

where: Q_P is the production of shovels

Then we calculate the yield of the spade hit—PC 3000 (**Table 3**).

4.3.2. Theoretical Determination of Drilling Rig Number

This clause adopts the following formula:

$$Ns = \frac{Lf/year \times Kr}{Q_{exp} \times Np/year}$$

with: *Lf*/*year*—length of boreholes per year; M.

Kr—reserve coefficient; $Kr = 1.2 \times 1.25$; $Kr = 1.25$.

Q_{exp}—Operating efficiency m/p.

Np/*year*—number of posts per year:

DM45E Type drilling rig:

We have: $N_{s(\text{DM45E})} = \dfrac{132187 \times 1.25}{144.64 \times 288} = 3.96 \approx 4$

Four DM45E drilling can ensure the planned production of 20,132,000 tons.

At present, CBG has used two [(2)] DM45E drilling in the field.

Table 3. Performance of different types of rigs.

Type drilling rig	Theorical Performance (Q_{th}) m³/p	Technical performance (Q_{tech}) m³/p	Operating efficiency (Q_{ex}) m³/p
DM45E	129,600	253.69	152.21
DM30E	1680	241.08	192.96

The technical efficiency of CBG (2) drilling is as follows:

It is determined by the following formula:

$$Q_{tch} = Q_{tch}\ (\text{DM45E}) \times 3 = 253.69 \times 3 = 761.07\ \text{m/p}$$

Determination of the theoretical annual production of the two drilling rigs.

It is determined by the following formula:

$$Rs/year\ (\text{DM45E}) = Q_{tch} \times Ku \times Np/j \times Nj/an \times V \times Y \times 1/n;\ \text{t/an}$$

Q_{tch}—The technical efficiency of CBG (2) drillings.

Ku—utilization coefficient $Ku = (0.5 - 0.8)$; $Ku = 0.6$.

Np/day—number of posts per day; $Np/d = 2$.

$Nd/$—days per year; $Nd/year = 288$.

V—One meter borehole efficiency; m³/m; $V = 58.8$ m3/m.

Y—ore density; 2 t/m.

N—drilling irregularity coefficient; $n = 1.1$.

Therefore:

$RS/$(DM45E) = 761 × 0.6 × 2 × 288 × 58.8 × 2 × 1 × 1.1 = 22,666,355.7 t/year.

$RS/$(DM45E) = 22,666,355.7 ≈ **22,666,356 t/year**.

Let's compare this output with the planned output

$Rs/an > Rp$

Rplanned annual production 20,132,000 tons/year

The results show that, according to the theoretical study of planned production, the three drilling can guarantee.

4.3.3. Verification of Firing Parameters

Linear capacity, $P = \frac{3.14(dtr)^2}{4}\Delta_{ex}$; **Kg/m, ANFO Δ_{ex} = 900 Kg/m**, P **= 21 Kg/m.**

Quantity of explosives per hole:

$Q_{ex/tr} = P \times Lch_{max}$; **Kg/hole;** $Q_{ex.tr} = 21 \times 5 = 105$ **Kg/hole**

Stemming = 7 − 5= 2 m.

Drilled Burden (*B*) and spacing (*S*) ("according to Pascal montaux EPC group, nitro Bickford"):

Borehole diameter = 40 × drilling diameter.

Information regarding drilling and blasting parameter as collected from EPC group Nitro Bickford for optimization is shown in Table 4. As presented (Table 4) the blasting pattern adopted at the Sangaredi mine has drilling diameter is 171 mm.

Table 4. Calculation table for a blasting pull bench.

Diameter	56 mm	76 mm	89 mm	102 mm	127 mm	165 mm	171 mm
Burden (m)	2	3.0	3.6	4	5	6.6	6.84

Where spacing is: $B < S < 1.3 \times B$ => $6 < S < 1.3 \times 6$ => $6 < S < 7.8$; I take $E = 7$ m

So the mesh is: $(6 \times 7)\ m^2$

Quantity of explosives per year: $Q_{ex/year} = N_{tir/year} \times Q_{ex/tir}$; kg/year

$Q_{ex/tir} = 84 \times 105 = 8820$ kg/tir

$Q_{ex/year} = 361 \times 8820 = 3{,}184{,}020$ kg/year < 4,799,022 kg/year

This table shows the results obtained when the design parameters were optimized with hole diameter and hole depth. The optimized results show that when drilling to a depth of 7 m with 171 mm hole diameter, the burden and spacing are 6 m and 7 m respectively while the weight of charge is 105 kg.

The results of the calculation in the Table 5 shows that:

- Due to the quality of use, the vertical firing mode must be respected.
- Respecting the amount of explosives in each hole can create good fragmentation.
- Respect the depth of the hole to limit the number of large blocks.
- According to the calculation, the annual explosion volume is 3184020 kg/year, which is a reasonable value for CBG.
- The box is the result of my calculations, which is reasonable considering the hardness of the rock.

Compliance with these analyses will bring CBG to planned production.

4.4. Measurement of Blasting Vibration

In blasting operations, emulsion (Emulite in wet blastholes) and Anfo (blasting agent) gelatine dynamite (priming) and delay detonators (in the activity contractor) were used as explosives during the work, it can be seen that blast holes were vertical with a 171-mm diameter for all blasts. The hole length is 7 m, with 2 m of stemming for all blast patterns. A non-electric millisecond delay system was used to initiate the charge. The timing pattern was designed as 42-ms delay between rows and 17-ms delay between holes within a row. An inner hole detonator was used at 25-ms intervals.

Within the scope of the current research project, ground vibrations induced by blasting were measured to estimate the damage risk and site-specific attenuation. While measured distances were recorded for the shots, the ground vibration components were measured by using Instantly Minimate plus Model and White Mini-Seis Model vibration instruments, whose specifications are summarized below.

Although two different types of vibration monitors were used during the study, they have similar specifications and their particle velocity measurement ranges are accepted by the commission of the International Society of Explosives

Table 5. Summary of firing parameters.

Parameters	Units	Values	
		Calculated	Used
Hole depth	M	7	4 - 9
Linear capacity	Kg/m	21	-
Load length	M	5	3 - 6
Quantity of explosive per holes	Kg/hole	105	76 - 150
The length of fluff	M	2	2.3 - 3.8
The drill mesh	m^2	6 × 7	6 × 7
Drilling diameter efficiency	m^3/m	58	-
Total length drilled per year	m/year	178,956	217,858
The total length of the shooting hole	m/tir	496	50 - 12
Holes per row	-	14	10 - 72
Volume of "explosive block"	m^3/tir	26,498	20,975
shot block width	M	37.85	25 - 55
The number of rows per shot	-	6	3 - 6
Holes per shot	-	84	50 - 284
Quantity of explosive per shot	Kg/tir	8820	900 - 13,700
Quantity of explosive per year	Kg/year	3,184,020	4,799,022
Annual relay consumption	-	3610	-
Consumption priming per year	-	30,324	-
Annual use of detonator capsules	-	722	-
Annual production	Tone	22,666,356	20,132,000

Engineers (ISEE). Both the models are portable seismographs for monitoring and recording seismic and sound signals produced from blasting. They mainly consist of three geophones (transversal, vertical and longitudinal), a microphone, a control and memory unit and a battery. They use microcomputer technology. They also have their own seismographic data analysis software, which provides the easiest way to access and analyses recorded data. They can be used for a single shot or in continuous mode.

The instruments record peak values of particle velocity of up to 25 cm/s in three directions and time-histories of seismic vibrations and sound pressures. The instruments calculate the peak particle velocity, zero crossing frequency (ZC freq), peak acceleration and peak displacement for each of the transverse, vertical and longitudinal axes. Compliance reports, which indicate these values, can also be examined. Blasting geometry applied in the mine and the charging process was designed by blasters from the company, and the vibration measurements were applied to this blasting geometry spontaneously by the research team. In other words, the necessary information, such as quantitative measurements and observations, were the only data obtained from the blast shots, which would be the basis of the monitoring.

Figure 6 tells us about all the activities that take place at the time of mining in the Sangaredi mine namely Drilled Burden (B) which generally varies between 6 and 7 m, Drilled Spacing (D) which varies between 5 and 6 m, bench height (BH) and Hole length (HL) equal to 7 m, Hole diameter (D) 171 mm and Crest Burden.

4.4.1. Bench Blasting

The main purpose of blasting is to obtain the fragments so that they can go through the effective treatment of the rock after blasting, such as loading and crushing. Several parameters influence the results of the explosion; of which the mechanical properties of the rock mass, the geometry of the blast hole, the type and quantity of explosives, the mode of initiation and the delay time are the key factors in the blast design. Figure 7 shows a brief terminology of the geometry of staged blasting.

The very first condition of any design of the blasting operation is to obtain the

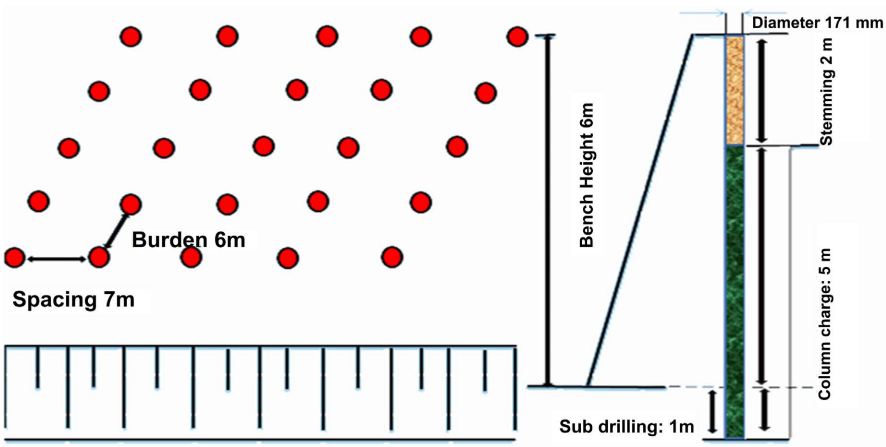

Figure 6. Design parameters of bench blasts for Sangaredi mine.

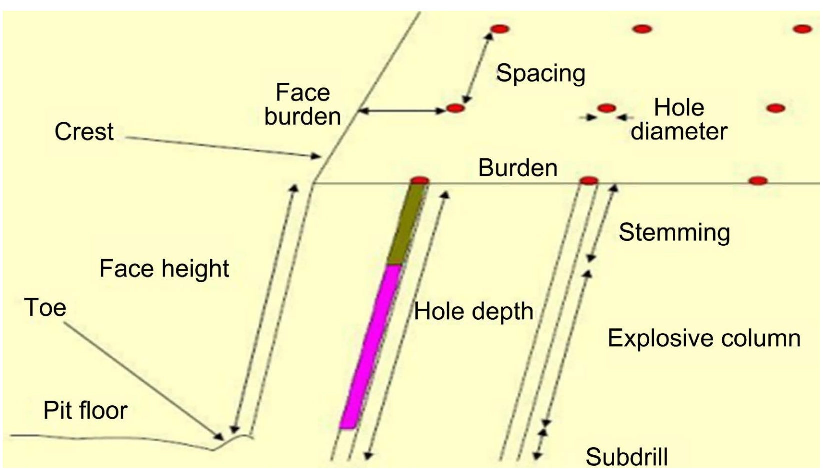

Figure 7. Bench blasting.

best results under the best existing operating conditions, with sufficient flexibility and relatively simple operation. The factors influencing the reaction of the explosion can generally be divided into two categories which are uncontrollable and controllable factors. The design of controllable factor blasting should aim to provide sufficient debris and ensure loading, transportation and subsequent processing at the lowest cost (Ataei & Osanloo, 2000).

In vibration prediction, although many empirical relationships have been established and used by various researchers in the past, the most reliable relationships are those which take distance and particle velocity as the basis. Scaling distance is a concept put forward using the amount of explosive energy in air shock waves and seismic waves, and this affects the distance basis. The distances between the firing points and the control stations are determined using measuring equipment and a GPS.

The scaling of the scale is derived by combining the distance between the source and the measurement points, and the maximum load per delay.

The equation used for the distance scaling is given below:

$$SD\,1/4\;R = Wd^{0.5}$$

where SD scaled the distance; R is the distance between the plane and the station (m); and Wd is the maximum load per delay (kg). On the other hand, the formula given below, which is proposed in most of the surveys, is used as a predictor to estimate the maximum particle velocity (PPV).

$$PPV\,(\text{mm/s}) = K \times (SD)^{-\beta}\;PPV\,(\text{mm/s}) = K \times (SD)^{-\beta}$$

K is the ground transmission coefficient and the specific geological constant.

A minimum of 30 data pairs is required for reliable analysis. To ensure the reliability of Equation; the attenuation formula should be statistically adjusted to a 95% confidence level and a "goodness of fit" or coefficient of determination (r) of the data should not be less than 0.7. The standard deviation, used to establish the confidence level, should be as close as possible to zero. When the goodness of fit is too low, below 0.7 or more, this is an indication that there is a problem or inconsistency in the data. When this occurs, a review of the data and testing procedures are recommended and a series of additional tests should be performed (source CBG Planning).

4.4.2. PPV Peak Particle Velocity

In this part despite that the holes have the same depth and same quantity of explosive but the different reaction. That depends on many factors namely: the density, the hardness, the physical mechanics of the rock, the amount of explosive and the presence of cracks. The graph of the relationship between particle velocity components and scaled distance is presented in **Figure 8**.

This case study in this operation has proven that it can be possible to design reliable blasts using this formula in Sangaredi mine. Therefore, to predict peak particle velocity in control blast design, the obtained equation, given above, was used for this mine.

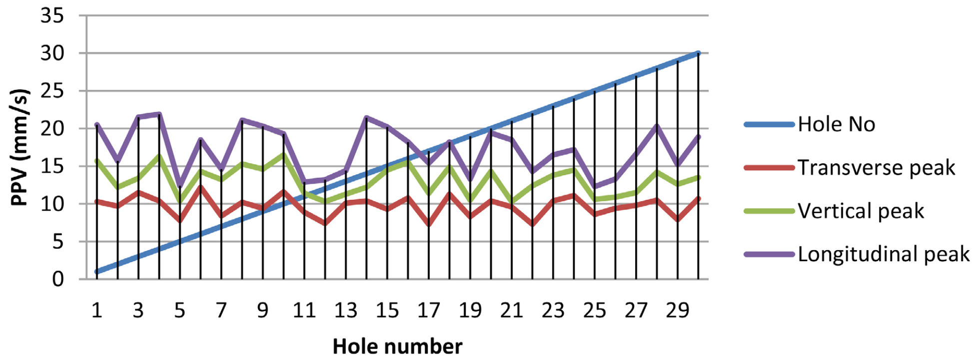

Figure 8. Particle velocity components versus scaled distance.

When studying this graph, it can be determined that 7% of frequency values are between 1 - 5 Hz, 80% are between 5 and 10 Hz, 10% are between 10 and 15 Hz and 3% are greater than 30 Hz. These results prove that measured event frequencies at this site are quite low because of the layer structure of the encountered rock masses, as was expected in accordance with the literature (Caccetta & Hill, 2016).

It is well known that low-frequency vibrations have a greater potential for damage than high-frequency vibrations (for a certain velocity). It is obvious that the damage risk can be very high because of the measured low frequencies and because building self-structural frequencies change between 5 and 10 Hz in general. These values show that it is important for the blasters and the mine authorities to record and evaluate all blast.

4.5. Result of Analysis and Optimization at the Sangaredi Mine (CBG)

In this article, there is a wide variation in annual production, approximately more than 2 million tons (20,132,000 to 22,666,356) tons per year, taking into account two factors, namely:

Parameters: drilling length, number of holes in each row, volume of rock blasted, degree of rock fragmentation.

For the time factor: number of blasting days per year, number of blasting operations per year, different blasting times between holes in the same row and row.

With all its requirements fulfilled, well-fragmented rocks have been observed to fill the excavator bucket well to provide fast and efficient production transport for the transport machine. For drilling grids, it depends on the mechanical and physical properties of the rock and the depth of the drilling. The mesh of this new plan is (7 × 6) square meters; the holes are 171 mm in diameter, the number of rows is 6, the number of holes per row is 14, and the number of holes per shot is (6 × 14) = 84 per shot hole. The quantity of explosives per hole is 105 kg and the annual production is 3,184,020 kg, which is strictly lower than the planned 4,799,022 kg (See the **Table 6**).

Table 6. Annual production of ore and quantity of explosive.

RESULT	Annual production	Quantity of explosive
Before	20,132,000 Tons	4,799,022 kg
After	22,666,356 Tons	3,184,020 kg

5. Conclusion

At the end of long studies and scientific analyzes materialized by practical experiences in accordance with the mining requirements proposed by CBG emphasized on the reduction of the nuisance phenomena due to the practice of blasting by explosives, which is mainly caused by vibrations, noises and environmental pollution: this required a study of the available vibratory data, and the proposal of methods to limit these vibrations. This task could not be solved without having to do with blasting optimization in general. This momentum has essentially made it possible to get to know and become aware of the reality of blasting practices, and in particular of the gap that there may be between theory and practice. Facing the subject towards analysis has been a positive thing: work on vibration reduction under the present circumstances of practice could not have been successful, due to too many uncertainties. The tasks that now need to be worked on are based on mastering each step of the shot, setting out, drilling, loading. By means of suitable tools, it is thus possible to achieve control of the overall blasting. It now appears essential to put in place procedures in order to work properly. This must be based on regular and detailed monitoring of blasting practices, then on the implementation of certain new elements and the correction of the uncertainties detected, with a view to continuous improvement in the control of slaughter.

From the material point of view, this requires moderate financial investments, for the tools for measuring hole deviations and lifting the front. The human level is also concerned: the driller and the miner are involved in their daily work. On the other hand, analyzing and improving shots requires additional human investment, in the order of half an engineer time. Its mission will consist of a round trip between theoretical and practical aspects, with on the one hand the drafting and installation of information collection procedures, on the other hand, the analysis of this information then the implementation. Corrective or new elements, and finally their follow-up. The control of this continuous blasting improvement procedure may be based on indicators such as production data from the primary station and slaughter results data. We can take a measurement to improve punctually, shot by shot, or more generally and especially with details.

For the implementation of concrete and appreciable work, this approach will lead us on the one hand to a better operation and a better result of the fragmentation of the rocks. On the other hand, a faithful and precise application of a shooting pattern constitutes the starting point for other studies, with different challenges. The resolution of optimization of the blasting effects depends on a good knowledge and a good control of the causes: by knowing and by controlling the realities on the ground, it will be possible then to envisage interesting studies at different levels, economic, operational, environmental, social, regulatory, reduction of vibrations for example, or the optimization of the overall production chain of ore extraction. Finally, it is indispensable and necessary even to understand and realize that blasting is not simply about placing and detonating explosives in a rock mass. It is an operation that must benefit from a more refined and clear approach; blasting must first of all be systematically adapted to the local context of exploitation and shooting.

Finally, it must be constantly questioned in relation to technical and technological advances, to be part of a process of permanent progress in order to avoid incidents and accidents during and after operations.

Conflicts of Interest

The authors declare no conflicts of interest regarding the publication of this paper.

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