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EXPERIMENTATION OF A BLASTING THEORY FOR THE DIGGING OF AN UNDERGROUND HORIZONTAL EXCAVATION FROM BOUKHADRA IRON MINE (EASTERN ALGERIA)

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Abstract: The digging of horizontal mining excavations where the rock is hard requires an appropriate blasting plan. Thus the recognition of all the parameters that affect the latter including section, advancement and stability of the structure. to do this, a test of a theory at the level of the iron mine of Boukhadra, based on the assurance of creation for another liberation surface inside the front to be cut down; in order to improve the progress of the construction site due to the reduction of the specific explosive consumption, also in order to avoid the back effect after blasting which negatively influences the desired section shape.
Keywords: Algeria, mining, civil engineering, tunnels, rock fragmentation, blasting, demolition, underground excavation

INTRODUCTION

The main technological process in underground ore body development is the blasting process. During development and sloping processes, drilling and blasting operations take up 50-70% of all the process and volume. The quality and the cost of development depend on the effectiveness of the drilling and blasting operation. The duration and laboriousness of drilling-blasting operations mainly depends on the physical and the mechanical properties of the blasted rock, the cross-section of the development, the blasting parameters and the blast hole and hole charge construction (Raskildinov B., U., 2001).

Good blast design and execution are essential ingredients for successful underground mining. Poor blasting practices can have a severely negative impact on the economics of mining. Military blasting, rather than precision blasting, can result in dilution of high-grade ores. Military blasting can damage sensitive or tender rock structures that make up the hanging wall or footwall so unwanted caving occurs with the possibility/probability of ore loss and/or dilution. Poor design and execution when ring drilling can mean that succeeding rings are damaged ailure to complete undercuts can mean the transmission of very high loads to the underlying structures and their subsequent failure. The results are lost revenues and added costs.

Good drilling and good blasting go hand in hand. If the drilling is poor, there is little one can do to correct the rest of the job. It is similar to building a house. If the foundation is poorly done, there are major problems along the rest of the way. A discussion of drilling practices is beyond the scope of this chapter, but the blast design must begin there. Prior to designing the blasting, one must be sure that the miners have the machines and the capabilities to build the design. If the design cannot be built with the tools at hand, then it is no design. Hence, one starts the design process by carefully examining the drilling capabilities (Roger H., William H., et Claude C., 2014).

Drilling and blasting method (DBM) is globally used for rock excavation due to low investment, cheap explosive energy, easy acceptability among the stakeholders, possibility of dealing with different shapes and sizes of openings and reasonably faster rate of advancement in a suitable geotechnical mining condition. This makes DBM a preferred method of rock excavation (Innaurato N., Mancini R. et Cardu, M., 1998). The drill and blast method is characterized by operations that occur in a repeated cyclic sequence. The level of automation and mechanization of these tasks is low and there is a high degree of hard manual labor involved. During temporary support installation and mucking, worker safety is a serious issue. This is because immediately after blasting there can be a high degree of risk from rock falls in the unsupported section of tunnel. The drill and blast excavation method is a very adaptable and flexible process in regard to the excavation of any tunnel cross section or intermediate section, and it allows for the installation of various kinds of temporary rock support. Further, the drill and blast method is characterized by a short mobilization time requirement due to the use of standard equipment (Gerhard G., Cliff S., et Asce, F., 2002).

Duvall and (Duvall, W.I., et Fogelson, D.E., 1962); Langefors U., et Khilstrom, B., 1973) and others, have published blast damage criteria for buildings and other surface structures. Almost all of these criteria relate blast damage to peak particle velocity resulting from the dynamic stresses induced by the explosion. While it is generally recognized that gas pressure assists in the rock fragmentation process, there has been little attempt to quantify this damage. The blasting sequence in a tunnel or drift always starts from the "cut", a pattern of holes at or close to the center of the face, designed to provide the ideal line of deformation. The placement, arrangement and drilling accuracy of the cut is crucial for successful blasting in tunneling. A wide variety of cut types have been used in mining and construction, but basically they fall into two categories: cuts based on parallel holes, and cuts that use holes drilled at certain angles.

The most common types of cut today is the parallel and V cut. The V cut is the older of the two and is still widely used in construction. It is an effective type of cut for tunnels with a fairly large cross-section and requires fewer holes than a parallel cut. The parallel cut was introduced when the first mechanized drilling machines came on the market making accurate parallel drilling possible.

When developing an underground mining project (Babyouk G., et Chaib R, 1988), one of the main operations is the establishment of a favorable blasting plan for the digging of the underground excavation. This is normally determined by a more rational method.

This paper is intended to ensure the creation of another liberation surface during digging of horizontal excavations, by the exact arrangement of the empty holes and that of cut from a mathematical theory based on the coefficient of expansion of the rock, in order to optimize the blasting process by improving the progress of the construction site due to the reduction of the specific explosive consumption.

MATERIALS AND METHOD

— Theoretical Calculation

» Elastic Models of Explosive–Rock

Detonation of an explosive charge in rock results in dynamic loading of the walls of the borehole and generation of a stress wave, which transmits energy through the surrounding (Brady B. H. G., Brown E.T., 2005). The generation of fractures can be assumed to be related in some way to the magnitudes of the transient stresses associated with the passage of the wave.

The most important solution for the stress distribution around an explosive source is that due to Sharpe (1942) for a spherical charge. The charge is detonated in a spherical cavity of radius a, generating a spherically divergent P wave.

The transient cavity pressure is taken to be represented by the expression (1):

$$p = p_0 e^{-at} \tag{1}$$

where; p_0 is the peak wall pressure, and α is decay constant.

» Phenomenology of Rock Breakage by Explosive

Explosive attack on rock is an extremely violent process, and experimental attempts to define the mechanics of rock breakage by explosives have not been highly successful. The following qualitative account of explosive interaction with rock is based mainly on the accounts by Duvall and Atchison (1957), and Kutter and Fairhurst (1971). In the period during and following the passage of a detonation wave along an explosive charge, the rock around the blast hole is subjected to the following phases of loading:

- dynamic loading, during detonation of the explosive charge, and generation and propagation of the body wave in the medium;
- quasi-static loading, under the residual blast area pressure applied by the detonation product gases;
- release of loading, during the period of rock displacement and relaxation of the transient stress field.

» Dynamic Load

Three zones of material response to the impulsive loading and high-intensity stress wave are recognized in the medium. In the immediate vicinity of the blast hole, the high stress intensity results in the generation of a shock wave in the rock. In this so-called shock zone, the rock behaves mechanically as a viscous solid.

Passage of the stress wave causes the rock to be crushed or extensively cracked, and the intensity of the wave is reduced by viscous losses. The attenuation process also results in reduction of the wave propagation velocity to the acoustic velocity. For a blast hole of radius r_h, the radius r_s of the shock zone may be about $2r_h$. In some cases, superficial observation may not, in fact, reveal a crushed zone around a blast hole.

The domain immediately outside the shock zone is called the transition zone. In this region, the rock behaves as a non-linear elastic solid; subject to large strain (i.e. the small strain elastic theory developed in this text is inadequate to describe rock behavior in this zone). New fractures are initiated and propagated in the radially compressive stress field, by wave interaction with the crack population. Crack development is in the radial direction, resulting in a severely cracked annulus, called the rose of cracks. Generation of the radial cracks extracts energy from the radial P wave, resulting in reduction in the stress intensity. The radius, r_t, of the transition zone is about $4–6r_h$.

In the transition zone, the intensity of the state of stress associated with the radial wave is reduced to a level at which the rock behaves linearly elastically. The behavior of the rock in this domain, called the seismic zone, although new cracks may be initiated in this region, crack propagation occurs exclusively by extension of the longest cracks of the transition zone. Thus, a short distance outside the transition zone, a few cracks continue to propagate, at a velocity of about $0.20–0.25C_p$. The P wave therefore rapidly outruns the crack tips, and propagation ceases. It appears that at a radius of about $9r_h$, macroscopic crack generation by the primary radial wave ceases.

However, during transmission of the wave towards the free face, fractures may be initiated at the Griffith cracks. The process may be considered as one of conditioning the rock mass for subsequent macroscopic rupture, or of an accumulation of damage in the rock fabric.

When the radial compressive wave is reflected at a free face, a tensile wave, whose apparent source is the mirror image of the blast hole reflected in the free face, is generated. It is possible that, in massive rock, surface will occur at the free face, although there is no convincing evidence for this.

In the interior of the medium, crack extension may be promoted by the more favorable mechanical environment of the tensile stress field, resulting in further accumulation of damage. In a jointed rock mass, the role of the reflected tensile pulse is limited, due to joint separation, trapping the wave near the free face.

» Quasi-Static Load

The dynamic phase of loading is complete when the radial wave propagates to the free face, is reflected, and propagates back past the plane of the blast hole. For an average rock mass (C_p = 4000 m s^{-1}), this process is complete within 0.5 ms/m of burden. Because mass motion of the burden does not occur for an elapsed time much greater than the dynamic load time, it appears that pressure exerted in the blast hole by the detonation product gases may exercise a significant role in rock fragmentation.

Sustained gas pressure in the blast hole increases the borehole diameter, and generates a static stress field around the blast hole. Gas may also stream into the fractures formed by dynamic loading, to cause fracture extension by pneumatic wedging.

In the following discussion, the effect of field stresses on the quasi-static stress distribution is neglected. The simplest case of quasi-static loading involves a pressurized hole, of expanded radius a, subject to internal pressure p_0. If the region around the hole boundary is not fissured, the state of stress at any interior point, of radius is given by equation (2):

$$\sigma_{rr} = p_0\, a^2/r^2 \,,\ \sigma_{\theta\theta} = -\,p_0 \frac{a^2}{r^2},\quad \sigma_{r\theta} = 0 \qquad (2)$$

Then, the hole boundary stresses are given by equation (3):

$$\sigma_{rr} = p_0 \quad , \quad \sigma_{\theta\theta} = -\,p_0 \qquad (3)$$

Thus, if the state of stress represented by equation. (3).is incapable of generating fractures at the hole boundary, that represented by equation (2) cannot generate fractures in the body of the medium. This suggests that the pattern of cracks produced during the dynamic phase may be important in providing centers from which crack propagation may continue under gas pressure.

Quasi-static loading may occur in the presence of radial cracks, with no gas penetration of the cracks. The presence of radial cracks means that no circumferential tensile stresses can be sustained in the cracked zone.

At any point within the cracked zone of radius r_c, the state of stress at any point r is defined by equation (4):

$$\sigma_{rr} = p_0\, a/r \,,\ \sigma_{\theta\theta} = 0 \qquad (4)$$

Moreover, at the perimeter of the cracked zone by:

$$\sigma_{rr} = p_0\, a/r_c \,,\ \ \sigma_{\theta\theta} = -\,p_0 a/r_c \qquad (5)$$

The implication of these last equations is that existing radial cracks around a hole may extend so long as the state of stress at the boundary of the cracked zone satisfies the macroscopic failure criterion for the medium.

A third possible case of quasi-static loading involves radial cracks, but with full gas penetration. If the volume of the cracks is negligible, the state of stress at the boundary of the cracked zone is given by equation (6):

$$\sigma_{rr} = p_0,\ \sigma_{\theta\theta} = -\,p_0 \qquad (6)$$

In practice, the degree of diffusion of gas into the fractures is likely to lie somewhere between the second and third cases, described by equations (5) and (6) In any event, the existence of circumferential tensile stresses about the blast hole provides a satisfactory environment for radial fracture propagation.

— Proposed Solution

the creation of another clearance surface is ensured by the determination of the expansion coefficient of the rock Ke, thus empty holes are drilled in the center of the section (or a single hole with a large diameter) and the volume of the empty created by that, the latter will occupy by the excess of expanded volume resulting after blasting of the section located between the cut holes. The arrangement of these that empty is parallel to this last.

The coefficient of expansion is determined by the ratio between the volume of the blasted rocks V_{ab} and that of the rocks in place V_{bb} (equation 7);

$$Ke = V_{ab}/V_{bb} \qquad (7)$$

Knowing that these volumes have the same length, so just do the calculations using the surfaces if all. So we have the equation (8):

$$S_{ab} = \frac{K_e}{S_{bb}} \qquad (8)$$

To determine the four cut holes around the first surface, we assume a square area value S_{bb} around the empty holes; then we check the condition $S_{eh} \geq S_{ab} - S_{bb}$. who says: the excess of expanded surface ($S_{ab} - S_P$) less than or equal to the surface of the voids generated by the uncharged holes by explosive S_{eh}.

Where the latter is determined by the calculation relation of the surface of the circle, and multiplied by the number of empty holes (equation 9):

$$S_{eh} = \pi\, D^2 \,/\, 4\ .N_{eh} \quad ; cm^2 \qquad (9)$$

where: D is the diameter of the hole (cm), N_{eh} is the number of empty holes.

Besides, to solve $S_{ab,}$ we can express it from the equation (8), so we get the equation 10:

$$S_{ab} = K_e.S_{bb} \quad ; cm^2 \qquad (10)$$

If the condition mentioned above is not verified (figure 1), we redo another supposition of the surface and then we check it until the optimum value is reached, the sides of which represent the cut holes. Therefore, the square root of the optimal surface S_{bb} gives the length of each side of square by the formula (11):

$$L = \sqrt{S_{bb}} \quad ; cm \tag{11}$$

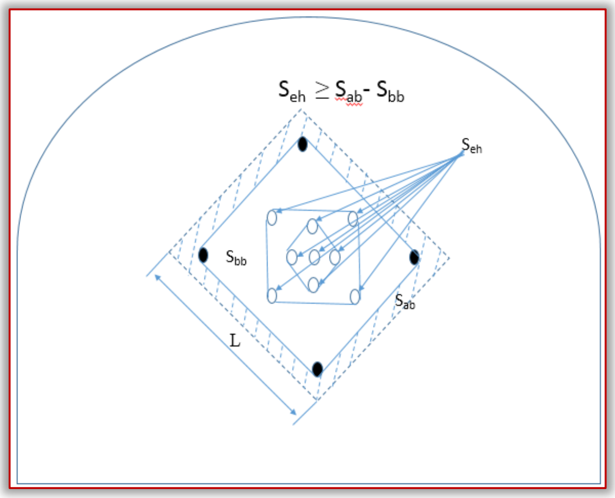

Figure 1: Schema of disposition of the empty holes and that of cut

To facilitate the work of the supposition buckle of the surface S_{bb}, it can be translated into a flowchart (figure 2) that can be used in a computer program, so it is as follows:

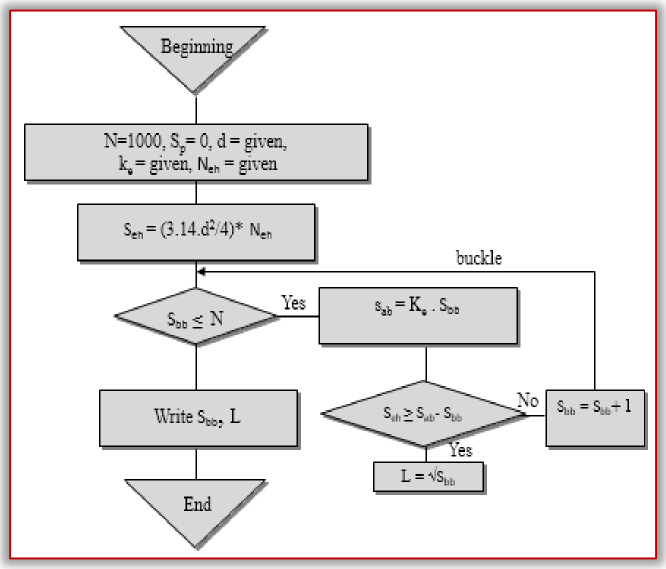

Figure 2: Flowchart of calculation a surface of cut holes

— Stop Hole

The holes surrounding the cut are called stop holes. The diameter of a stop hole is typically between 41 - 51mm. Holes smaller than 41mm may require drilling an excessive number of holes to ensure successful blasting. Holes bigger than 51mm can result in excessive charging and an uncontrolled blast.

Holes are placed around the cut section in an evenly distributed pattern using a space/burden ration of 1:1.1. If hole size is between 45 - 51mm, typical spacing and burden are both between 1.0m - 1.3m. Actual rock conditions and ability to drill in the required positions are factors that can reduce or add to the number of holes needed. The design of the drilling pattern can now be carried out and the cut located in the cross section in a suitable way.

— Contour Hole

Floor holes have approximately the same spacing as stop holes, but the burden is somewhat smaller; from 0,7m to 1,1m. Inaccurate or incorrect drilling and charging of the floor holes can leave not blasted bumps, which are difficult to remove later. The contour holes lie in the perimeter of the drilling pattern. In smooth blasting, contour holes are drilled closer to each other and are specially charged for smooth blasting purposes. Spacing is typically from 0.5m to 0-7m and burden varies between 1 and 1.25 times the space. This type of layout makes it possible to use special smooth blasting explosives, which limits the width and depth of the fracture zone in the walls and roof caused by blasting. In special circumstances, two or more smooth blasting rows can be used.

Rock hardness is occasionally incorrectly considered to be the only dominant factor when optimizing the drilling pattern. The change from very hard rock to soft rock therefore causes a change in the drilling pattern. Rocks that are hard but abrasive are easily blasted, whereas the blast ability of rocks such as some limestone, although relatively soft, is poor.

However, it is beneficial to redesign and optimize the drilling pattern long before this stage is reached and, more important still, to consider rock blast ability.

— EXPERIMENTATION: Geographical and Geological Location of the Experimental Site

We tried this theory at the iron mine of Boukhadra Tebessa Algeria in the main horizontal gallery that gives access to the iron deposit of which a section of 12 m^2, its shape is vaulted. The drill chariot used is MERCURY (1FP) with a diameter of hole 65 mm, the rock is a mineralized marl.

The Jebel Boukhadra (Gadri L., 2012) is located in the east of Algeria 45 km north of the wilaya of Tebessa, 47 km from the mine of Ouenza, 13 km from the Algerian-Tunisian borders, and 190 km south of the steel complex of El Hadjar (wilaya of Annaba). The Jebel of Boukhadra is an isolated massive that rises above the Morsott valley from 700 to 800 m of altitude. With a climax of 1463m. The deposit is between the meridians 8°- 01'and 8°-04' East and the parallels 35°- 40'et 35°- 50' North. The Boukhadra area consists of Mesozoic sediments and with a part of Tertiary and Quaternary for Mesozoic, formed by Triassic and Cretaceous sediments.

RESULTS AND DISCUSSION

The parameters of the blasting plan were calculated for a progression gallery (figure 3 and 4) using the above-mentioned theory, therefore the summary results were found in table 1. While the operation of loading the holes, the type of the explosive material used is the Marmanit I which has a detonation speed of 4000 m/s, a density of 0.95 and a diameter of cartridge of 50mm, the priming is in electric with a schema of the blasting order (figure 5) as follows:

- The four cut holes: instantaneous electric detonators N°0.
- The four stop holes: micro-delay electric detonators N°3.
- The eight contour holes: micro-delay, electric detonators N°5.

Table 1: Blasting Plan Parameters

Number of empty holes	09 holes
Number of cut holes	04 holes
Number of stopeholes	04 holes
Number of contour holes	08 holes
Number of total holes	25 holes
Depth of the hole	1.2 m
Number of holes loaded	16 holes
The amount of the explosive charge	24,50 kg
Volume of rocks blasted	12,96 m^3
Average explosive consumption per hole	1.53 g/m^3

Figure 3: Photo gallery of holes disposition

Figure 4: Blasting result in the gallery

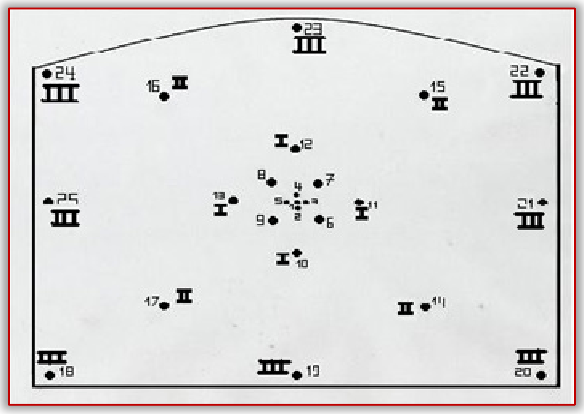

Figure 5: Schema of the blasting order

From the figure 4.we notice that the form of the gallery is well formed in comparison with that old, which is among the problems of the company for the guarded, the deformation of the form generates auxiliary work during a digging of the gallery which is the void filling created between the support and the wall of it, and this is mandatory to ensure the stability of the structure. So the void created by the holes not loaded with explosive material allowed us to ensure another liberation surface inside the massive to be fragmented, which in turn to avoid the fragmentation clamping of the rocks and also to avoid the back effect of blasting due to the deformation of the gallery form.

CONCLUSION

This theory of digging horizontal mining excavations based on the real disposition of the cut holes, suggests that there is a good blasting process running in the rock massive, a good advancement of the front of the gallery and also a good form of its section which consequently to a reduction of the specific consumption of explosives.

This theory did not take into account the voids of the discontinuities, cracks ... etc which are in the interval of the cut holes, because it is difficult to quantify their volume in the massive, in any case it is an advantage for the existence of these because it will increase the void more than the empty holes which in turn will also create another liberation surface.

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