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A THEORETICAL STUDY INTO DAMAGE TO PILLARS CAUSED BY BLASTING

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ABSTRACT: Surface subsidence when mining is dependent in the final analysis on the behaviour of the pillars supporting the overlying strata. There are different factors which influence pillar strength, and one of them is the damage caused to the pillars from cracks which are induced or propagated in the rock by blasting. There are several approaches used to estimate an extension of the damage zone, but most of them are based on empirical study being therefore of limited utility.

This paper presents an attempt to estimate the thickness of a damage zone to the pillar caused by blasting. It is based on a theoretical investigation of crack behaviour (crack initiation and propagation) in the stress field generated in a rock massif by blasting. The study considers the mechanism of rock failure, on one hand, and the stress state of rock induced by a blast, on the other. To predict the behaviour of a crack having a certain length and space orientation it is necessary to know the principal stresses and their change with time.

An engineering method of calculating the stress at any point in the rock is developed allowing to take into account the main parameters of the blast charge (type of explosive, blasthole diameter, charge mass, etc.) as well as of the rock properties (density, elasticity, strength, etc.). The method makes it possible to calculate the stress change with time, i.e. the blast wave shape. A certain relation between the wave duration and the length of a crack to be initiated is obtained.

The model facilities the determination of the distance from a charge, at which a crack of a particular length can be initiated and developed. Knowing this, one can estimate the thickness of the damage zone induced to the pillar by the blast having certain parameters, and/or work out the measures to reduce the zone. The thickness of the damage zone can be taken into account when designing pillar strength and stability.

1 INTRODUCTION

In underground excavation engineering, good blasting is just as important as the design of the correct support system, because the supporting structures are affected in the final analysis by the action of blasting. "The innocent rock mass is often blamed for insufficient stability which is actually the result of rough and careless blasting" [1]. Thus, blasting can cause a significant damage to the rock mass left behind. In general terms, the influence of blasting damage on the carrying capacity of a pillar can be expressed as follows [2].

$$\sigma_p / \sigma_o = (C/T)^{-D_B}; \qquad (1)$$

Where
σ_p = pillar rock strength,
σ_o = stength of intact rock,
C = pillar cross dimension,
T = thickness of a damage zone induced by blasting,
D_B = factor characterizing the blasting damage intensity.

Obviously, the thickness of a damage zone depends both on the type of blast and on the condition of the rock mass.

There exists a number of empirical formulae to estimate the thickness of the rock mass damaged by blasthole charges. An often-used method, which roves over several books, was developed by Holmberg and Persson [1]. It is related to the peak particle velocity induced by the blast.

A certain empirical equation was proposed for tunnel blasting conditions in competent Swedish bedrock.

The constants in this equation depend upon the type of blast and the conditions of the rock mass in which the blast is carried out. Ideally, these constants should be determined for each site by conducting a series of trials and monitoring the induced particle velocity at different distances from the charges. Otherwise, the calculation would be too approximate and rough.

In such a manner, empirical formulae do not allow one to perform preliminary calculations of an expected intensity of damage before introducing a particular blasting technology into practice.

This paper presents an attempt to work out a model of rock cracking induced by the blast based upon the fracture mechanics and the mathematical simulation of blast action on the rock mass left behind.

2 MECHANICS OF ROCK DAMAGE BY BLASTING

When a borehole charge is detonated, the high pressure gasses impact the walls of the borehole and generate an intensive stress wave which travels outwards into the rock. In the immediate vicinity of the borehole wall, the stress can exceed the strength of the rock and crushing of the rock can occur. Since the intensity of the stresses falls off rapidly with distance from the borehole, the rock crushing dicreases respectively.

The radius of the crushing zone ranges from 2 to 4 blasthole radii (Ro).

The physical meaning of the static toughness is the critical stress initiating a crack of a unit length, whereas that of the dynamic toughness is the critical pulse of a unit duration.

The dynamic toughness allows for an assessment of the interaction of a dynamic stress pulse and the rock under consideration.

To simplify the matter let us take a plane triangular pulse described by the equations:

$$\sigma = \sigma_R \left(1 - t/T\right); \qquad (11a)$$

$$\sigma_R = \sigma_o R^{-a}; \qquad (11b)$$

where: R = distance from the pulse source,

t = time,

σ_R = pulse amplitude at distance R from the source,

σ_o = pulse amplitude at the source,

T = pulse duration (const),

a = coefficient.

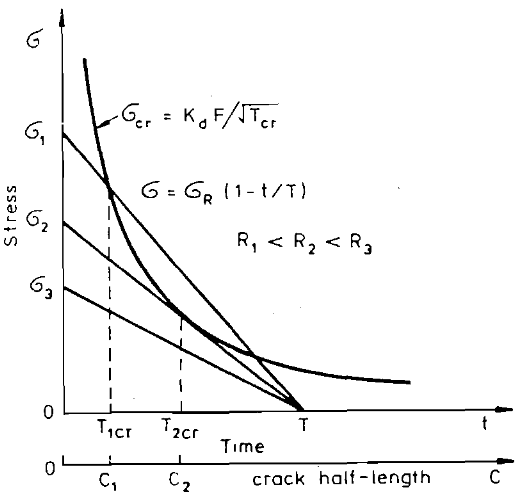

Figure 1. Interaction of a triangle pulse and a rock

Figure 1 gives a graphic interpretation of formulae (9) and (11). The straight lines represent the triangular pulse at the distances R_1, R_2 and R_3. the curve simulates the dependence of the critical stress upon the critical pulse duration. The first intersection of the curve with the pulse line indicates minimum pulse duration required to initiate the cracks of the half-length larger than C_1, to be determined from equation (8) as $C_1 = T_{cr}\, 2C_p/\pi$. As the pulse moves further and further away from the source, it attenuates, and thus, the critical duration required to react with the cracks increases. It means that the minimum limit of the length of the cracks reacting with the pulse increases respectively, up to the distance behind which the pulse can not initiate any crack.

In the figure that is the second pulse at distance R_2, which should be taken as the maximum distance of the rock damage by the pulse in question.

In such a manner, an analysis of any pair "rock-pulse" can be fulfilled. The only thing to be known, besides the rock properties, is the function describing the pulse variation with distance and time.

4 BLASTING MODEL

The stress state behind the front of a blast wave is characterised by two stresses: radial stress, σ_r, and tangential stress, σ_t, which can be considered as principal stresses, i.e. $\sigma_r = \sigma_1$ and $\sigma_3 = \sigma_t$. They are related to each other as follows [10]:

$$\sigma_\tau = -\frac{\nu}{1-\nu}; \qquad (12)$$

where ν = Poisson's ratio.

In general terms, the radial stress induced by blasting in a competent rock is described by the expression:

$$\sigma_r(R,t) = \sigma_o f(R)\,\varphi(t); \qquad (13)$$

where: σ_o = peak stress at the borehole wall,

$f(R)$ = dimensionless function, describing the amplitude attenuation with distance R from the borehole,

$\varphi(t)$ = dimensionless function describing the pulse shape.

As is shown in [9] the amplitude variation with distance can be represented in the form:

$$f(R) = (R/R_o)^{-n} \exp[-\alpha(R - R_o)]; \qquad (14)$$

where: R_o = borehole radius,

n = wave divergence factor,

α = attenuation coefficient.

The divergence factor is a complex function of the following factors: charge radius, R_o; charge length, ℓ_{ch}; accoustic properties of rock (i.e., sound velocity, C_p); rock density, ρ; distance from the charge, R:

$$n = n(R_o, \ell_{ch}, \rho, C_p, R). \qquad (15)$$

6 DISCUSSION

The model presented in the paper is, to a certain extent, an idealization of the process, even though it facilitates an analysis of the dependence of the damage zone extension upon a set of the main factors influencing the process, such as properties of a particular rock, explosive parameters (density, detonation velocity, strength), charge mass and geometry (diameter, length, air gaps).

It should be particularly empasized that an investigation of the dependences and tendencies can be performed without time consuming and costly in-situ measurements.

What is more, the model can be combined with a model of the stress state of the rock created by a mining process, by super positioning the principal stresses induced by blasting with those caused by the geomechanical situation at the place of interest.

The model considers an intact competent rock without fissuring which separates the rock into blocks. This corresponds to the situation when rock blocks are large enough in comparrison to the blasting scale, i. e. in a case of shotholes 40mm dia, the distance and dimensions under consideration are around 0.5 to 1.0m. If a rock is cracked very intensively (i.e. 10 joints per meter long), this model is not applicable.

7 CONCLUSIONS

In considering rock damage induced by blasting, static failure criteria by themselves are not sufficient. A dynamic toughness and a further criterion of dynamic failure are introduced to satisfy the conditions. The dynamic failure criterion allows the determination of the dimension of the rock imperfections reacting with a particular blasting pulse as well as the extension of a damage zone in the intact rock. A mathematic model of blasting pulse is proposed allowing for a prompt analysis of the influence of different blasting parameters on the intensity of rock damage. This enables one to work out measures to minimize the damage induced by blasting on pillars.

REFERENCES

1. Holmberg, R and Persson, P.-A. 1990. Design of tunnel perimeter blasthole patterns to prevent rock damage Trans. Inst. Min. Metall, London, vol 89, pp A 37-40.
2. Borsch - Komponiets, V.I. and Makarov, A. B. 1986. Rock pressure when mining thick inclined ore bodies. Pull "Nedra," Moscow.
3. Hoek, E and Brown, E. T. 1980. Underground excavations in rock. Inst. Min. Metall., London.
4. Barron, K 1974. Brittle fracture initiation in and ultimate failure of rocks. Int. J. Rock Mech. Min. Sci. Vol 8, pp 541-575.
5. Hellan, K 1984. Introduction to the mechnics of failure. McGraw Hill.
6. Gunsallus, K L and Kulhawy, F H 1984. A comparative evaluation of rock strength measures. Int. J. Rock. Mech. Min. Sci. Vol 21, No. 5, pp. 233-248.
7. Boukharov, G N 1990. Mechanics of failure of crystaline structures. Pull M Geol. Pros. Inst., Moscow.
8. Steverding B., Lehnigk S. H. 1976. The fracture penetration depth of stress pulses. Int. J. Rock Min. Sci., Vol 13, pp 75-80.
9. Boukharov, G N and Chanda, M W 1988. General investigations of rock blasting for gemstone deposits. Zamb. J. Appl. Earth Sci., Vol 2, No. 2, pp. 55-70.
10. Boukharov, G N 1990. Mathematic modelling of mining technology at and optic mineral deposits. MGPI, Moscow (a research report).
11. Mosinets, V N and Abramov, A V 1974. Blasting in fissured rocks. Publ. Nedra, Moscow.