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DEVELOPMENT OF BLASTING DESIGNS FOR UNDERGROUND MINING IN THE KAULDY MINE OF ALMALYK MINING AND METALLURGICAL COMPANY

The implemented analysis of the perimeter control blast designs in the Kauldy Mine of Almalyk MMC shows that the method efficiency depends on the correct selection of a spacing between perimeter blastholes, ratio of the blasthole spacing to the burden relative to the buffer blast row and a volume ratio of explosive charge and blasthole.

The experimental procedure of the perimeter control blasting design in underground mining is developed. The procedure made is possible to estimate the influence exerted on the perimeter control blasting efficiency by: shattering force and strength of explosives, spacing of the perimeter blastholes, closeness factor of the blastholes, charge design, incline angles of the perimeter blastholes and geological structure of the ambient environment.

An 'effective stemming' technique is proposed as blasting of an additional shortened blasthole to make a face cut in treated rock mass. The varying density stemming generated by blasting an additional inclined and shortened blasthole increases burning of blastholes by 2 times.

From the hydrodynamic theory of cumulation of explosive effect, the depth of breaking in rock mass is determined as function of the cumulative jet length and density, density of rocks as well as compressibility of rocks and the jet material. The studies show that efficient rock fragmentation by blasting with cumulation of explosive effect is achieved by means of changing the angle of implosion of cumulative liner, dependent on the initial/final velocity ratio of the cumulative jet, the time of the jet action on rocks, as well as the height and thickness of the cumulative liner. Adjustment of the implosion angle can reduce the yield of oversize by 1.2 times.

The method of rock breaking by blasthole charges with cumulation of explosive effect is developed and tested on a full scale in underground mining. The method allows enlarging the blasting pattern effect owing to the complete use of explosion energy, increasing the utilization factor of blastholes, decreasing the powder factor and, thereby, cutting down the expenses connected with drilling and blasting in underground mining by 20%.

The perimeter control blasting design is developed for underground mining, and a set of interrelated parameters which govern the blasting performance at the perimeter of mine roadways is determined, including the spacing of blastholes, the charging ratio, the closeness ratio and the delay of electric detonators in the perimeter blastholes.

Keywords: *underground mining, blasting, blasthole, mine roadways, perimeter control blasting, method of rock breaking by blasthole charges with cumulation of explosive effect, perimeter control blasting design in underground mining*

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Introduction

At the current stage of very complicated mining operations, mines carry out investigations [1–14] aimed to improve performance through the enhanced efficiency of drilling and blasting, improved utilization factor of blastholes (UFB) and other parameters of heading, introduction of modern blasthole charge designs, laying out of blasting patterns with regard to the depth-dependent physical and mechanical properties and processing behavior of rocks. To this end, particular attention is given to adherence to severe requirements imposed on smooth wall blasting in underground mines, identification of causes of overbreaks in blasting, development of technological concepts of heading using different blasting techniques, analysis of blast effects of different explosive charges and to novel and innovative methods of blasting in underground mining.

Problem formulation

Despite an ample amount of accomplished research, the analysis of theory and practice of drilling and blasting in mines of Almalyk and Navoi Mining and Metallurgical Companies reveals the lack of theoretical and practical knowledge on heading in underground mines in difficult geological conditions, deficiency of the analysis of perimeter control blast designs in underground mines, as well as insufficient understanding of the effect of shaped charges.

The literature review shows that the UFB reaches on the average 0.87 in Uzbekistan and is one of the major criteria of the blast quality [15–21]. It is considered that a blast is good when the UFB is 0.90 and higher, normal — UFB 0.80–0.85 and bad — UFB lower than 0.65–0.75. It is expedient to undertake scientific research to find new science-based ways to improve drilling and blasting techniques in underground mining.

Smooth wall blasting parameters in underground mines

Research and application of the smooth wall or perimeter control technology in blasting show [1–5] that efficiency of the technology depends on the correct choice of spacing of blastholes in the perimeter row (a_p), closeness ratio K_{close} (the ratio of spacing in the perimeter row to burden relative to the buffer row) and charging ratio K_{charge} (the ratio of charge volume to blasthole volume). Chosen correctly for specific geological conditions, these parameters allow charges in the perimeter row to produce sufficient blast energy to cut off rock along the pre-set perimeter without significant overbreaks.

The experimentation procedure is developed, including a series of tests of the influence exerted by the main smooth wall blasting parameters on the blasting efficiency in the method of closer spaced charges in a few representative geological conditions.

The procedure stipulates six groups of tests aimed to determine smooth wall blasting parameters with the further grapho-analytical processing of the results.

The scope of the experimental investigations embraced a few influences:

Group 1 — influence of blasthole spacing within a heading on overbreaks;

Group 2 — influence of closeness ratio of blastholes within a heading on overbreaks;

Group 3 — influence of shattering effect and explosive force of perimeter charges, charging ratio and charge design on overbreaks and final wall damage;

Group 4 — influence of blasting sequence and delays in perimeter blastholes on roof surface quality;

Table 1. Experimental data on smooth wall condition after blasting

Spacing of perimeter blastholes, cm	Post-blasting wall condition
40	No traces of blastholes. The wall is a damaged zone with overbreaks ≈15 cm
50	Vague traces of blastholes. Major fracturing at the locations of charges and between them. Overbreak is 10 cm
60	Apparent traces of blastholes. Weakly visible fracturing length-wise blastholes. Front solid block without observable damage. Overbreak is 8 to 10 cm.
70	Clear traces of blastholes without visible damage of rocks and no observable overbreak

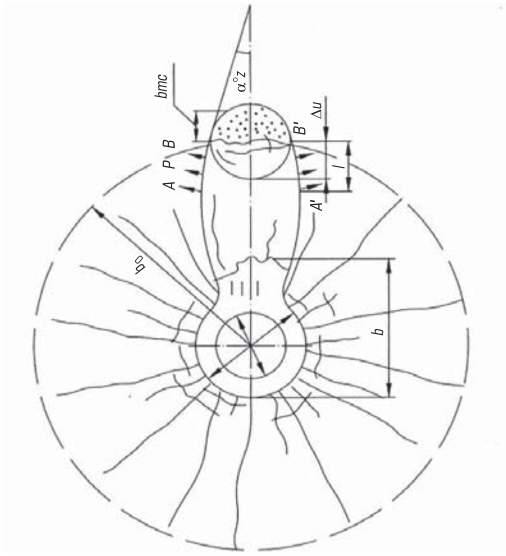

Fig. 1. Damage of rocks around production blasthole with uncharged buffer blasthole in radiating cracking zone

Group 5 — influence of incline of perimeter blastholes on the closer spaced blasting method efficiency;

Group 6 — influence of structural geology on blasting efficiency.

Regarding the influence of shattering effect and explosive force of explosive charges on the smooth wall blasting efficiency shows [6–12] that one of the determinants in this case is the correct choice of a type of explosive. Practice of the smooth wall blasting says that charges in the perimeter row should possess a low shattering effect at a satisfactory capacity (or explosive force) to ensure a shock wave that is sufficient for rock breaking, or the shattering effect of a charge at the moment of explosion should be artificially reduced.

In the study of the influence of spacing in the perimeter row on the blasting efficiency, the first series tests aimed to determine an optimum spacing in slightly jointed sandstone with thin siltstone interbeds. The factor of hardness on Protodyakonov's scale was $f = 12–14$, and the compressive strength of rocks was 840–960 kg/cm^2. Before starting the tests, with regard to the available range of explosives and firing agents, a perimeter row charge was designed as 5 cartridges of powdered ammonite (no. 6 ZhV) fastened on a guide rod at a distance of 10 cm from one another. An explosive cord was laid along the charge to transfer detonation between the cartridges. Earlier, 5 charges of that design were tested in terms of the detonation efficiency and completeness at a test site of an explosive base depot. Firing used delayed-action electric detonators no. 7 (delay of 500 ms) actuated by a blaster. All charges exploded in full (**Table 1**).

The influence of a closeness ratio on the smooth wall blasting efficiency was studied in two stages. The first stage was the test of overbreaks versus K_{close} in rocks having $f = 12$. The second stage revealed the relationship between K_{close} and the rock hardness using the data obtained during introduction of the smooth wall blasting technology.

The first stage test site was a face with a cross section of 8 m^2 in compact rocks ($f = 14$) with a few tectonic fractures. The blasting procedure used in the face testing was proposed when determining optimum values of the process parameters.

The most effective actions to reduce initial pressure of explosion products is the use of decoupled and elongated charges of standard-diameter explosive cartridges. The construction involves decoupling of charges by inert gaskets, or smaller diameter cartridges [13–14]. For the purpose of transmitting pressure along a decoupled charge with the gaps of 5 cm between the cartridges (the distance of transmission of detonation for most industrial explosives), a detonating cord is connected to a primer placed on the bottom of a blasthole. The charging ratio in such design charges is varied subject to the size of the inert gasket.

Analysis of cavity formation in blasting and development of *effective stemming* by blasting additional shortened blasthole on a full scale

The completed research made it possible to develop a heading procedure with smooth wall blasting for the Kauldy Mine of Almalyk Mining and Metallurgical Company.

The study was performed into the process of breaking through an unbroken rock block by directional blasting toward a buffer space nearby. The objective was to examine formation of cracks around the production blasthole and buffer blasthole and in adjacent rock mass, and to compare the actual and theoretical data.

In the tests, the sets of two blastholes — production blasthole and buffer blasthole — were drilled. The distance between the sets was such that to exclude their cross effect during explosion. The length of the blastholes was 2 m. The diameter of the production blastholes was 42 m, and the diameters of the buffer blastholes were 42 and 50 mm. The production blastholes were charged with cartridges of ammonal with a diameter of 36 mm.

After explosion the shapes of initiated cracks were examined, the azimuthal angles between the cracks and between the formed shear surfaces were measured, the radius of the fine crushing zone was determined, and the cases of pressing of the cushion space were recorded.

Around the production blasthole, a fine crushing zone with the radius $в$ and a zone of radiating cracks with the radius $в_r$ formed. The number of the radiating main cracks was approximately the same in blasting with and without the cushion space — 24–28 cracks. The cracks were bow-shaped and created shear surfaces, and rocks displaced along these surfaces toward the cushion space.

The structure and mechanisms of rock compression during breaking through a solid rock block were studied. To this effect, the cavity resultant from displacement of disturbed rock mass was measured and inspected visually. The results are given in **Fig. 1**.

Visual inspection of the blasted sets shows that compressed rock is structurally nonuniform and consists of two sharply different zones: the zone of minor disturbed sheared inter-blasthole solid block and the finely crushed zone compressed by the sheared rock block.

Density of the compressed zone decreases as the production blasthole gets closer to the buffer blasthole, and the width of this minor disturbed and sheared zone grows. Then density of this zone was measured using a thin-walled drifbolt made of stainless steel. The drifbolt was driven in this zone and was removed together with core. The density was determined as the ratio of the core mass to the volume of the crater formed.

Displacement of the inter-blasthole rock block is recommended to be found from the formula [15]:

$$\Delta U = \left[1 - \frac{\rho_{bd}}{\rho_r}\left(1 + 0.43\left(\bar{a}\right)^2\right)\right] d_b = 0.3 d_b \left(1 - \left(\bar{a}^2\right)\right), \text{ m}, \qquad (1)$$

where ρ_{bd} is the bulk density of rock, t/m^3; ρ_r is the rock density, t/m^3; d_b is the buffer blasthole diameter, m; $\bar{a}$ is the block width, m.

Unit pressure P_{sb} on this solid rock block is given by [15]:

$$P_{sb} = \frac{0.3d_k\left[1-\left(\frac{a\left[\sigma_t\right]_d K_h}{\left[\sigma_c\right]_d d_h K_{burn}}\right)^2\right]^2 \mathrm{tg}\alpha_z E}{2S_{AB}}, \text{ MPa,} \tag{2}$$

where a is the spacing of the blastholes, m; $[\sigma_c]_d$ and $[\sigma_t]_d$ are the dynamic compressive and tensile strength limits, respectively, MPa; K_h is the coefficient of the length of the blasting hole; K_{burn} is the burn of the production blasthole; d_h is the blasthole diameter, m; S_{AB} is the unit area of deformation in the inter-blasthole block, m^2; α_z is the angle between a tangent to the sliding surfaces at their closing to the plane that bridges the centers of the production and buffer blastholes, deg; E is the longitudinal elasticity, MPa.

The optimum spacing of the production and buffer blastholes is determined for rocks of different hardness.

The width b of the formed cavity is related with the spacing between the production and buffer blastholes, a and a_{lim}, based on the data of experimental blasting in the Kauldy Mine and from the calculations for the same rocks from the relations:

$$a < 0.3a_{lim};$$

$$b = \frac{d_h K_{burn} + d_b}{2} + a, \tag{3}$$

and when $d_h \ge d_b$, $b = \frac{d_h\left(K_{burn}+1\right)}{2} + a.$

If $a \ge 0.35$

$$b = d_h K_{burn} + 0.3_d\left(1-a^2\right) + C\left(u\right), \text{ m;}$$

$$d_h = d_b b_h = d_h\left(K_{burn} + 0.3d_b\left(1-a^2\right)\right) + C\left(u\right), \tag{4}$$

where $C\left(u\right)$ is the function to take into account partial failure of the solid block at the contact with the charge cavity at the moment of the block displacement along the shear surfaces.

The influence of an *effective stemming* made by explosion on the increase in the useful work of the production charge is analyzed.

It is found that the properties of the *effective stemming* depend on the blasthole-to-blasthole spacing a. On this basis, a new design of the *effective stemming* is found as blasting of an inclined additional blasthole drilled such that its axis and the production blasthole axis are two meeting but non-intersecting curves (**Fig. 2**).

The tests determined the angle between the paths of the *stemming* and production blastholes such that the blasting-caused damage is maximal. The relation of the efficiency of the production blasthole (the indicator was the blasthole burn K_{burn} as function of the stemming blasthole incline β_{stem} was determined.

For finding this relationship, the sets of two blastholes, the stemming blasthole 1 m long and the production blasthole 2 m long, were drilled using a hand drill. The angle between the axes of the blastholes in the sets was varied from 0 to 45°. The distance between the axes of the blastholes at the intersect of their paths was 3 diameters d_h. Diameter of the blastholes in a set was 40 mm. The spacing of the set excluded their cross effect. The blastholes were charged with cartridged ammonite (no. 6 ZhV) with a diameter of 36 mm. The stemming and the production blastholes were undercharged by 0.4 and 1 m, respectively.

After blasting in a set of blastholes, the diameter of the cavity of the production charge was measured and K_{burn} was determined in each test set. The results show that the fracture work of an explosive charge is the most efficient at the stemming blasthole angles of 8–12°. At the larger angle

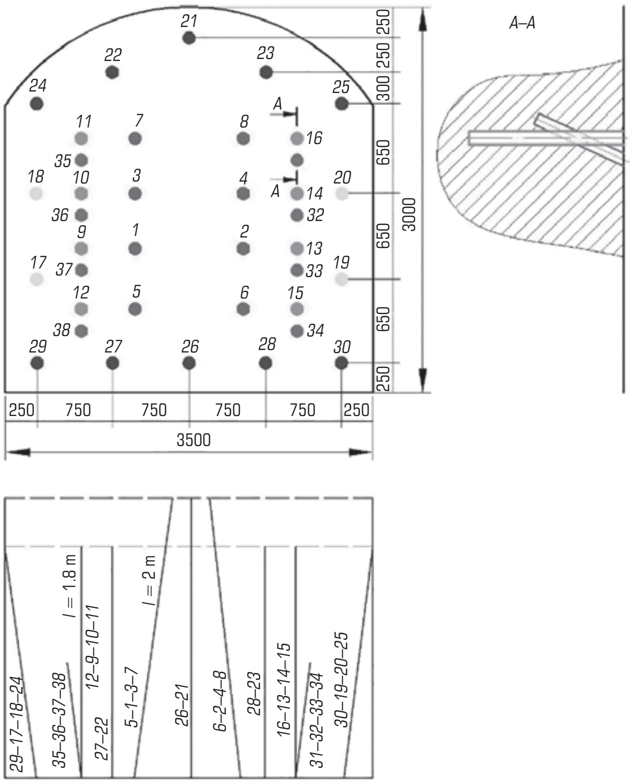

Fig. 2. Blasting pattern at mining face to study optimum angle β between stemming 1 and production 2 blastholes:
1–8 – cutting blastholes; *9–16* – production blastholes; *17–30* – perimeter blastholes; 31–38 – shortened inclinead stemming blastholes

β_{cut}, the effect of the stemming made by the shorter blasthole decreases, and the blasting-caused fracture effect drops accordingly. The pilot tests of the designed *effective stemming* proved the efficiency of the stemming made by blasting a shortened inclined *stemming* blasthole drilled at an angle of 8–12°. The designed stemming improved burning of blastholes.

Effect of explosive cumulative charges in underground heading

The penetration depth of a cumulative jet is determined as function of the jet length equal to the length of a generated cumulative cavity, its density, rock mass density, as well as compressibility of rocks and the jet material:

$$l_{pd} = l\sqrt{\frac{\rho_j}{\rho_r}\frac{\varepsilon_{fp}}{\varepsilon_j}}, \tag{5}$$

$$l_{pd} = l\sqrt{\frac{\rho_j}{\rho_r}k_{com}}, \tag{6}$$

where l is the jet length equal to the length of a generated cumulative cavity, m; ρ_j is the density of the jet material, t/m^3; ρ_r is the rock mass density, t/m^3; ε_{fp} and ε_j are the relative compressibilities of the fused point and jet, respectively; k_{com} is the ratio of rock/jet material compressibilities.

Figure 3, *a* depicts the fracture depth in rocks relative to the density of the jet material and density of rock mass, and **Fig. 3, *b*** shows the change in the depth of rock fracture relative to the rock and jet compressibilities.

The most promising way of enhancing the utilization factor of a blasthole is employing the cumulation effect which makes it possible to distribute explosion energy in space and to concentrate it depthward rock mass to be fractured [20–21].

The implemented numerical modeling of a cumulative charge allowed a new design of a shaped explosive charge (**Fig. 4**).

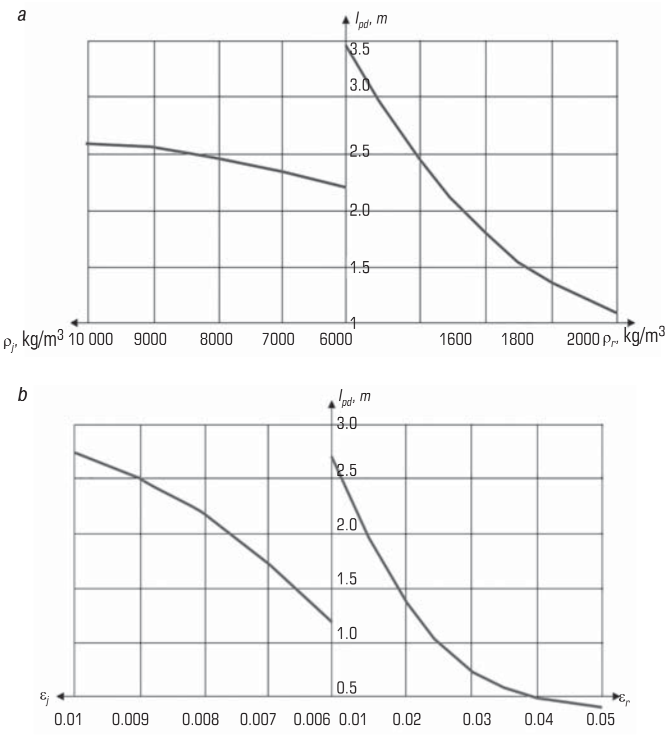

Fig. 3. Change in penetration depth of rock fracture versus:
a – densities of jet material and rock mass; *b* – relative compressibilities of rock mass and jet material

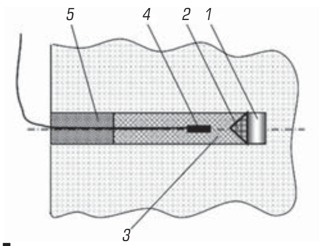

Fig. 4. Design of shaped explosive charge:
1 – wooden cylinder; *2* – cumulative fan; *3* – explosive; *4* – intermediate detonator; *5* – stemming

On the bottom of a blasthole, a polyvinyl chloride or wooden cylinder 1 is placed to create a focus; on the cylinder, a cumulative fan 2 is placed; then explosive 3 filled, an intermediate detonator 4 is arranged, the rest of explosive 3 is filled and a stemming 5 is formed.

The use of the charge design with the cumulative fan at the blasthole bottom increases the UFB and the fan intensifies explosive fracture of rocks.

Development and commercial test of shaped explosive charges in underground heading

The created cumulative jet enhances energy applied by explosion of rock mass, provides a preset quality of rock fragmentation and smoother wall after blasting, eliminates unexploded bootlegs in blastholes, and increases the utilization factor of blastholes.

From the theoretic investigation of the action of shaped charges with cumulative effect in rocks mass, based on the hydrodynamic theory of cumulation and by solving equations of continuum mechanics, the angle of implosion of cumulative encasing (liner) α_{imp} is correlated with the cone radius and height, the muzzle and jet velocities, the time of implosion and the liner thickness as follows:

$$\alpha_{imp} = \alpha r\,\text{ctg}\left[\frac{r_h}{h_c} + \frac{V_m}{V_j h_d}\left(V_m t + h_l\right)\right], \tag{7}$$

where r_h is the blasthole radius, mm; h_c is the cone height, mm; V_m is the muzzle velocity, km/s; V_j is the jet velocity, km/s; t is the time of implosion, ms; h_l is the thickness of the liner, mm.

This method is implemented in the following manner. At a mining face, a set of production, buffer and perimeter blastholes is drilled according to a certain blasting pattern, and the parameters of the blastholes are determined using the known procedures and experimental blasting results. At the bottoms of the blastholes, a cone made of plastic, wood, metal etc. is placed; the cone diameter equals the blasthole diameter, the height of the cone is calculated from the formula [20]:

$$h = \frac{d}{2\text{tg}\left(\frac{\alpha}{2}\right)}, \tag{8}$$

where α is the angle between the liner walls.

Then, the explosive charge and the primer are placed in the blasthole; their quantity is found from the formula [20]:

$$Q = (0.6 - 1.0)q, \text{ kg}, \tag{9}$$

where q is the powder factor, kg/m^3 (assumed usually as 0.5–0.6 kg/m^3).

Then, the blastholes are charged with a cartridged explosive in use in a mine, and primers with initiation systems are installed. The method of blasting can be whichever, and the explosive amount per blasthole is determined by the drilling and blasting chart. The blasting system is assembled and connected to the main blasting circuit.

The proposed method of blasting using the cumulative effect allows expanding the blasting effect owing to the complete use of explosion energy, increases the UFH and reduces the powder factor, which enables cost saving in drilling and blasting up to 20%.

The theoretical study finds out that a critical component of a shaped explosive charge, which governs its piercing performance in many ways, is the liner of the cumulative cavity. Manufacture of a cumulative liner from iron during underground blasting operations is simple and inexpensive. The best preload is provided by the cumulative liner made of metal with a cubic lattice (copper, iron, aluminum), and the maximal velocity of the head of the cumulative jet in this case reaches 10 740 m/s.

Perimeter blastholes are blasted lattermost in a blasting pattern, as a rule, and the fracture conditions become more favorable owing to an additional exposed surface formed after blasting the previous series of charges. Therefore, a strong compression as in blasting with a single exposed surface is absent. Consequently, there are pre-conditions for a volume system of cracks to appear in the early phase of an explosion, which, as known, leads to the second-phase tensile fracture of rock mass at the lesser input of energy. Accordingly, it is possible to reduce the energy density and, respectively, the charging density artificially, without the loss of the blasting effect. This condition is the main prerequisite for a possible decrease of the blast energy in the perimeter blastholes, which allows minimization of rock mass damage at the blasting perimeter and reduction of overbreaks.

However, mechanical cutdown of the charging density ρ_{ch} in the perimeter blastholes, at the same drilling and blasting parameters (the spacing of the perimeter blastholes a_p; distance to perimeter hole W_k; the closeness ratio K_{close}) calculated by the known procedure with regard to the strength of rocks, despite two exposed surfaces available, can lead to an increase in oversize or to insufficient fracture of a rock block between the perimeter blastholes. On the other side, the increase in the charge in the perimeter blastholes with a view to decreasing the oversize and increasing the perimeter blast reliability can result in excessive fracture of the solid rock blocks between the perimeter blastholes, in overbreaks and in needless damage of adjacent rock mass. In this way, the perimeter control blasting parameters is a set of interrelated values that define the perimeter blasting sequence. According to the experimental research, these values are: the spacing of the perimeter blastholes a_p, the closeness ratio K_{close}, the charging ratio K_{ch} and the delay of electric detonators in the perimeter blastholes t_d.

The spacing of the perimeter blastholes can be found based on the blast energy balance as follows [16]:

$$a_p = 2(a_{pd} + a_{sw}), \text{ cm}, \quad (10)$$

where a_{dp} is the radius of action of detonation products, cm; a_{sw} is the radius of action of shock wave, cm.

The value of a_{pd} in formula (10) defines the zone of aggravated structural rupture of rocks under the action of detonation products, i.e. there is an overload of rocks under compressive strains.

The size of the action zone of detonation products is recommended to find from the formula [20]:

$$a_{dp} = K\left(\frac{P_i}{P_{ov}}\right)^{\frac{1}{3}}\left(\frac{P_{ov}}{P_0}\right)^{\frac{1}{\gamma}} r_h, \quad (11)$$

where K is the average proportionality factor determined by the physical and mechanical properties of rocks; P_{in} is the initial pressure of detonation products in blasthole, kg/cm^2; P_{ov} is the overall pressure of detonation products (from many experiments, a reference value of $P_{ov} \approx 2000$ kg/cm^2; P_0 is the strength characteristic of rocks — the boundary of the shattering effect zone of explosive; γ is the adiabatic index $=7/5–5/4$; r_h is the blasthole radius, cm.

The parameters of the perimeter control blasting are developed for underground mines, and a set of inter-related values that define the perimeter control blasting sequence is determined to include the spacing of the perimeter blastholes, the charging ratio, the closeness ratio and the electric detonator delays in the perimeter blastholes.

It is found that the decrease in the charging ratio by 50% allows diminishing the radius of the action zone of detonation products by 20 times approximately, and, consequently, the minimum radius of the effect of detonation products can be reached with an air gap between the blasthole walls and the explosive charge, or between the explosive cartridges. That is, only with the decreased charging ratio, it is possible to have the minimum overbrake and to see clearer the trace of a blasthole, which implies the absence of the action zone of detonation products.

The question of selecting an optimum burden for the perimeter blastholes is closely related with determining main parameters of the perimeter control blasting. That is, for decreasing overbreaks and the action zone of detonation products, it is necessary to diminish artificially the charge in the perimeter blastholes.

One of the influences on the final wall quality in the perimeter control blasting in underground mines is the correct determination of an allowable time spread in blasting perimeter blastholes using electric detonators. The initiation time of the first cracks in blasting perimeter blastholes depends on the physical and mechanical properties of rock mass, and on the spacing of the perimeter blastholes. The latter circumstance is of special importance in selection of the delays for the electric detonators in the perimeter blastholes; for this reason, in case of the artificial or other-why reduction in the spacing of the perimeter blastholes, the requirements for the actuation time spread of the firing agents should be strengthened.

Conclusions

The tests of the quality of the perimeter control blasting in underground mining shows that the use of this technology allows diminishing the impact zone of blasting in rock mass.

The influence of a perimeter control blasting technique on adjacent rock mass damage was analyzed as a case-study of the Kauldy Mine of Almalyk Mining and Metallurgical Company. The evaluation of the blasting impact zone was a part of an integrated research of the overall fractured zone evaluation with a view to determining parameters of rock bolting.

The comparative analysis of the perimeter control blasting efficiency in underground mining allows concluding that the perimeter control blasting is economical and cost-effective in underground openings with a cross-section of 8–10 m^2.

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