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Using the methods to calculate parameters of drilling and +blasting operations for emulsion explosives

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Abstract
Mathematical modelling of rock mass breaking using blasting has been applied to obtain formulas for the calculation of crush zone radii, intensive fragmentation, and crack formation around the charging cavity in the structure, diameter of the charging cavity, diameter of the charge itself, detonation characteristics of an explosive, boundaries of the rock strength, rock mass jointing, and mineral compression under the effect of rock pressure. The methods have been developed to calculate parameters of drilling and blasting operations (DBOs) while driving the mine workings based upon the idea of the arrangement of blastholes in terms of areas they occupy in a fore-breast as well as upon their location relative to break-off outline. Stage one of the methods involves calculating and designing burn cuts where a distance between blastholes is determined with the help of a fragmentation zone radius. Stage two means calculation of both specific and total explosion consumption per borehole bottom, line of least resistance (LLR) for a borehole in terms of intensive fragmentation, areas of borehole groups, borehole number, analytical and actual distance between boreholes, actual charge amount per borehole, and actual specific and the total explosive (E) consumption per borehole bottom. The methods have been tested in the operating ore mine while driving a mine working. Emulsion explosive (EE) Ukrayinit-PP-2 (Україніт-ПП-2) has been applied. The developed methods have been used to calculate DBOs parameters for the explosive. Trial blasts demonstrated the good firing of a borehole bottom and uniform ore fragmentation; a high coefficient of borehole use has been supported.

Keywords
borehole, emulsion explosive, intensive fragmentation zone, line of least resistance, drilling and blasting operations

Introduction

Currently, rock mass breaking using blast energy is one of the key mining problems (Persson et al., 2018). Depending upon the calculation accuracy of DBOs parameters, the mining cost-performance ratio may vary (Lyashenko et al., 2018). If horizontal and incline mine working is being driven, DBOs make demands connected with the required rock fragmentation as well as with keeping the mine working design outline after the blast. Another tendency for improving DBO procedures is to make blasting safer (Krysin et al., 2004) and reduce their environmental impact (Myronova, 2015) while replacing domestic TNT-containing explosives with EE (Myronova, 2016). As is well known, EE is safe from the viewpoint of transportation and storage (Kholodenko et al., 2014); it is environmentally friendly (Myronova and Borysovs'ka, 2014) and economically sound (Khlodenko et al., 2015). Hence, one of the current topical problems for mining is to improve the efficiency of rock fragmentation by the blast with the help of EE while driving the mine workings and breaking the rock mass.

Rock mass breaking by blasting is characterized by load application to the amount of medium to be ruined. It depends upon numerous factors. The abovementioned is connected with the diversity, complexity, and transience of the events following a blast in a solid environment. Paper (Andrievskii et al., 1997) lists such phenomena of a firing chain as E charge detonation; charging cavity widening; mechanical interaction between detonation products and rock mass; formation and propagation of impact waves; propagation and interaction between stress waves within rock mass and its breakage; movement of the underworked fragmented material; and fly-rock (Dychkovskyi, 2015). As the paper (Kononenko and Khomenko, 2021a) mentions, numerous hypotheses explain the physics of blast destroying rock mass.

In mining practice, other rock mining techniques are used. One of them is mechanical mining, e.g. using full-cut Bolter Miner machines Dyczko et al., 2022) using rotary tools (Rupik and Romanyshyn, 2021; Prysyazhnuk et., 2022) for coals and easily machineable rocks, or TBM techniques for hard-to-machine rocks. There are also other experimental methods of mining rocks of high brevity, which are aimed at eliminating explosives. These are methods of reducing the cohesion of rocks by uprooting them (Siegmund, 2021; Jonak et al., 2021). However, methods using explosives (Kononenko et al., 2022) are the most common in conditions of hard-to-machine rocks due to the rapid mining effect and low operating costs.

Decades of theoretical studies and experiments concerning blast effect mechanisms under different conditions have resulted in the fact that some hypotheses disagree but do not deny the plausibility of the hypotheses themselves. Mainly, researchers have different visions of how to evaluate failure share after wave and quasistatic blast action. The abovementioned factored into numerous theoretical ideas as well as qualitative descriptions of the nature of solid medium breakage. Largely, it also results in the use of numerous empiric calculation formulas while developing DBO parameters. Currently, most such methods to calculate DBO parameters involve characteristics of TNT-containing explosives, ignoring detonation properties of EEs being higher than those in TNT-containing analogues. The main criterion used up to now is the quantity of boreholes being directly proportional to E quantity which is required to break down the specified rock volume. Consequently, relying upon common action of quasistatic and wave hypotheses of blast effect in rock mass, it is required to develop such calculation methods for DBO parameters, which should take into consideration the impact of physical and mechanical properties of medium as well as detonation characteristics of a blasting agent.

Methods of the research

Development of the methods, simulating DBO parameters while driving a mine working, involved the following stages.

1. Mathematical modelling of rock mass failure around charging cavity using blasting and determination of regularities to identify radii of crush zones, intensive fragmentation, and crack formation;
2. Development of the methods to calculate DBO parameters for driving a mine working based upon the idea of arrangement of blastholes in terms of areas they occupy in a fore-breast as well as upon their location behind break-off outlines; and
3. Approvement of the methods to calculate DBO parameters while driving a mine working with the use of EEs.

The development of the methods to calculate DBO parameters while driving a mine working involved common laws of elasticity theory and the basic postulates of quasistatic and wave hypothesis of a mechanism for solid medium breakage by means of a blast.

Results and discussion

Mathematical modelling of rock mass breaking around a charging cavity using blasting

According to a new theory of rock mass breaking using blast energy (Kononenko and Khomenko, 2021a), after the EE charge was blown out, an impulse wave would propagate everywhere in the rock mass (Kononenko and Khomenko, 2021b). A certain amount of rock, being close to a charging cavity, will be compressed in the normal direction and stressed in the tangential one. Within the zone front, a wave of mechanical stresses will excess a module of volume medium compression. Hence, rock breaks in the neighbourhood of the charge, creating a plastic deformation zone, the so-called shear zone. After the shear zone shaping and distancing from the E charge location, compression stress by an impact wave drops rapidly, becoming less away compared with the rock compression strength. In such a way, rock stops its breakage owing to the radial stress compressing it. A decrease in radial stress action results in the increased tangential stress stretching rock in both directions. As a result of the blast effect, the impact wave turns into a stress way with the formation of a following zone, i.e. fragmentation zone. Within the area, shear stress acts as well as tensile and compression stresses. Hence, elastoplastic strain arises in the rock. The stresses form two following zones: the intensive fragmentation zone, where compressive stress acts and the crack formation zone, where shear and tensile stresses act.

The mathematical models of crush, intensive fragmentation, and crack formation taking place in rock mass around the charging cavity have been developed with the help of an analytical model shown in Fig. 1. Paper (Kononenko and Khomenko, 2021a) gives its complete explanation.

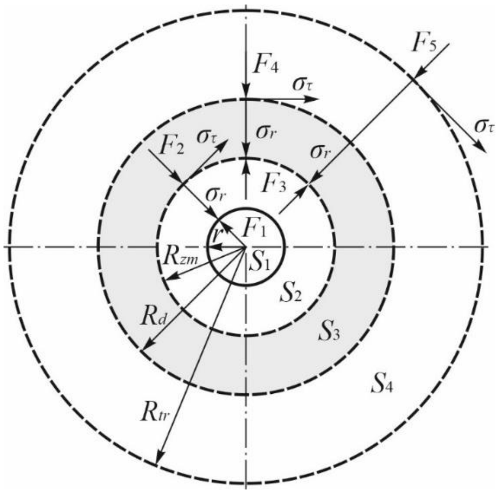

Fig. 1. A parametric model to identify zones of crush, intensive fragmentation, and crack formation shaped around a charging cavity in the process of E blast

Mechanical stresses arising in rock mass because of blast are

$\sigma = \frac{P_1 \cdot r^2}{R_{zm}^2 - r^2}$, N/m^2 for a crush zone; (1)

$\sigma = \frac{P_2 \cdot R_{zm}^2}{R_d^2 - R_{zm}^2}$, N/m^2 for an intensive fragmentation zone; and (2)

$\sigma = \frac{P_2 \cdot R_{zm}^2}{R_{tr}^2 - R_{zm}^2}$, N/m^2 for a crack formation zone (3)

where r is the radius of a charging cavity, m; R_{zm} is the crush radius, m; R_d is the intensive fragmentation zone radius, m; R_{tr} is the crack formation zone radius, m; and P_2 is the pressure drop of the explosion products on the rock mass owing to the increase in a contact area (Andrievskii et al., 1996)

$P_2 = (P_1 \cdot r)/R_{zm}$, Pa, (4)

where P_1 is pressure by the explosion products (paper (Torbica and Lapčević, 2015)

$P_1 = \frac{\rho \cdot D^2}{8} \cdot K_{dz}$, Pa, (5)

where ρ is E density or EE density (Kononenko et al. 2021), kg/m^3; D is detonation velocity of E, m/s; and K_{dz} is the coefficient taking into consideration changes in the explosion product pressure on the charging cavity walls depending upon E charge diameter (Kononenko et al., 2021b)

$K_{dz} = (d_z/d)^3$, (6)

where d_z is E charge diameter, m; and d is charging cavity diameter, m.

(1)-(3) are equations of Lame problem. According to the theory, while calculating thick-wall cylinders, if the action of internal pressure takes place, one should remember that radial stress σ_r in each point of the cylinder will be negative (compression stress) and σ_τ stress will be positive (tensile stress). Hence, σ_r and σ_τ stresses are the key ones. To determine equivalent stress σ_{ekv} in the volume stress state, it is expedient to apply the Third strength theory supported sufficiently by the research for materials responding similarly to stretching and compression and can be broken by shear. The main stresses are

$\sigma_1 = \sigma_\tau = \sigma$; $\sigma_2 = \sigma_o = 0$ and $\sigma_3 = \sigma_r = -\sigma$.

According to the Third strength theory, equivalent stress is

$\sigma_{ekv} = \sigma_1 - \sigma_3$; or
$\sigma_{ekv} = \frac{2 \cdot P_1 \cdot r^2}{R_{zm}^2 - r^2}$, N/m^2 for a crush zone;
$\sigma_{ekv} = \frac{2 \cdot P_2 \cdot R_{zm}^2}{R_d^2 - R_{zm}^2}$, N/m^2 for an intensive fragmentation zone; and
$\sigma_{ekv} = \frac{2 \cdot P_2 \cdot R_{zm}^2}{R_{tr}^2 - R_{zm}^2}$, N/m^2 for a crack formation zone.

Triaxial compression condition $\sigma_{ekv} = \sigma_{st}$ is fulfilled for the crush and intensive fragmentation zones. As it is known from the elasticity and plasticity theory, if the external cylinder diameter is 4 times more than the internal one and calculation divergence is up to 6%, then in this case, the solution is not connected with an external outline shape, and the cylinder is under the conditions of pure shear. In such a way, the condition is $\sigma_{ekv} = \tau_z$ for a crack formation zone. Relying upon the abovementioned, taking into consideration dynamic coefficient under impact loading, and having performed the required transformations, the mathematical models of the zone radii have been derived (Kononenko and Khomenko, 2021a):

$R_{zm} = 0.5 \cdot d \cdot \sqrt{1 + \frac{\rho \cdot D^2 \cdot K_{dz}}{2 \cdot \sigma_{st}}}$, m for a crush zone; (7)

$R_d = R_{zm} \cdot \sqrt{1 + \frac{\rho \cdot D^2 \cdot d \cdot K_{dz}}{8 \cdot R_{zm} \cdot \sigma_{st}}}$, m for an intensive fragmentation zone; and (8)

$R_{tr} = R_{zm} \cdot \sqrt{1 + \frac{\rho \cdot D^2 \cdot d \cdot K_{dz}}{8 \cdot R_{zm} \cdot \tau_z}}$, m for a crack formation zone (9)

where σ_{st} is rock compressive strength, Pa; and τ_z rock shear strength, Pa.

The obtained (7)-(9) formulas calculate radii of crush, intensive fragmentation, and crack formation zones shaped around a charging cavity involving diameters of the charging cavity as well as E charge, detonation characteristics of an explosive, and rock strength. However, the formula cannot take into consideration rock mass fissility and rock consolidation under the rock pressure effect. Consequently, to improve the calculation accuracy of radii of the zones, introduce coefficients of structural rock mass loosening and rock consolidation under the effect of gravitation (rock pressure) in the formulas. Performance of the required transformations results in the formulas to calculate the zone radii

$R_{zm} = 0.5 \cdot d \cdot \sqrt{1 + \frac{\rho \cdot D^2 \cdot K_{dz}}{2 \cdot \sigma_{st} \cdot K_s \cdot K_u}}$, m for a crush zone; (10)

$R_d = R_{zm} \cdot \sqrt{1 + \frac{\rho \cdot D^2 \cdot d \cdot K_{dz}}{8 \cdot R_{zm} \cdot \sigma_{st} \cdot K_s \cdot K_u}}$, m for an intensive fragmentation zone; and (11)

$R_{tr} = R_{zm} \cdot \sqrt{1 + \frac{\rho \cdot D^2 \cdot d \cdot K_{dz}}{8 \cdot R_{zm} \cdot \tau_z \cdot K_s \cdot K_u}}$, m for a crack formation zone (12)

where K_s is the coefficient of structural rock mass loosening; and K_u is the coefficient of rock consolidation under the effect of gravitation, which can be calculated on the formulas represented by (Kononenko and Khomenko, 2021a)
$K_u = (\gamma + 0.1 \cdot H)/\gamma$; (13)

where γ is rock density, kg/m^3; and H is mining depth, m.

Developing methods to calculate DBO parameters of driving a mine working

Direct cut parameters

Correct selection of a set of boreholes, providing the maximum coefficient of their use while influencing the velocity of the mine working drive, is an important condition for efficient roadway construction. Such rational DBO parameters as a cut type (Andrievskii, 1993), arrangement of boreholes and their number, and E type and explosive-charge size depend upon specific mining and geological conditions. Cuts are quite a popular modern technique for underground ore mining. Depending upon the arrangement of cutter boreholes relative to the borehole bottom, they are classified as inclined cuts (disintegration) and direct cuts (fragmentation). Review of DBO practices while developing mine working and crosscut driving in Kryvbas mines as well as in mines of *Zaporizky zalizorudny kombinat* (ZZRK) PJSC has helped understand that the cut cavity shaping involves direct prismatic cuts either with compensatory boreholes or without them. The cuts are characterized by simplicity, flexibility, capacity, and stability. Inclined vertical and wedge cut is used rarely. That is governed by the restricted possibility of drilling inclined boreholes with the help of self-propelled rigs since their depth depends upon transverse dimensions of mining workings. The following order is proposed to calculate direct cuts.

Paper (Kononenko and Khomenko, 2021a) proposes to identify a breakthrough distance between the direct cut boreholes (Andrievskii et al., 1997) using the crush zone radius according to formula (10).
The minimum area of a direct cut is (Kononenko et al., 2021b)

$$S_{vr} = 0.45 \cdot (l_{sh} \cdot \eta)^{0.91} S_{vr} = 0.45 \cdot (l_{sh} \cdot \eta)^{0.91}, \text{ m}^2 \quad (14)$$

where l_{sh} is the length of a set of boreholes, m; and η is the efficiency of the boreholes varying from 0.85 to 0.95.
Minimum number of boreholes in the cut, ignoring compensatory (idle) ones

$$N_{vr} = \frac{S_{vr}}{\pi \cdot R_{zm}^2} N_{vr} = \frac{S_{vr}}{\pi \cdot R_{zm}^2}, \text{ pieces.} \quad (15)$$

After the cut calculation and acceptance, one calculates the number of boreholes, their arrangement within a borehole bottom, and the total E consumption per borehole.

Methods to calculate DBO parameters

In the context of underground iron ore mining, the rectangular and arch shape of mine workings is the most reasonable for ZZRK PJSC mines. An arch crosscut shape of mine workings is the most acceptable for Kryvbas mines. Hence, calculation methods for DBO parameters to drive horizontal and incline mine workings will be formulated for the types.

The new methods for calculation of DBO parameters to drive horizontal and incline mine workings B are based upon the idea of groups of borehole arrangement in terms of the areas they occupy within the borehole bottoms as well as their placement behind the outlines to be cut off (Khomenko et al., 2011). In the context of the new methods (Kononenko et al., 2021b), the DBO parameters will be calculated using the following basic stages. Stage one is cut calculation and design (Khomenko et al., 2019), and stage two involves the calculation of the number of boreholes, their arrangement within a borehole bottom as well as the total E consumption per borehole (Kononenko et al., 2019).

Borehole drilling area

$$S_{pr} = B_{pr} \cdot \left(H_{pr} - \frac{B_{pr}}{3} + 0.26 \cdot B_{pr}\right) S_{pr} = B_{pr} \cdot \left(H_{pr} - \frac{B_{pr}}{3} + 0.26 \cdot B_{pr}\right), \text{ m}^2 \quad (16)$$

for a rectangular and arch shape of mine workings; and

$$S_{pr} = B_{pr} \cdot (H_{pr} - 0.5 \cdot B_{pr}) + 0.125 \cdot \pi \cdot B_{pr}^2 S_{pr} = B_{pr} \cdot (H_{pr} - 0.5 \cdot B_{pr}) + 0.125 \cdot \pi \cdot B_{pr}^2, \text{ m}^2 \quad (17)$$

for an arch crosscut shape of mine workings;

where B_{pr} is drilling width, m; and H_{pr} is drilling height, m. Analytical specific E consumption is (Kononenko et al., 2021b)

$$q = 0.01 \cdot \sigma_{st} \cdot K_{SP} \cdot k \cdot eq = 0.01 \cdot \sigma_{st} \cdot K_{SP} \cdot k \cdot e, \text{ kg/m}^3 \quad (18)$$

where σ_{st} is rock compression strength, MPa; K_{SP} is the coefficient taking into consideration the rock structure being 0.8 for fine-pored and loose rock, 1.1 for fragile rocks, 1.3 for shale rocks with varying strength and layering, perpendicular to a borehole direction, 1.4 for rocks with irregular occurrence and shallow fissility, 2.0 for viscous rocks; e is coefficient of relative E efficiency calculated according to the methods represented by paper (Kononenko et al., 2019); and k is rock clamping coefficient

$$k = 6{,}5/\sqrt{S_{pr}}, \quad (19)$$

Amount of rock separated from the formation

$$V = S_{pr} \cdot l_{sh}, \text{ m}^3. \quad (20)$$

Analytical E amount per borehole

$$Q = q \cdot V, \text{ kg}. \quad (21)$$

Taking into consideration the operation of borehole charges as well as their placement relative to the open surface, it is recommended to calculate LLR in terms of the radius of an intensive fragmentation zone (Kononenko and Khomenko, 2021a; Kononenko et al., 2022).
Analytical LLR values are as follows:

$$W = R_{zm} \cdot \left(1 + \frac{\rho \cdot D^2 \cdot d \cdot K_{dz}}{8 \cdot R_{zm} \cdot \sigma_{st} \cdot K_S \cdot K_u}\right)^{0.5}, \text{ m} \quad (22)$$

for charge explosives and emulsion ones; and

$$W = R_{zm} \cdot \left(1 + \frac{\rho \cdot D^2 \cdot d}{8 \cdot R_{zm} \cdot \sigma_{st} \cdot K_S \cdot K_u}\right)^{0.5}, \text{ m} \quad (23)$$

for granular explosives and emulsion ones.

The area of groups of boreholes for a rectangular and arch shape as well as an arch shape is calculated according to the analytical design represented in Fig. 2.

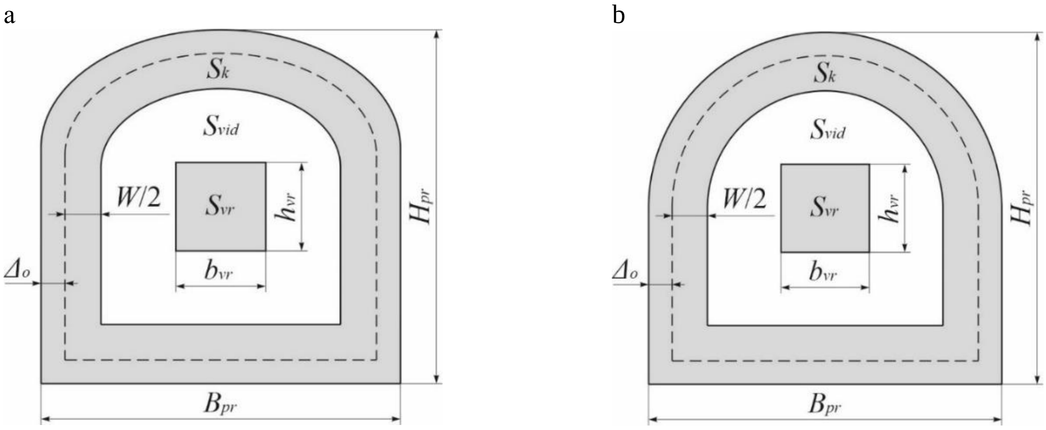

Fig. 2. Analytical design to identify the area of groups of boreholes for a rectangular and arch shape (a) as well as an arch shape (b) of mine workings

The area of outline boreholes is calculated as follows:

$$S_k = S_{pr} - \left(B_{pr} - 2 \cdot \Delta_o - W\right) \times$$
$$\times \left((H_{pr} - 2 \cdot \Delta_o - W) - \frac{(B_{pr} - 2 \cdot \Delta_o - W)}{3} + 0.26 \cdot (B_{pr} - 2 \cdot \Delta_o - W)\right), \text{ m}^2 \quad (24)$$

for a rectangular and arch shape of mine workings; and

$$S_k = S_{pr} - (B_{pr} - 2 \cdot \Delta_o - W) \times$$
$$\times \left((H_{pr} - 2 \cdot \Delta_o - W) - 0.5 \cdot (B_{pr} - 2 \cdot \Delta_o - W)\right) - 0.125 \cdot \pi \cdot (B_{pr} - 2 \cdot \Delta_o - W)^2, \text{ m}^2 \quad (25)$$

for an arch shape (b) of mine workings
where Δ_o is the distance from the mine working boundary to a line of outline boreholes being equal to a value of crush zone radius R_{zm}. In practice, the distance varies from 0.15 to 0.25 m.

The area of a borehole bottom for cut boreholes is

$$S_{vid} = S_{pr} - (S_{vr} + S_k), \text{ m}^2, \quad (26)$$

where S_{vr} is cut area, m^2.

If the borehole bottom area for cut boreholes S_{vid} is either equal to zero or negative, then there is no group of cut boreholes in the borehole bottom. Consequently, the average design LLR is determined with the help of (22) or (23) formula.
The design number of cut boreholes is

$$N_{r.v} = \frac{1{,}27 \cdot q \cdot S_{vid}}{\rho \cdot d_z^2 \cdot k_z}, \text{ pieces}, \quad (27)$$

where k_z is the coefficient of borehole charging varying from 0.30 to 0.85 or being determined by the formula

$$k_z = 0.225 \cdot \sigma_{st}^{0.25}, \quad (28)$$

where σ_{st} is compressive rock strength, MPa.

The obtained quantity of cut boreholes should be analyzed. Their large number results in the increased labour intensity and duration of drilling operations, which decelerates mine working driving (Falshtynskyi et al., 2020). By contrast, small borehole quantity results in poor rock fragmentation, complicating its loading and transporting. Blasting practices to drive mine workings have helped us understand that the optimum number of cut blastholes is when 1-2 boreholes account for a square meter of a borehole bottom separated by them. A large number of boreholes supports the idea that inadequate E was selected with insufficient capacity; in addition, a charge diameter is too small. In this case, it is required to apply the most powerful E, increase the charge diameter, and recalculate the quantity of boreholes.

The borehole area, falling on one cut borehole is

$$S_{v.sh} = \frac{S_{vid}}{N_{r.v}}, \text{ m}^2. \quad (29)$$

The corrected analytical LLR of a borehole is

$$W_{sr.v} = \sqrt{S_{v.sh}}, \text{ m}. \quad (30)$$

The analytical distance between cut boreholes in a line is

$$a_{r.v} = m \cdot W_{sr.v}, \text{ m} \quad (31)$$

where m is approaching coefficient for cut boreholes varying from 1.0 to 1.3. The number of lines of cut boreholes is

$$n_{h.r.v} = \frac{0.5 \cdot B_{pr} - \Delta_o - 0.5 \cdot b_{vr}}{W_{sr.v}} - 1, \text{ pieces in terms of a mine working length}; \quad (32)$$

and

$$n_{v.r.v} = \frac{0.5 \cdot H_{pr} - \Delta_o - 0.5 \cdot h_{vr}}{W_{sr.v}} - 1, \text{ pieces in terms of a mine working height} \quad (33)$$

where b_{vr} is cut width, m; and h_{vr} is cut height, m. The actual distance between the lines of cut boreholes is:

$$W_{h.f} = \frac{0.5 \cdot B_{pr} - \Delta_o - 0.5 \cdot b_{vr}}{n_{h.r.v} + 1}, \text{ m in terms of a mine working width}; \quad (34)$$

and

$W_{v.f} = \frac{0.5 \cdot H_{pr} - \Delta_o - 0.5 \cdot h_{vr}}{n_{v.r.v} + 1}$, m in terms of a mine working height. (35)

The optimum outlines to arrange the lines of cut boreholes are the lines following the shape of a mine working crosscut.

The number of cut boreholes in the i^{th} outline of the walls and roof is:

$n_{bp.v(i)} = \frac{2 \cdot \left(h_i - \frac{b_i}{3}\right) + 1.33 \cdot b_i}{a_{r.v}} - 1$, pieces for rectangular and arch shape of mining workings; (36)

and

$n_{bp.v(i)} = \frac{2 \cdot (h_i - 0.5 \cdot b_i) + 0.5 \cdot \pi \cdot b_i}{a_{r.v}} - 1$, pieces for arch shape of mine workings (37)

where b_i is the width of the i^{th} outline of the cut boreholes, m; and h_i is the height of the i^{th} outline of the cut boreholes, m.

The actual distance between the cut boreholes in the i^{th} outline from the walls and roof is:

$a_{f.bp.v(i)} = \frac{2 \cdot \left(h_i - \frac{b_i}{3}\right) + 1.33 \cdot b_i}{n_{bp.v(i)} + 1}$, m for a rectangular and arch shape of mining workings; (38)

and

$a_{f.bp.v(i)} = \frac{2 \cdot (h_i - 0.5 \cdot b_i) + 0.5 \cdot \pi \cdot b_i}{n_{bp.v(i)} + 1}$, m for an arch shape of mining workings. (39)

The number of cut boreholes in the i^{th} outline of a bottom is:

$n_{p.v(i)} = (b_i / a_{r.v}) + 1$, pieces. (40)

The actual distance between the cut boreholes in the i^{th} outline of a bottom is:

$a_{f.p.v(i)} = \frac{b_i}{n_{p.v(i)} - 1}$, m. (41)

The number of cut boreholes in the i^{th} outline is:

$N_{v(i)} = n_{bp.v(i)} + n_{p.v(i)}$, pieces. (42)

The total number of the cut boreholes is:

$N_{vid} = \sum_{i=1}^{n} N_{v(i)}$, pieces. (43)

The analytical distance between the outline boreholes is:

$a_{r.k} = m \cdot W_{sr.v}$, m (44)

where m is the approaching coefficient of charges for the outline boreholes varying from 0.75 to 0.95, where a lesser value is assumed for the outline boreholes of bottom.

The number of outline boreholes from the walls and roof is:

$N_{bp.k} = \frac{2 \cdot \left((H_{pr} - 2 \cdot \Delta_o) - \frac{B_{pr} - 2 \cdot \Delta_o}{3}\right) + 1.33 \cdot (B_{pr} - 2 \cdot \Delta_o)}{a_{r.k}} - 1$, (45)

pieces for a rectangular and arch shape of mine workings,

and

$$N_{bp.k} = \frac{2\cdot((H_{pr}-2\cdot\Delta_o)-0.5\cdot(B_{pr}-2\cdot\Delta_o))+0.5\cdot\pi\cdot(B_{pr}-2\cdot\Delta_o)}{a_{r.k}} - 1, \quad (46)$$

pieces for an arch shape of mining workings.

The actual distance between the outline boreholes from walls and roof is:

$$a_{f.bp.k} = \frac{2\cdot\left((H_{pr}-2\cdot\Delta_o)-\frac{B_{pr}-2\cdot\Delta_o}{3}\right)+1.33\cdot(B_{pr}-2\cdot\Delta_o)}{N_{bp.k}+1}, \text{ m} \quad (47)$$

for a rectangular and arch shape of mine workings; and

$$a_{f.bp.k} = \frac{2\cdot((H_{pr}-2\cdot\Delta_o)-0.5\cdot(B_{pr}-2\cdot\Delta_o))+0.5\cdot\pi\cdot(B_{pr}-2\cdot\Delta_o)}{N_{bp.k}+1}, \text{ m} \quad (48)$$

for an arch shape of mine workings.

The number of outline boreholes in a bottom:

$$N_{p.k} = \left((B_{pr} - 2\cdot\Delta_o)/a_{r.k}\right) + 1, \text{ pieces.} \quad (49)$$

The actual distance between the outline boreholes in the bottom:

$$a_{f.p.k} = (B_{pr} - 2\cdot\Delta_o)/(N_{p.k} - 1), \text{ m.} \quad (50)$$

Cut boreholes, breaking boreholes, and outline boreholes are arranged according to a scheme shown in Fig. 3.

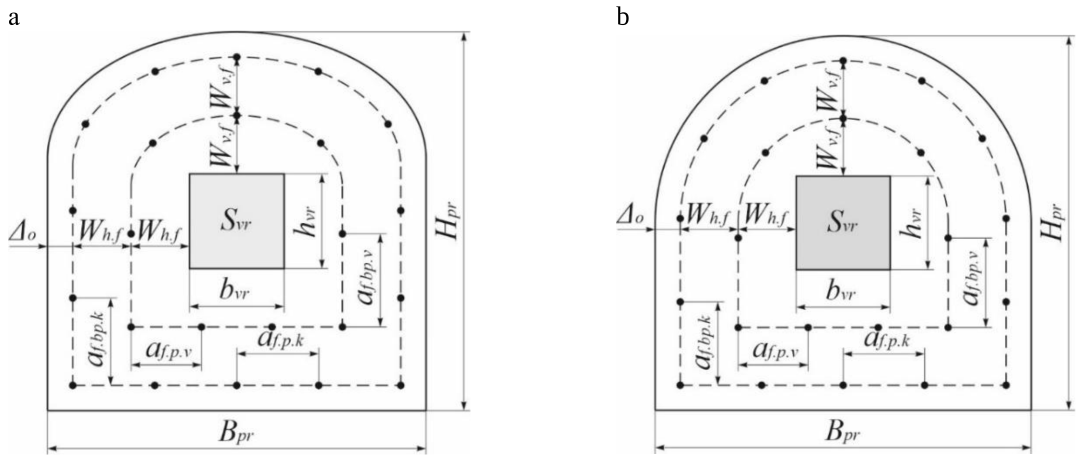

Fig. 3. Analytical scheme to arrange boreholes in terms of rectangular and arch shape of mine working (a), and in terms of the arch shape of mine working (b)

The total number of boreholes to be charged is:

$$N = N_{vr} + N_{vid} + N_{bp.k} + N_{p.k}, \text{ pieces} \quad (51)$$

where N_{vr} is the number of cut boreholes to be charged, pieces.

Specification of DBO parameters assumes an increase in the total amount of boreholes; however, the number cannot exceed 10%; in the mine workings, where crosscut is up to 5m^2, no more than 4 boreholes may be added.

The average charge amount per borehole is:

$$Q_{sh} = \frac{Q}{N}, \text{ kg.} \quad (52)$$

The charge amount in a borehole is:

$$Q_z = K_p \cdot Q_{sh}, \text{ kg} \tag{53}$$

where K_p is the coefficient involving the increase or decrease of E charge in a borehole depending upon its purpose, i.e. K_p = 1.1-1.2 for a cut borehole; K_p = 1.0 for a breaking borehole; K_p = 0.9-1.0 for an outline borehole from walls and roof; and K_p = 1.0-1.2 for an outline borehole of a bottom.

Actual E consumption per face:

$$Q_f = N_{vr} \cdot Q_{vr} + N_{vid} \cdot Q_{vid} + N_{bp.k} \cdot Q_{bp.k} + N_{p.k} \cdot Q_{p.k}, \text{ kg,} \tag{54}$$

where Q_{vr} is E charge in the cut borehole, kg; Q_{vid} is E charge in the breaking borehole, kg; $Q_{bp.k}$ is E charge in the outline borehole from walls and roof, kg; and $Q_{p.k}$ is E charge in the outline borehole in a bottom, kg.

Charge length of granulated E or emulsion E in the borehole, ignoring a primer, is:

$$l_z = \frac{Q_z}{0.785 \cdot d^2 \cdot \rho}, \text{ m.} \tag{55}$$

The actual specific E consumption is:

$$q_f = \frac{Q_f}{V_f} = \frac{Q_f}{S_{pr} \cdot l_{sh} \cdot \eta}, \ kg/m^3. \tag{56}$$

Testing the methods to calculate DBO parameters for driving a mine working using EE

The developed methods were tested in the *Prokhidnytska* mine of *ZZRK* PJSC to drive ort of 910 m level using emulsion E *Ukrayinit-PP-2* in terms of output data shown in Table 1.

Table 1 *Output data to calculate DBO parameters while driving a horizontal mine working in the context of ZZRK PJSC*

Name	Measurement unit	Index
Mine working	-	ort
Mine working width	m	3.80
Mine working height	m	3.65
Compressive ore strength	MPa	50 – 70
Ore density	kg/m^3	3950
Mine working depth	m	910
Fissility	-	average
Depth of borehole set	M	2.7
Borehole diameter	M	0.043
Cut	-	prismatic
Density of *Ukrayinit-PP-2* EE	kg/m^3	1250

Table 2 demonstrates the calculation results of DBO parameters for a horizontal mine working in the context of *ZZRK* PJSC according to the developed methods.

Table 2 *Calculation results of DBO parameters for a horizontal mine working in the context of ZZRK PJSC*

Name	Measurement units	Index
Coefficient of rock mass consolidation, K_u	c.u.	1.02
Coefficient of structural rock mass loosening, K_s	c.u.	0.80
Analytical specific E consumption, q	kg/m^3	2.82
Analytical E amount per mine working, Q	kg	97.62
Value of a crush zone radius, R_{zm}	m	0.35
Analytical LLR for a borehole, W	m	1.05
Area of a mine working crosscut in sinking, S_{pr}	m^2	12.81
Area of the outline boreholes, S_k	m^2	7.60
Cut area, S_{vr}	m^2	1.0
Face area for cut boreholes, S_{vid}	m^2	4.21

Corrected analytical LLR, $W_{sr.v}$	m	0.73
Number of cut boreholes to be charged, N_{vr}	pieces	8
Number of breaking boreholes, N_{vid}	pieces	11
Number of outline boreholes		
– from walls and roof, $N_{bp.k}$	pieces	10
– in a bottom, $N_{p.k}$	pieces	6
Total number of boreholes, N	pieces	36 + 1
Average charge amount per a borehole, Q_{sh}	kg	2.7
Charge amount		
– for a cut borehole, Q_{vr}	kg	3.3
– for a breaking borehole, Q_{vid}	kg	2.7
– for an outline borehole from walls and roof, $Q_{bp.k}$	kg	2.6
– for an outline bottom borehole, $Q_{p.k}$	kg	3.3
Actual E consumption per a face, Q_f	kg	104.5
Charge length without primer for		
– a cut borehole, $l_{z.vr}$	m	1.82
– a breaking borehole, $l_{z.vid}$	m	1.49
– an outline borehole from walls and roof, $l_{z.bp.k}$	m	1.43
– an outline bottom borehole, $l_{z.p.k}$	m	1.82
TS, η	-	0.85-0.95
Actual prognostic specific E consumption, q_f	kg /m^3	3.2-3.6

As a result of the calculated SBO parameters, the arrangement of boreholes in the face has been planned, as well as the cut design (see Fig. 4).

a b

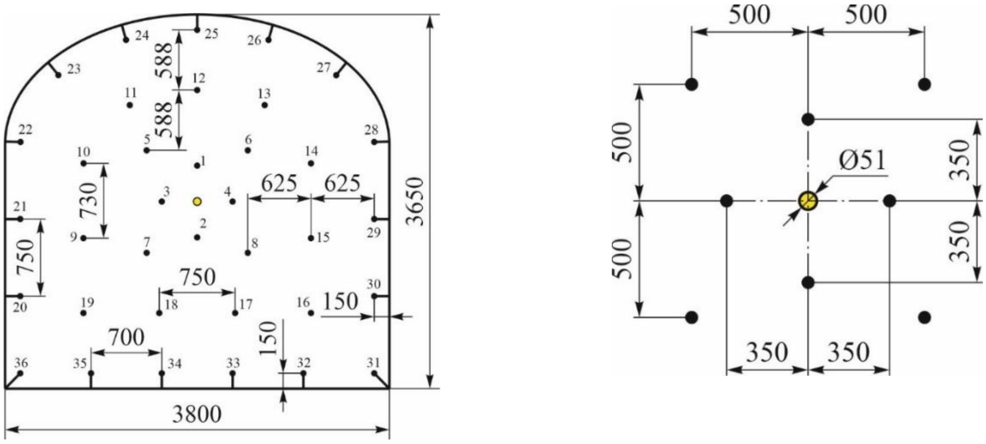

Fig. 4. Arrangement of the boreholes in a horizontal mine working face (a) and cut design (b)

Fig. 5 demonstrates the results of the experimental ort face explosion according to the calculated DBO parameters in the context of *ZZRK* PJSC.

a b

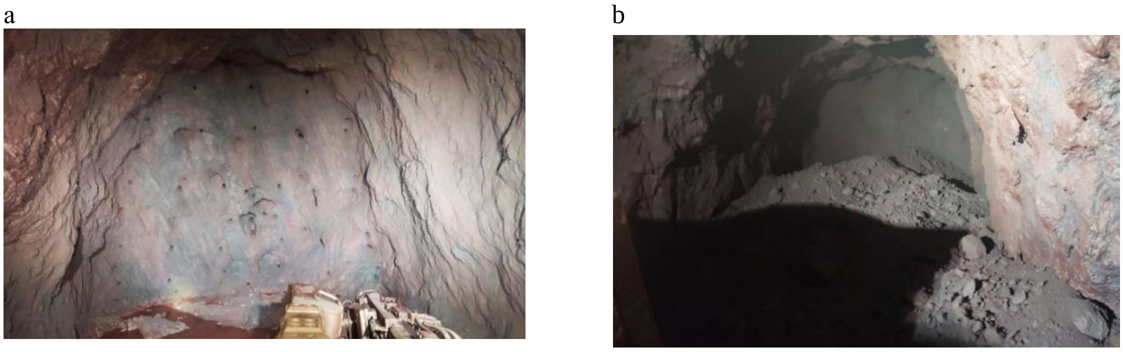

Fig. 5. Photofixation of the ort face of 910 m level before the explosion (a) and after it according to the calculated DBO parameters in the context of ZZRK PJSC.

As the photo of an ort face after blast (Fig. 5, b) demonstrates, the ore is crushed well and has an almost uniform granulometric composition with less than 0.2 m in size. However, there are several prills with 0.2-0.4 m size. Their amount is less than 5% of the total amount. No refusals of charges and bodies are observed. The efficiency coefficient of the boreholes is 0.95-0.97. Results of the explosions with the use of *Ukrayinit-PP-2* EE have helped conclude that the developed methods calculating DBO parameters while driving a mine working are suitable for the design of DBO parameters, and charge blasts demonstrate good results.

For the efficiency evaluation, the authors have compared the EE used in the mine conditions (Ukrayinit-PP-2 and Amonit # 6 ZhV). The efficiency determination while driving a mine working with the help of EE using formulas from paper (Kononenko et al., 2022) has made it possible to understand that the prime cost to sink 1m^3 of a mine working is influenced by E-type mining equipment as well as by DBO parameters. Analysis of prime cost values to drive 1 m^3 of a mine working with the use of various mining facilities has defined that if the packaged Ukrayinit-PP-2 EE is applied, then the prime driving cost is decreased by 11%. When Ukrayinit-PP-2 EE is applied, the decrease is up to 18% compared with the packaged Amonit # 6 ZhV (Амоніт №6 ЖВ) E.

Conclusions

Based on the above, the following conclusions were drawn:

1. Mathematical models of the radii of crush zones, intensive fragmentation zones, and crack formations zones, shaped in the rock mass around a charge in the process of explosion load, have been determined. The models involve wholistically a charge cavity diameter, detonation E characteristics, and rock strength in addition to the rock fissility and consolidation under the effect of rock pressure as well as the diameter of E charges itself.
2. Determination of the radii of rock mass deformation around a charge cavity has helped formulate innovative methods to calculate DBO parameters while driving horizontal and incline mining workings. The methods are based upon the idea of arranging groups of boreholes in terms of the areas they occupy in a mine working face as well as their placement depending upon the borehole outlines. Borehole LLR is identified according to the intensive fragmentation zone radius involving wholistically a crush area radius, diameters of boreholes and E charge, detonation E characteristics, compressive rock strength, fissility, and consolidation under the effect of rock pressure. The borehole arrangement parameters have actual calculated values as well as charge values for each borehole in the group.
3. The methods have been tested in the *Prokhidnytska* mine of *ZZRK* PJSC while ort driving at a 910 m level. *Ukrayinit-PP-2* EE has been applied, for which DBO parameters have been calculated relying upon the developed methods. The test shots have demonstrated good results and uniform ore fragmentation into prills whose size is less than 0.2 m (95%) and 0.2-0.4 m (5%). According to the explosions, the borehole efficiency turned out to be 0.95-0.97.
4. The use of the calculation methods for DBO parameters while driving a mine working makes it possible to rationalize the uniform arrangement of the boreholes, which will reduce the prime cost of 1 m^3 mine working sinking by 11% if the packaged Ukrayinit-PP-2 EE is applied. When Ukrayinit-PP-2 EE is applied, the decrease is up to 18% compared with the packaged Amonit # 6 ZhV E.

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