Raises advance using emulsion explosives

Maksym Kononenko[1*], *Oleh* Khomenko[2], *Andrii* Kosenko[3], *Inna* Myronova[4], *Vitaliy* Bash[1], and *Yuliya* Pazynich[5]

[1]Dnipro University of Technology, Department of Transport Systems and Energy-Mechanical Complexes, 19 Yavornytskoho Ave., 49005 Dnipro, Ukraine
[2]Dnipro University of Technology, Department of Mining Engineering and Education, 19 Yavornytskoho Ave., 49005 Dnipro, Ukraine
[3]M.S. Poliakov Institute of Geotechnical Mechanics of the NAS of Ukraine, Branch for Physics of Mining Processes, Department of Rock Mass Condition Control, 15 Simferopolska St., 49005 Dnipro, Ukraine
[4]Dnipro University of Technology, Department of Ecology and Environmental Protection Technologies, 19 Yavornytskoho Ave., 49005 Dnipro, Ukraine
[5]AGH University of Krakow, Faculty of Management, 30 Adama Mickiewicza Al., 30059 Krakow, Poland

Abstract. Using the well-known laws of the theory of elasticity and the basic principles of the quasi-static wave hypothesis of the mechanism of destruction of a solid medium by an explosion, methods have been developed for calculating the parameters of drilling and blasting (D&B) for raises advance using the methods of blast-hole and borehole charges. It has been established that the calculating D&B parameters is carried out in the same sequence as when drifting operation. To check the calculating D&B parameters using the new method during raise advance, a numerical simulation of changes in the stress-strain state of a rock mass under the influence of an explosion was carried out. According to the results of numerical simulation, the formation of zones of inelastic deformation in the face of a rising mine working under blast load, uniform grinding of the rock was obtained, which will avoid the release of oversized pieces after the explosion. The developed methodology was tested in the conditions of the "Yuvileina" mine of PJSC "Sukha Balka" during the raise advance of a 1420 m level using a sticked emulsion explosive (EE) Anemix P. Test explosions obtained good results in blasting the face of a raise, uniform crushing of the rock and a high coefficient of use of bore-holes has been established.

1 Introduction

Drivage preparatory and subordinate mining works within stope blocks or drawns stope for the extraction of ores using the underground mining method is one of the main and most labor-intensive production processes necessary to prepare for actual mining [1]. Technological development of various underground mining methods of ore deposits has led to the emergence of mine workings, which are characterized by a small cross-section and constitute the structural design of the mining methods [2]. In these mining methods, to first

---

* Corresponding author: kmn211179@gmail.com

workings for actual mining, raises of different sizes and shapes are used, performing various functions, as well as inclined or vertical draw holes. To first workings for actual mining, up to 50% of the total labor and material costs are spent on these mine workings [3]. Therefore, the development of a new methodology for calculating D&B parameters for raises advance is of current importance.

2 The choice of object to study

Depending on the purpose, raise mines are carried out through ore or rocks and are equipped with one, two or three compartments for minerals, rocks, staircases, etc. According to their purpose, rising mine workings are divided into running, vent-escape, transfer, fill, material and economic, cut and drill. The shape and cross-sectional dimensions of raises depend on their purpose, fastening material and the number of compartments. The most common are square, rectangular, and round cross-sectional shapes of raises with an area from 1.44 m^2 to 8.0 m^2. Each ore mine develops its own standard sections and dimensions of the raises, which are most appropriate to the specific conditions of mining. Today, there are two ways to carry out raise advances – drilling and blasting and machine (combine). The drilling and blasting method include: the hole-hole method with the equipment of temporary shelves and ladders and using tunneling complexes such as raisebore, and the sectional method with the detonation of deep borehole charges [4]. An analysis of scientific and technical sources has established that in the iron ore mines of the Kryvyi Rih basin, about 27 thousand m of raised mine workings pass annually [5], and in the conditions of the Private Joint Stock Company "Zaporizhzhia Ore Plant" (PJSC "ZOP") – up to 3 thousand m [6].

Using the blast-hole method with the equipment of temporary shelves in the mines of the Kryvyi Rih basin, raised mine workings are carried out in a square shape with an area of 1.44 m^2 and dimensions of 1.2 m × 1.2 m, 2.25 m^2 and dimensions of 1.5 m × 1.5 m, as well as a round shape with an area of 1.8 m^2 and a diameter 1.5 m. The height of these raises does not exceed 25 – 30 m. The main working processes in this method are drilling blast-hole, loading and blasting, ventilation, removing the rock mass and arranging shelves. This technology is characterized by a low level of safety, high labor intensity of operations, low monthly penetration rates, unfavorable working conditions, and huge labor costs [5]. However, it has advantages in simplicity, versatility, low weight of equipment and cost-effectiveness with a low height of the raised mine workings. In the mines of the Kryvyi Rih basin, the annual volume of raises with the equipment of temporary shelves and ladders is 6% for an area of 1.44 m^2, 73% for an area of 2.25 m^2. In the conditions of the PJSC "ZOP" mines, this method is not used due to technological limitations and features of the design elements of the underground mining method.

The use of self-propelled raisebore complexes for raises advance makes it possible to mechanize the delivery of drifters, equipment, and materials to the face of the mine workings. This method is used for raises advance square and rectangular shapes with an area of 3.2 m^2 and 4 m^2 with dimensions of 1.6 m × 2 m and 2 m × 2 m and a height of at least 60 – 80 m [4]. The tunneling cycle consists of the following work processes: removing the rock mass, drilling holes for anchors, building up a monorail, drilling blast-holes in the face, loading and blast-holes, and ventilating the face. However, the disadvantage of this method is that the miners carry out most of the working operations manually since they are constantly in the face of the raise [5]. In the mines of the Kryvyi Rih basin, the volume of raises advance with an area of 3.2 m^2 using tunneling complexes is 11%, and with an area of 4 m^2 – 10%. In the mines of PJSC "ZOP", this method is no longer used.

Also, another way to raises advance using D&B devices is sectional blasting of deep borehole charges. This method assumes that there are no people in the face of the raise. All

work on drilling, charging and blasting boreholes is carried out from adjacent horizontal mine workings. To raises advance using this method, boreholes with a diameter of 0.089 – 0.105 m are drilled from the upper mine workings to the full height, with a distance between them of 0.5 – 0.9 m [4]. A raise is formed by sequentially blasting individual sections 2 – 4 m long in the direction from bottom to top. This method can be used to make raises advance up to 40 m high, which is due to the curvature of the boreholes during drilling. Before charging, the lower part of the boreholes is covered with wooden conical plugs, which are lowered into the well using twine or wire, and loading occurs from above from the mouth side. A layer of rock or sand is poured above the charge to a height of 0.5 – 1.0 m, used as a plug. This method is used for cutting raises, ventilating raises and gravity ore passes that do not require fastening. Today, in the mines of the Kryvyi Rih basin, raises advance by sectional blasting of deep boreholes is not used for various reasons, but in the conditions of PJSC "ZOP", the above method is used to drifting out up to 72% of the total length of raises with an area of 4 m^2 and 6 m^2.

In the conditions of the mines of the Kryvyi Rih basin, 3.3% of raises advance out using a machine (combine) method, and in PJSC "ZOP" – 28% of their total length. The cross-sectional area of raises advance from 2.5 m^2 to 4.5 m^2. This method is competitive with D&B only when raises advance with a height of more than 80 m. Usually drilling is raises advance between two concentration levels. The most common technology is drilling a pilot hole with a diameter of up to 0.3 m to the full height along the axis of a raise, followed by expansion from bottom to top to the design diameter. First workings begin with selecting a location for drilling a pilot hole and drilling a chamber with a volume of 60 – 140 m^3 to accommodate the combine and erect a foundation for it at the wellhead. The basis is a concrete platform (foundation), on which the paws are attached to the combine frame with collet bolts. The pilot hole is drilled with a roller bit using guide rods that prevent the pilot hole from deviating from the given direction. After the pilot hole reaches the lower level, the roller bit is removed, and the drilling rod is equipped with an expander of a given end diameter of the raise. Raise advance with a height of up to 80 m using a combine is considered impractical due to the high cost of manual labor in the construction of drilling chambers and concrete foundations, the high labor intensity of installation, dismantling and transportation of combines. The disadvantages of this method are the high cost of combines and rock-destroying tools, their dimensions and large weight [4].

An analysis of the technology for raises advance has established that in the conditions of the mines of the Kryvyi Rih basin, up to 97% of their total length is carried out using D&B, and in the conditions of the mines of PJSC "ZOP" – up to 72%. This indicates that the use of self-propelled complexes and combines for raises advance cannot completely solve the problem of increasing the efficiency of stope blocks for actual mining. The time for stope blocks and putting them into operation largely depends on the rate of penetration of raises advance. The length of the raise, which are carried out at the mines of the Kryvyi Rih basin and PJSC "ZOP" when stope blocks or cuts for actual mining, together with low productivity and difficult working conditions during their excavation, require the search for modern technological and technical solutions to increase efficiency their buildings. This is possible thanks to the development and determination of rational parameters of D&B for the raises advance using methods of blast-hole and borehole charges, as well as the use of domestically produced emulsion explosives (EE) [7], which are safe in transportation [8] and storage [9], environmentally friendly [10] and economically beneficial [11].

3 Research methods

The development of methods for determining D&B parameters for raises included the following steps:

– design and sequence of calculation of straight cuts along the radius of the crumple zone;

– development of methods for calculating D&B parameters for raises advance using the methods of blast-hole and borehole charges;

– verification of D&B parameters calculated using a new method for real conditions using numerical simulation of the stress-strain state of a rock mass in the face of a raise under a blast load;

– approbation of the methodology for calculating D&B parameters for carrying out raise using EE.

To develop methods for calculating D&B parameters when carrying out raising mine workings using the methods of blast-hole and borehole charges, we used the well-known laws of the theory of elasticity and the basic provisions of the quasi-static wave hypothesis of the mechanism of destruction of a solid medium by an explosion. To determine zones of inelastic deformation, because of changes in the stress-strain state of the rock mass in the face of a raise under the influence of an explosion, numerical simulation was carried out using the finite element method in a licensed software product for engineering analysis SolidWorks Simulation. Approbation of the calculating D&B parameters using the developed methodology for carrying out raising mine workings, using cartridgeed EE, was carried out under the conditions of a real object.

4 Research results

The calculating D&B parameters for raises advance using the blast-hole charge method is carried out in the same sequence as when drifting out horizontal mine workings [12] but differs in that when drifting out raises advance there is no group of outside holes. This is because the area of raises is of little importance. Therefore, in some cut designs, auxiliary drill blast-holes are used to expand its area. From practical experience, when raises advance using the blast-hole charge method, straight prismatic cuts are used. According to the results of studies given in [13, 14], the punching distance between blast-holes in a straight cut is equal to the radius of the crush zone and is determined by the formula

$$R_{zm} = 0.5 \cdot d \cdot \sqrt{1 + \frac{\rho \cdot D^2 \cdot K_{dz}}{2 \cdot \sigma_{st} \cdot K_{sp} \cdot K_s \cdot K_u}} \text{ , m,} \tag{1}$$

where d is the blast-hole or borehole diameter, m; ρ is the explosive density (E) or EE density [15], kg/m³; D is the E detonation velocity, m/s; σ_{st} is the compressive strength of rocks, Pa; K_{dz} is the coefficient taking into account the change in the pressure of explosion products on the walls of the charging cavity depending on the diameter of the E charge [14]; K_{sp} is the rock structure coefficient, depending on the properties of the massif [14]: for viscous, elastic and porous rocks $K_{sp} = 2.0$, for dislocated rocks, with varying occurrence and fine fracturing $K_{sp} = 1.4$, for shale rocks, with varying strength and layering perpendicular to the direction of the charge cavities.$K_{sp} = 1.3$, for massive, brittle $K_{sp} = 1.1$, for monolithic $K_{sp} = 1.0$, for monolithic $K_{sp} = 0.8$; K_s is the coefficient of structural weakening of the array, determined by one of the formulas provided in [13, 16]; K_u is the empirical coefficient of rock compaction under the influence of rock pressure [14].

$$N_{vr} = \frac{0.144 \cdot (l_{sh} \cdot \eta)^{0.91}}{R_{zm}}, \tag{2}$$

where l_{sh} is the length of the set of blast-holes equal to 1.3 – 2.2 m.

Estimated specific expenditures of E are determined using the most universal formula of Pokrovsky M.M.

$$q = 0.01 \cdot \sigma_{st} \cdot K_{sp} \cdot k_{zat} \cdot e \text{, kg/m}^3, \tag{3}$$

where σ_{st} is the compressive strength of rocks, MPa; e is the coefficient of relative explosive performance is calculated according to the methodology presented in [17]; k_{zat} is the rock clamping coefficient, varying within 1.2 – 1.5 or calculated by the expression

$$k_{zat} = 3 \cdot l_{sh} / \sqrt{S_{pr}} \text{ ,} \tag{4}$$

where S_{pr} is the overall cross-section area of the raise, m^2.

The volume of blasted rock in the massif is

$$V = S_{pr} \cdot l_{sh} \text{, m}^3. \tag{5}$$

The estimated amount of E per face is

$$Q = q \cdot V \text{, kg.} \tag{6}$$

Taking into account the operating conditions of the blast-hole charges and their location relative to the open surface, as well as when drifting out horizontal mineworkings, the line of least resistance (LLR) for a hole is equal to the radius of the intensive grinding zone and is determined by the formula [14]

$$W = R_{zm} \cdot \sqrt{1 + \frac{\rho \cdot D^2 \cdot d \cdot K_{dz}}{8 \cdot R_{zm} \cdot \sigma_{st} \cdot K_{sp} \cdot K_s \cdot K_u}} \text{ , m.} \tag{7}$$

Number of cropper holes
– across the width of the raise rectangular and square shape

$$N_b = \left((B_{pr} - 2 \cdot \Delta_o) / W \right) + 1 \text{;} \tag{8}$$

– along the length of the raise rectangular and square shape

$$N_h = \left((H_{pr} - 2 \cdot \Delta_o) / W \right) - 1 \text{;} \tag{9}$$

– for raise round shape

$$N_k = \left(2 \cdot \pi \cdot (0.5 \cdot D_{pr} - \Delta_o) \right) / W \text{ .} \tag{10}$$

where B_{pr} is the width of the raise in the drifting, m; H_{pr} is the length of the raise in the drifting, m; Δ_o is the distance from the excavation contours to the line of cropper holes, equal to the radius of the crush zone R_{zm}, according to practical experience, this distance is taken within 0.15 – 0.25 m; D_{pr} is the diameter of the raise in the drifting, sq.m.

The total number of cropper holes for raise rectangular and square shapes is

$$N_k = 2 \cdot (N_b + N_h - 2) \text{ .} \tag{11}$$

Actual distance between cropper holes
– across the width of the raise rectangular and square shape

$$a_b = (B_{pr} - 2 \cdot \Delta_o) / (N_b - 1) \text{ , m;} \tag{12}$$

– along the length of the raise rectangular and square shape

$$a_h = \left(H_{pr} - 2 \cdot \Delta_o \right) / \left(N_h + 1 \right), \text{ m;} \tag{13}$$

– for raise round shape

$$a = \sin\left(180 / N_k \right) \cdot \left(D_{pr} - 2 \cdot \Delta_o \right), \text{ m.} \tag{14}$$

Cutholes and cropper holes in the face of the raise are located according to the calculation scheme presented in Fig. 1.

The total number of blast-holes in the battle of the insurgent will be

$$N = N_{vr} + N_k. \tag{15}$$

The average value of the charge per blast-hole will be

$$Q_{sh} = Q / N, \text{ kg.} \tag{16}$$

The value of the charge in the cuthole and cropper hole is found by the expression

$$Q_z = K_p \cdot Q_{sh}, \text{ kg,} \tag{17}$$

where K_p is the coefficient that takes into account the increase or decrease of the explosive charge in the blast-hole, depending on its purpose: for a cuthole $K_p = 1.1 - 1.2$, for a cropper hole $K_p = 0.9 - 1.0$.
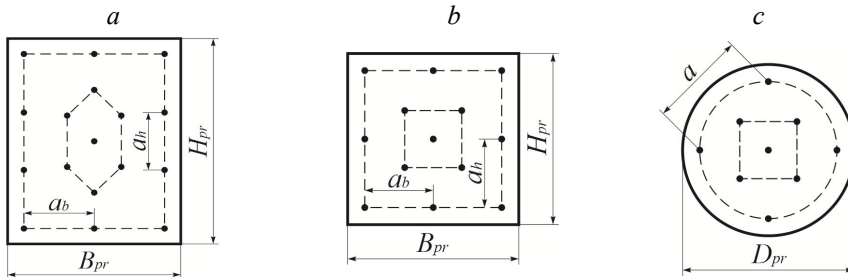

Fig. 1. Design diagram for the location of blast-holes in the face of a raise of rectangular (a), square (b) and round (c) shapes.

When using a sticked E, the charge value is adjusted taking into account the mass of the E sticked.

The actual costs of E for face are

$$Q_f = N_{vr} \cdot Q_{vr} + N_k \cdot Q_k, \text{ kg,} \tag{18}$$

where Q_{vr} is the amount of E charge in the cuthole, kg; Q_k is the amount of explosive charge in the cropper hole, kg.

The length of the charge of placer E or liquid EE in the blast-hole without taking into account the priming cartridge

$$l_z = Q_z / (0.785 \cdot d^2 \cdot \rho), \text{ m.} \tag{19}$$

Actual specific consumption of explosives

$$q_f = \frac{Q_f}{S_{pr} \cdot l_{sh} \cdot \eta}, \text{ kg/m}^3. \tag{20}$$

where η is the coefficient of use of blast-holes (CUB), is equal to 0.85 – 0.95.

It is proposed to calculating D&B operations parameters for raises advance method of sectional blasting of boreholes in the following sequence. Having carried out an analysis of standard D&B drilling passports, as well as the technology of raises advance method of sectional blasting of boreholes in the conditions of the mines of PJSC "ZOP", an empirical dependence was established on the minimum number of boreholes in a cut depending on the compressive strength of rocks or ore.

$$N_{vr} = 0.5 \cdot k_s \cdot \sqrt{\sigma_{st}}, \tag{21}$$

where k_s is the coefficient taking into account the shape and cross-sectional area: for round arises when $S_{pr} = 2.5 \text{ m}^2$ and 3.8 m^2 $k_s = 0.2$, for rising square and rectangular shapes when $S_{pr} = 1.7 \text{ m}^2$ and 2.25 m^2 $k_s = 0.2$, when $S_{pr} = 4 \text{ m}^2$ and 6 m^2 $k_s = 1.0$.

According to research results [13, 14], the punching distance between boreholes in a cut is equal to the radius of the crush zone, determined by formula (1). Due to the small cross-sectional areas, both when raises advance method of blasthole charges, and when sectional blasting boreholes, there will be no group of outside holes. Taking into account the operating conditions of borehole charges in a compressed environment and their location relative to the open surface of the LLR for a borehole, it will be equal to the radius of the zone of intensive grinding, determined by formula (7).

Number of cropper borholes
– across the width of the raise rectangular and square shape

$$N_b = \left(B_{pr} / W \right) + 1 ; \tag{22}$$

– along the length of the raise rectangular and square shape

$$N_h = \left(H_{pr} / W \right) - 1 ; \tag{23}$$

– for raise round shape

$$N_k = \left(\pi \cdot D_{pr} \right) / W . \tag{24}$$

The total number of cropper boreholes for raise rectangular and square shapes is determined by the formula (11).

Actual distance between cropper borholes
– across the width of the raise rectangular and square shape

$$a_b = B_{pr} / \left(N_b - 1 \right), \text{ m}; \tag{25}$$

– along the length of the raise rectangular and square shape

$$a_h = H_{pr} / \left(N_h + 1 \right), \text{ m}; \tag{26}$$

– for raise round shape

$$a = \sin \left(180 / N_k \right) \cdot D_{pr}, \text{ m}. \tag{27}$$

Cutholes and cropper boreholes in the raise face are located according to the design diagram presented in Fig. 2.
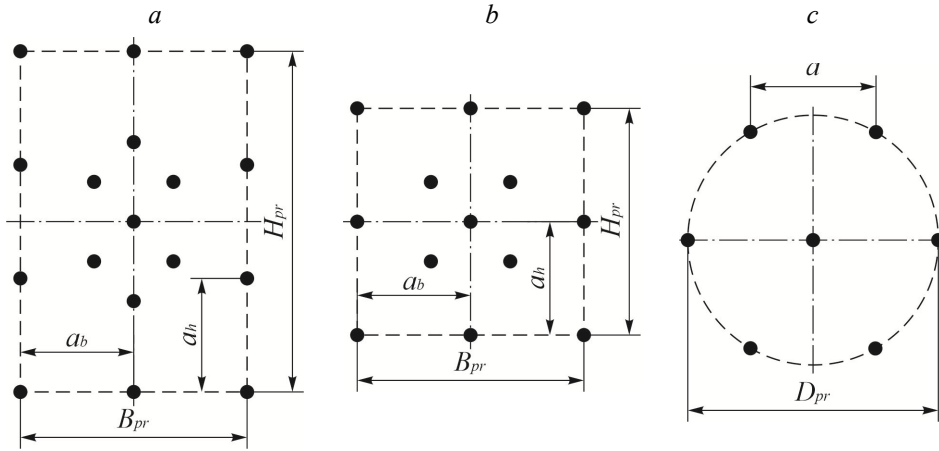
Fig. 2. Design diagram for the location of boreholes in the face of a raise of rectangular (a), square (b) and round (c) shapes.

The total number of boreholes in the bottom of the riser is determined by formula (15). Charge amount per borehole
– for placer E or liquid EE

$$Q_s = 0.25 \cdot \pi \cdot d^2 \cdot (l_s - l_{zab} - l_{pb}) \cdot \rho \text{ , kg;} \tag{28}$$

– for sticked E

$$Q_s = n_{pat} \cdot m_{pat} \text{ , kg,} \tag{29}$$

where l_c is the length of the section, which is equal to $2 - 4$ m; l_{zab} is the driving length equal to 1 m; m_{pat} is the mass of sticked E, kg;; n_{pat} is the number of sticked E, taking into account the priming cartridge

$$n_{pat} = (l_s - l_{zab}) / l_{pat} , \tag{30}$$

where l_{pat} is the Length of sticked E, m.
The resulting number of sticked E cartridges is rounded to the nearest whole number.
Actual consumption of explosives per slaughter

$$Q_f = N \cdot Q_s \text{ , kg.} \tag{31}$$

Actual specific consumption of E

$$q_f = Q_f / \left(S_{pr} \cdot l_s \right) \text{, kg/m}^3. \tag{32}$$

As an example, let us consider the calculating D&B parameters when raises advance in the conditions of the "Yuvileina" mine of PJSC "Sukha Balka" according to the following initial data given in Table 1.
The results of calculating D&B parameters when raises advance in the conditions of the "Yuvileina" mine PJSC "Sukha Balka" according to the developed methodology are given in Table. 2.

Table 1. Initial data for calculating D&B parameters when raises advance in the conditions of the "Yuvileina" mine PJSC "Sukha Balka".

Naming	Units of measurement	Meaning
Mine width	m	1.6
Mine width	m	1.8
Mine height	m	54
Cross-sectional area of the mine working	m^2	2.88
Rock compressive strength	MPa	90 – 100
Poisson's ratio	–	0.25
Rock Density	kg/m^3	3200
Depth of mine working	m	1420
Rock fracturing	–	Cracked
Depth of cutholes	m	1.5
Blast-hole set depth	m	1.3
Blast-hole diameter	m	0.04
Cut	–	Prismatic
Density of sticked EE Anemix P	kg/m^3	1140
Detonation velocity of sticked EE Anemix P	m/s	5500

Table 2. Results of calculating D&B parameters when raises advance in the conditions of the "Yuvileina" mine PJSC "Sukha Balka".

Naming	Units of measurement	Meaning
Rock structure coefficient, K_{sp}	fractions of units	1.4
Structural weakening coefficient of the array, K_s	fractions of units	0.4
Array compaction coefficient, K_u	fractions of units	1.04
Estimated specific consumption of E, q	kg/m^3	3.55
Estimated quantity of explosives per face, Q	kg	11.83
The radius of the crush zone, R_{zm}	m	0.25
Design breaker LLR for hole, W	m	0.66
Number of cutholes, N_{vr}	units	5
Number of cropper holes, N_k	units	8
Total number of blast-holes, N	units	13
Actual distance between cropper holes along the width of the raise, a_b, m	m	0.65
Actual distance between cropper holes length, a_h, m	m	0.75
Average charge per blast-hole, Q_{sh}	kg	0.91
Diameter of sticked EE Anemix P	m	0.032
Mass of sticked EE Anemix P	kg	0.20
Length of EE Anemix P	m	0.24
Charge amount:		
– for cuthole, Q_{vr}	kg	1.2
– for cropper hole, Q_k	kg	1.0
Actual consumption of E at face, Q_f	kg	14
Charge length from a priming cartridge:		
– cuthole, $l_{z.vr}$	m	1.4
– cropper hole, $l_{z.k}$	m	1.2
CUB, η	fractions of units	0.9 – 0.95
Actual forecast specific losses of E, q_f	kg/m^3	4.15 – 3.94

Based on the results of calculating the D&B parameters for the recovery, the location of the blast-holes in the face of the mine working was drawn, shown in Fig. 3.
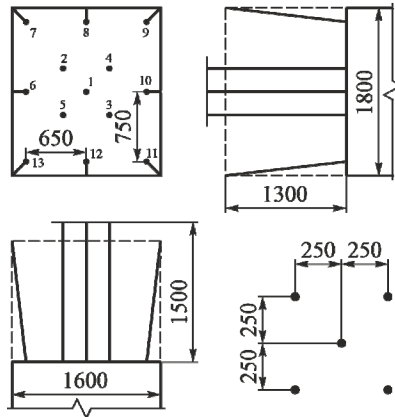
Fig. 3. The location of the blast-holes in the face of the raise and the design of the cut.

Comparing the developed D&B passport with the current passport for raise advance in the conditions of the "Yuvileina" mine of PJSC "Sukha Balka", it was found that the calculating D&B parameters using the developed method will make it possible to reduce the number of blast-holes in the face of the riser by 24% out of 17 pcs. up to 13 pcs. This indicates that the proposed method for calculating D&B parameters for blasting considers not only the physical and mechanical properties of the rock mass, but also the detonation characteristics of explosives, which makes it possible to obtain correct results.

To check the calculating D&B parameters using the new method during raise advance, it is necessary to conduct a numerical simulation of the change in the stress-strain state of the rock mass under the influence of an explosion. Today, one of the most effective methods for studying the mechanism of explosive destruction of rocks is the use of numerical simulation by the finite element method (FEM) [18, 19], which is one of the most developed methods for simulation phenomena and processes with maximum approximation to reality [20, 21]. The use of numerical modeling makes it possible to study processes and phenomena that are impossible or economically impractical to study under practical conditions [22 – 24]. Despite the development of mathematical and computer technologies, there is a limited number of software products that can simulate the explosion process with high accuracy. The leaders in this area are the software products SolidWorks [24] and ANSYS [25], which use FEM and are widely used in mining to simulation changes in the stress-strain state of rocks under the influence of rock pressure [26] and explosion.

According to the theory of strength of materials [27], dynamic calculations can be replaced by static ones by considering the dynamic coefficient. Thus, to simulate the change in the stress-strain state of a rock mass during its explosive destruction, the licensed software of the SolidWorks Simulation engineering analysis system, available at the Dnipro University of Technology, was used. Compared to ANSYS, using SolidWorks Simulation to solve static problems reduces the time required to complete a single experiment. In addition, working with SolidWorks Simulation is easier than working with the highly advanced ANSYS package because it requires fewer approaches to defining the problem.

Therefore, the goal of numerical simulation is to determine zones of inelastic deformation due to changes in the stress-strain state of the rock mass in the face of a raise under the influence of an explosion. The numerical simulation technique using the finite element method in the licensed software for engineering analysis SolidWorks Simulation included the following steps:
   – creation of model geometry;
   – determination of physical and mechanical properties of the model material;

– setting the initial load, constraints and model mesh;

– conducting a computational experiment and processing the results.

At the first stage of simulation, the geometry of the model was created. Since the raise has dimensions of 1.6 m × 1.8 m, to exclude edge effects on the modeling results, the following model geometry parameters were adopted: width and height – 9.4 m, length – 7.5 m.

At the next stage of modeling, we determined the basic physical and mechanical properties of the model mass necessary for numerical simulation of the stress-strain state of rocks under the influence of an explosion: elastic modulus, Poisson's ratio, shear modulus, density, tensile and compressive strength. Considering the structural structure of rocks, fracturing and compaction under the influence of rock pressure, the compressive strength of the model material is

$$\sigma_{st}^{m} = \sigma_{st} \cdot K_{sp} \cdot K_{s} \cdot K_{u}, \text{MPa.} \tag{33}$$

The accepted physical and mechanical properties of the model material are presented in Table 3.

Table 3. Physical and mechanical properties of the model material.

Naming	Meaning
Elastic modulus, Pa	$5.82 \cdot 10^{10}$
Poisson's constant	0.25
Shear modulus, Pa	$2.328 \cdot 10^{10}$
Density, kg/m^3	3200
Tensile strength, $\times 10^6$ Pa	5820000
Compressive resistance, $\times 10^6$ Pa	58200000

The accuracy of the results of numerical simulation of the destruction of a rock mass by an explosion depends on the correctness of the specified conditions. Since an explosion is an impact, and then the action of pressure from the explosion products [28], to simulate this process, pressure was created perpendicular to the faces and bottom in all holes, considering the dynamic coefficient. The pressure value of explosion products in holes [29] for simulation the blasting of a raise face, taking into account the diameter of the charge, is determined by the expression

$$P_m = \frac{\rho \cdot D^2 \cdot K_{dz} \cdot K_d}{8}, \text{Pa,} \tag{34}$$

where K_d is the dynamic coefficient.

A computational experiment to determine changes in the stress-strain state of rocks in a raise face under the influence of an explosion was carried out in the following sequence. First, the geometry of the model was crossed out. A new statistical analysis was selected in the SolidWorks Simulation environment. Then the model material with physical and mechanical properties given in table was adopted. The Kulona-Mora criterion, used for materials with different tensile and compressive properties, was used as a rock failure criterion. After which the geometry of the model was fixed, the pressure of the explosion products was applied in the blast-holes along the entire length. To exclude the influence of the magnitude of the undercharge on the experimental results, it was not considered. Next, a model mesh was created based on curvature with a high density. The computational program for the current study was then run. Next, the results were debugged in the parameters. Diagrams of the deviator of the main stresses and the zone of inelastic deformations arising in the rock mass in the raise face of the uprising under the influence of an explosion are presented in Fig. 4.
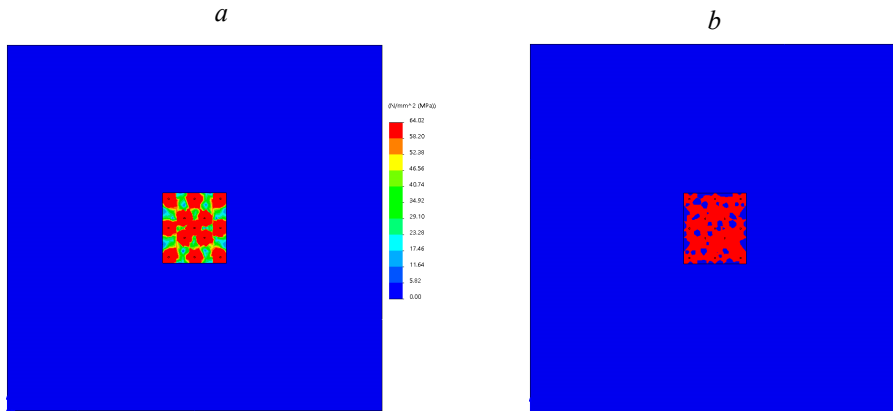
a   *b*

Fig. 4. Plots of the deviator of principal stresses (a) and zones of inelastic deformations (b) formed of a rock mass in the face of a raise under a blast load.

The places in the raise face of the uprising, where the diagrams of the deviator of the main stresses exceed the tensile strength of the model material [30], outline the areas of rock destruction under the influence of an explosion (Fig. 4a). This is confirmed by the red areas around the blast-holes in the raise face of the uplift, where the rock has a strength factor ≤ 1 (Fig. 4b), which means the formation of zones of inelastic deformation. The analysis of the results of numerical modeling of changes in the stress-strain state of the model material in the raise face under the influence of an explosion showed that the predicted granulometric composition of the rock after blasting will be < 0.2 m. This indicates that the use of the calculating D&B parameters according to the proposed methodology for raise in given mining and geological conditions will avoid the release of oversized pieces of rock after the explosion.

To confirm the obtained results of numerical simulation of D&B parameters for the raise, calculated using the developed methodology, a test explosion was carried out under natural conditions at a real object. Therefore, as an example, the results of a test explosion of the face of a rising horizon of 1420 m in highly fractured goethite-hematite quartzite with a compressive strength of 90 – 100 MPa are given according to the calculating drilling-and-blasting operations parameters in the conditions of the "Yuvileina" mine of PJSC "Sukha Balka" (Fig. 5).

Fig. 5. Photo recording of the face of a raise of a 1420 m after the explosion according to the calculating D&B parameters device in the conditions of the "Yuvileina" mine of PJSC "Sukha Balka".

As can be seen from the photo of the face of the raise after the explosion (Fig. 5), the goethite-hematite quartzite is well crushed and, according to actual measurements, has an almost uniform granulometric composition with a size of less than 0.2 m. Failure of charges in the face has not been established. The coefficient of CUB was 0.95 – 0.97. Based on the results of explosions carried out in raise using a cartridgeed EE Anemix P, it can be argued that the developed methodology for calculating D&B parameters when raises advance is suitable for calculating D&B parameters, and detonation of charges in the face shows good results.

5 Conclusions

Based on the radii of the deformation zones of the rock mass around the charging cavity, fundamentally new methods for calculating D&B parameters techniques have been developed for raises advance using the methods of blast-hole and borehole charges. Due to the small cross-sectional areas, both when Raises advance using the blast-hole charge method, and when sectional blasting of boreholes, there will be no group of outside holes. Considering the operating conditions of blast-hole and borehole charges in a clamped environment and their location relative to the open surface of the line of least resistance for a blast-hole or borehole, it is proposed to determine the intensive crushing zone by the radius of the zone. When determining the line of least resistance of a blast-hole or borehole along the radius of the zone of intensive grinding, the radius of the crushing zone, the diameters of the charging cavity and the E charge, the detonation characteristics of the E, the compressive strength of rocks, their structural structure, fracturing and compaction under the influence of rock pressure are comprehensively taken into account. The location parameters of blast-holes or boreholes have actually calculated values, as well as the amount of charge for each blast-hole or borehole in the group.

To check the calculating D&B parameters using the new method during raise, numerical simulation of changes in the stress-strain state of a rock mass under the influence of an explosion was carried out. By analyzing the results of numerical simulation of the formation of zones of inelastic deformation in the face of a raise under the influence of an explosion, it was established that the predicted granulometric composition of the rock after blasting will be <0.2 m. This indicates that the use of the calculating D&B parameters using a new technique for raises advance in given mining and geological conditions will avoid the release of oversized pieces of rock after an explosion. The use of the proposed D&B parameters when raises advance in the conditions of the "Yuvileina" mine of PJSC "Sukha Balka" makes it possible to rationalize the uniform arrangement of blast-holes in the face, which will lead to a reduction in their number by 24% from 17 pieces up to 13 pieces.

The developed methodology was tested in the conditions of the "Yuvileina" mine of PJSC "Sukha Balka" during the excavation of a raise of a level of 1420 m using an Anemix P sticked EE, for which calculating D&B parameters using the proposed method and numerical simulation was carried out. Test blasts showed good results in blasting the face of a raise and uniform crushing of goethite-hematite quartzite to a piece less than 0.2 m in size. Based on the blasting results, the CUB was established at 0.95 – 0.97.

Based on the simulation and test explosion, it was established that the developed methods are suitable for calculating D&B parameters for raises advance using the methods of blast-hole and borehole charges.

References

1. Stupnik, M., Kalinichenko, O., Kalinichenko, V., Pysmennyi, S., & Morhun, O. (2018). Choice and substantiation of stable crown shapes in deep-level iron ore mining. *Mining of Mineral Deposits*, *12*(4), 56-62. https://doi.org/10.15407/mining12.04.056

2. Pysmennyi, S., Fedko, M., Chukharev, S., Sakhno, I., Moraru, R., & Panayotov, V. (2023). Enhancement of the rock mass quality in underground iron ore mining through application of resource-saving technologies. *IOP Conference Series: Earth and Environmental Science*, *1156*(1), 012029. https://doi.org/10.1088/1755-1315/1156/1/012029

3. Pysmenniy, S., Shvager, N., Shepel, O. Kovbyk, K., & Dolgikh, O. (2020). Development of resource-saving technology when mining ore bodies by blocks under rock pressure. *E3S Web of Conferences*, (166), 02006. https://doi.org/10.1051/e3sconf/202016602006

4. Khomenko, O., Kononenko, M., & Lyashenko, V. (2021). Improvement of safety for the underground vertical workings. *Occupational Safety in Industry*, (2), 41-48. http://doi.org/10.24000/0409-2961-2021-2-41-48

5. Babec, E.K., Lyash, S.I., Chepurnoi, V.I., & Dobrovol'skaya, Z.S. (2012). Analysis of the State of Raise Drivings at Krivbas Mines. *Metallurgical and Mining Industry*, *4*(4), 6-8.

6. Shvyd'ko, P.V., Usatyj, V.V., Maslov, V.A., Usatyj, V.Yu., Furman, A.I., & Postoyalko, A.V. (2001). Experience of imported mining equipment application and explosives using in exploitating of Yuzhneu-Belozerskey iron-ore deposit. *Metallurgicheskaya i Gornorudnaya Promyshlennost*, (4), 67-70.

7. Krysin, R.S., Ishchenko, N.I., Klimenko, V.A., Piven, V.A., & Kuprin, V.P. (2004). Explosive ukranit-PM-1: Equipment and fabrication technology. *Gornyi Zhurnal*, (8), 32-37.

8. Lyashenko, V., Vorob'ev, A., Nebohin, V., & Vorob'ev, K. (2018). Improving the efficiency of blasting operations in mines with the help of emulsion explosives. *Mining of Mineral Deposits*, *12*(1), 95-102. https://doi.org/10.15407/mining12.01.095

9. Kholodenko, T., Ustimenko, Y., Pidkamenna, L., & Pavlychenko, A. (2014). Ecological safety of emulsion explosives use at mining enterprises. *Progressive Technologies of Coal, Coalbed Methane, and Ores Mining*, 255-260. http://doi.org/10.1201/b17547-45

10. Khomenko, O., Kononenko, M., & Myronova, I. (2017). Ecological and technological aspects of iron-ore underground mining. *Mining of Mineral Deposits*, *11*(2), 59-67. https://doi.org/10.15407/mining11.02.059

11. Kholodenko, T., Ustimenko, Y., Pidkamenna, L., & Pavlychenko, A. (2015). Technical, economic and environmental aspects of the use of emulsion explosives by ERA brand in underground and surface mining. *New Developments in Mining Engineering 2015: Theoretical and Practical Solutions of Mineral Resources Mining*, 211-219. http://doi.org/10.1201/b19901-38

12. Kononenko, M., Khomenko, O., Cabana, E., Mirek, A., Dyczko, A., Prostański, D., & Dychkovskyi, R. (2023). Using the methods to calculate parameters of drilling and + blasting operations for emulsion explosives. *Acta Montanistica Slovaca*, *28*(3), 655-667. https://doi.org/10.46544/ams.v28i3.10

13. Kononenko, M., & Khomenko, O. (2021). New theory for the rock mass destruction by blasting. *Mining of Mineral Deposits*, *15*(2), 111-123. https://doi.org/10.33271/mining15.02.111

14. Kononenko, M., Khomenko, O., Sadovenko, I., Sobolev, V., Pazynich, Yu., & Smolinski, A. (2023). Managing the rock mass destruction under the explosion. *Journal of sustainable mining*, *22*(3), 240-247. https://doi.org/10.46873/2300-3960.1391

15. Kononenko, M., Khomenko, O., Kovalenko, I., & Savchenko, M. (2021). Control of density and velocity of emulsion explosives detonation for ore breaking. *Naukovyi Visnyk Natsionalnoho Hirnychoho Universytetu*, (2), 69-75. https://doi.org/10.33271/nvngu/2021-2/069

16. Babets, D., Sdvyzhkova, O., Shashenko, O., Kravchenko, K., & Cabana, E. C. (2019). Implementation of probabilistic approach to rock mass strength estimation while excavating through fault zones. *Mining of Mineral Deposits*, *13*(4), 72-83. https://doi.org/10.33271/mining13.04.072

17. Kononenko, M., Khomenko, O., Kovalenko, I., Kosenko, A., Zagorodnii, R., Dychkovskyi, R. (2023). Determining the performance of explosives for blasting management. *Rudarsko-Geološko-Naftni Zbornik*, *38*(3), 19-28. https://doi.org/10.17794/rgn.2023.3.2

18. Babets, D., Sdvyzhkova, O., Hapieiev, S., Shashenko, O., & Prykhodchenko, V. (2023). Multifactorial analysis of a gateroad stability at goaf interface during longwall coal mining – A case study. *Mining of Mineral Deposits*, *17*(2), 9-19. https://doi.org/10.33271/mining17.02.009

19. Lapčević, V., Torbica, S., Stojanović, M., & Vojinović, I. (2023). Development and Validation of Universal 3D Blast Fragmentation Model. *Applied Sciences*, *13*(14), 8316. https://doi.org/10.3390/app13148316

20. Dyczko, A. (2023). Real-time forecasting of key coking coal quality parameters using neural networks and artificial intelligence. *Rudarsko-Geološko-Naftni Zbornik*, *38*(3), 105-117. https://doi.org/10.17794/rgn.2023.3.9

21. Dychkovskyi, R., Tabachenko, M., Zhadiaieva, K., Dyczko, A., & Cabana, E. (2021). Gas hydrates technologies in the joint concept of geoenergy usage. *E3S Web of Conferences*, (230), 01023. https://doi.org/10.1051/e3sconf/202123001023

22. Moldabayev, S.K., Sdvyzhkova, O.O., Babets, D.V., Kovrov, O.S., & Adil, T.K. (2021). Numerical simulation of the open pit stability based on probabilistic approach. *Naukovyi Visnyk Natsionalnoho Hirnychoho Universytetu*, (6), 29-34. https://doi.org/10.33271/nvngu/2021-6/029

23. Aitkazinova, S., Sdvyzhkova, O., Imansakipova, N., Babets, D., & Klymenko, D. (2022). Mathematical modeling the quarry wall stability under conditions of heavily jointed rocks. *Naukovyi Visnyk Natsionalnoho Hirnychoho Universytetu*, (6), 18-24. https://doi.org/10.33271/nvngu/2022-6/018

24. Khomenko, O., Rudakov, D., Lkhagva, T., Sala, D., Buketov, V., & Dychkovskyi, R. (2023). Managing the horizon-oriented in-situ leaching for the uranium deposits of Mongolia. *Rudarsko-Geološko-Naftni Zbornik*, *38*(5), 49-60. https://doi.org/10.17794/rgn.2023.5.5

25. Petlovanyi, M., Lozynskyi, V., Saik, P., & Sai, K. (2019). Predicting the producing well stability in the place of its curving at the underground coal seams gasification. *E3S Web of Conferences*, (123), 01019. https://doi.org/10.1051/e3sconf/201912301019

26. Smoliński, A., Malashkevych, D., Petlovanyi, M., Rysbekov, K., Lozynskyi, V., & Sai, K. (2022). Research into Impact of Leaving Waste Rocks in the Mined-Out Space on the Geomechanical State of the Rock Mass Surrounding the Longwall Face. *Energies*, *15*(24), 9522. https://doi.org/10.3390/en15249522

27. Tereshchuk, R.M., Khoziaikina, N.V., & Babets, D.V. (2018). Substantiation of rational roof-bolting parameters. *Naukovyi Visnyk Natsionalnoho Hirnychoho Universytetu*, (1), 19-26.

28. Shvabiuk, V.I. (2016). *Opir materialiv*. Kyiv, Ukraina: Znannia, 407 s.

29. Persson, P.-A., Holmberg, R., & Lee, J. (2018). *Rock Blasting and Explosives Engineering*. Boca Raton, USA: CRC Press, 560 p. https://doi.org/10.1201/9780203740514

30. Torbica, S., & Lapčević, V. (2015). Estimating extent and properties of blast-damaged zone around underground excavations. *Rem: Revista Escola de Minas*, *68*(4), 441-453. https://doi.org/10.1590/0370-44672015680062

31. Borozenets, H.M., Pavlov, V.M., Holubnychii, O.V., Babenko, Ye.M., & Koltsov, V.O. (2015). *Prykladna mekhanika i osnovy konstruiuvannia*. Kyiv, Ukraina: NAU, 356 s.