ISSN 2500-0632 (ON-LINE)
MINING SCIENCE AND TECHNOLOGY (RUSSIA)
GORNYE NAUKI I TEKHNOLOGII
2020;5(3):235-252

MISIS National University of Science and Technology

ORIGINAL PAPERS

DOI: 10.17073/2500-0632-2020-3-235-252

Ensuring Wall Stability in the Course of Blasting at Open Pits of Kyzyl Kum Region

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Abstract: Involvement of deep deposits in mining predetermined the trend of development of open pit mining towards increasing the depth of open pits. The main limitation imposed on drilling and blasting in the near-contour zone of an open pit is the need to protect the pit walls and engineering structures on the walls from seismic effects of huge blasts. As practice shows, the most effective and proven method of protecting pit walls is the use of blasting by presplitting method, creation of a shielding gap and a shielding layer of blasted rock mass, i.e. pre-splitting of the pit walls, preceding the huge blast. Therefore, the study of stress-strain state of the near-contour rock mass, determination of the parameters of blastholes for edge pre-splitting (preliminary shielding gap formation) in open pits is an urgent task. The analysis of the pit wall design and stress-strain state of rock mass at Kokpatas deposit exploited by Navoi Mining and Metallurgical Combine allowed to determine the model, as well as the method for calculating stress-strain state of the rock mass. When assessing stability of the pit walls, an approach known as the displacement method was used. Applying the boundary integral equations method allowed to develop an algorithm for calculating stresses in the rock mass for the conditions of Kokpatas deposit. A technique has been developed for experimental studies of blasting contour blasthole charges (blasting by pre-splitting method) using models, allowing to study fracturing on volumetric models and wave interaction by the method of high-speed video recording of the blasting process in transparent models, as well as to determine the parameters of stress waves during blasting in samples of real rocks. A method for formation of stable pit wall slopes, an excavator method for bench pre-splitting on ultimate envelope (contour) of a pit, and a method for initiating blasthole charges in the near-contour zone of a pit have been developed and implemented in the industry.

Keywords: blasting, pit wall stability, pit envelope, pit wall control, blasting by presplitting method, zone of residual deformation, method for calculating effective parameters, blasthole charges, explosives

For citation: Zairov Sh. Sh., Urinov Sh. R., Nomdorov R. U. Ensuring wall stability in the course of blasting at open pits of Kyzyl Kum region. *Gornye nauki i tekhnologii = Mining Science and Technology (Russia)*. 2020;5(3):235-252. (In Russ.) DOI: 10.17073/2500-0632-2020-3-235-252

Формирование устойчивости бортов при ведении взрывных работ на карьерах Кызылкумского региона

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Аннотация: Вовлечение в разработку глубокозалегающих месторождений предопределило тенденцию развития открытого способа добычи в направлении увеличения глубины карьеров. Основным ограничением, налагаемым на ведение буровзрывных работ (БВР) в приконтурной зоне карьера, является необходимость предохранения бортов карьера и инженерных сооружений на бортах от сейсмического воздействия массовых взрывов. Как показывает практика, наиболее эффективным и опробованным методом защиты бортов карьера является применение контурного взрывания, создание экранирующей щели и экранирующего слоя взорванной горной массы, т.е. проведение заоткоски бортов карьера, предшествующей массовому взрыву. Поэтому исследования напряженно-деформированного состояния пород приконтурного массива, определение параметров взрывных скважин при предварительном щелеобразовании на карьерах является актуальной задачей. Проведенный анализ конструкции бортов и напряженно-деформированного состояния пород месторождения Кокпатас Навоийского горно-металлургического комбината позволил определить модель, а также метод расчета напряженно-деформированного состояния массива горных пород. При оценке устойчивости бортов карьера использован подход, известный как метод перемещений. Использование метода граничных интегральных уравнений позволило разработать алгоритм расчета напряжений в массиве для условий месторождения Кокпатас. Разработана методика проведения экспериментальных исследований взрывов контурных скважинных зарядов на моделях, позволяющая исследовать трещинообразование на объемных моделях и волновое взаимодействие методом высокоскоростной видеорегистрации процесса взрыва в прозрачных моделях, а также определить параметры волн напряжений при взрыве в образцах реальных горных пород. Разработаны и внедрены в промышленность способ формирования устойчивых откосов бортов карьера, экскаваторный способ заоткоски уступов на предельном контуре карьера и способ инициирования скважинных зарядов взрывчатых веществ в приконтурной зоне карьера.

Ключевые слова: взрывные работы, устойчивость бортов карьера, контур карьера, заоткоска уступов карьера, контурное взрывание, зона остаточных деформаций, методика расчета эффективных параметров, скважинные заряды, взрывчатые вещества

Для цитирования: Заиров Ш. Ш., Уринов Ш. Р., Номдоров Р. У. Формирование устойчивости бортов при ведении взрывных работ на карьерах Кызылкумского региона. *Горные науки и технологии*. 2020;5(3): 235-252. DOI: 10.17073/2500-0632-2020-3-235-252

Introduction

Involvement of deep deposits in mining predetermined the trend of development of open pit mining towards increasing the depth of open pits. As known, in deep open pits, extraction of minerals is carried out in difficult mining and geological conditions, when it is necessary to reliably ensure the stability of pit walls and their components.

In the course of blasting, as a result of disturbance of the rock mass outside the pit contour, rock mass weakening occurs due to changes in fracturing, the appearance of residual deformations, and decreasing rock strength at the contacts of structural blocks. The main limitation imposed on drilling and blasting in the near-contour zone of an open pit is the need to protect the pit walls and engineering structures on the walls from seismic effects of huge blasts. As practice shows, the most effective and proven method of protecting pit walls is the use of blasting by presplitting method, creation of a shielding gap and a shielding layer of blasted rock mass, i.e. pre-splitting of the pit walls, preceding the huge blast.

To date, significant progress has been achieved in the field of application of blasting by pre-splitting method for pre-splitting benches in open pits. However, a number of key issues on predictive assessment and selection of rational method for bench pre-splitting, as well as improving the parameters of blasthole charges in the course of blasting by pre-splitting method have not been resolved. At the same time, it is necessary to continue research of the stress-strain state of the near-contour rock mass, improve blasthole parameters in the course of edge pre-splitting (preliminary shielding gap formation) in open pits, and develop a method of experimental assessing the blasting by presplitting method when shaping slopes.

Deposits of the Kyzyl Kum region are characterized by complicated structure of ore bodies, high variability of the useful components grades, steep dip angles, and uneven thickness of the ore bodies [1–3]. Such variability significantly affects the efficiency of mining, significantly complicating selection of technological parameters of drilling and blasting operations.

In the conditions of the Kyzyl Kum region, the formation of open pit walls with slope angles up to 70^{o} is possible only if their parameters are determined by structural elements only, and not by the rock mass stability. The change in the design parameters of the pit wall slope angles is carried out by doubling, tripling the bench height with slope angles of 80–90° and the width of berms of 10–15 m [4–8].

As a result of the research, a model and method for calculating the stress-strain state of rock mass for the conditions of Kokpatas deposit exploited by Navoi Mining and Metallurgical Combine have been developed.

Models and algorithms for assessing stability of open pit walls

When assessing stability of the pit walls, an approach known as the displacement method was used. The method is equivalent to minimizing the total potential energy of a pit wall, expressed through the displacement field; it led to the following sequence when performing calculations to determine the stress-strain state and assess the stability of benches using the finite element method:

– partition of the pit wall section into finite elements and assigning nodes in which displacements are determined;

– determination of the relationship between efforts and displacements in the nodes of the element;

– comparison of the system of algebraic equilibrium equations;

– solution of the system of equations;

– determination of displacements and components of the stress-strain state of the pit wall and assessment of its stability.

It is believed that the most convenient method for calculating the stress field for areas with a complicated contour is the method of integral equations, which includes solution of the system of Fredholm integral equations. The method positive features include:

– decreasing the problem dimension;

– discretizing only the *S* area boundaries, in contrast to the finite element method.

Let us consider the method of boundary integral equations and the algorithm for calculating stresses in rock mass for the conditions of Kokpatas deposit. In the method of boundary integral equations, a space (half-space) is considered, in which the contour boundary is divided discretely into a finite number of sections. Fig. 1 shows loading diagram, and Fig. 2, the design skeleton diagram presenting the determined boundary conditions and volume forces.

The load intensity at each given section is constant. The stresses within the *S* area can be represented as

$$\bar{\sigma}_{ij}(t) = \int_{ds} \bar{k}_{ij,l}(t,\tau) P_l \, dS,$$
$$(i,j \equiv x,y); \quad t \in S; \tau \in \partial S \qquad (1)$$

where $\bar{k}_{ij,l}(t,\tau)$ is the fundamental solution for the action of a concentrated force on a homogeneous and isotropic half-plane; P_l – fictitious components of surface forces along ∂S in the *l*-th direction; *t* – interior points of *S* area; τ – boundary points ∂S.

When solving the elasticity problem, the conditions at the boundary are determined as follows:

$$\int_{\partial S} \bar{k}_{ij,l}(\Omega,\tau) P_l \cdot n_j(\Omega) dS(\tau) = \sigma_i^0(\Omega); \; \Omega,\tau \in \partial S, \quad (2)$$

where n_j is the cone of the angle between the normal to the boundary section and the coordinate axes; $\sigma_i^0(\Omega)$ – specified loads on ∂S.

Stresses in the *S* area, caused by the action of forces concentrated on the contour ∂S in the infinite half-plane, can be represented in the form

$$k_{xx,l} \cdot P_l^S = -\frac{p_x^S r_x\left(b_1 r_x^2 + b_2 r_y^2\right) + p_y^S r_y\left(b_3 r_x^2 + b_2 r_y^2\right)}{4\pi r^4};$$
$$k_{yy,l} \cdot P_l^S = -\frac{p_x^S r_x\left(b_3 r_y^2 + b_2 r_x^2\right) + p_y^S r_y\left(b_1 r_y^2 + b_2 r_x^2\right)}{4\pi r^4}; \quad (3)$$
$$k_{xy,l} \cdot P_l^S = -\frac{p_x^S r_y\left(b_1 r_x^2 + b_2 r_y^2\right) + p_y^S r_x\left(b_2 r_x^2 + b_1 r_y^2\right)}{4\pi r^4},$$

where p_x^S, p_y^S – are the components of fictitious loads within the area *S*; r_x, r_y, are the *x*-, *y*-components of the radius of vector drawn from the point in ∂S to the point in *S*:

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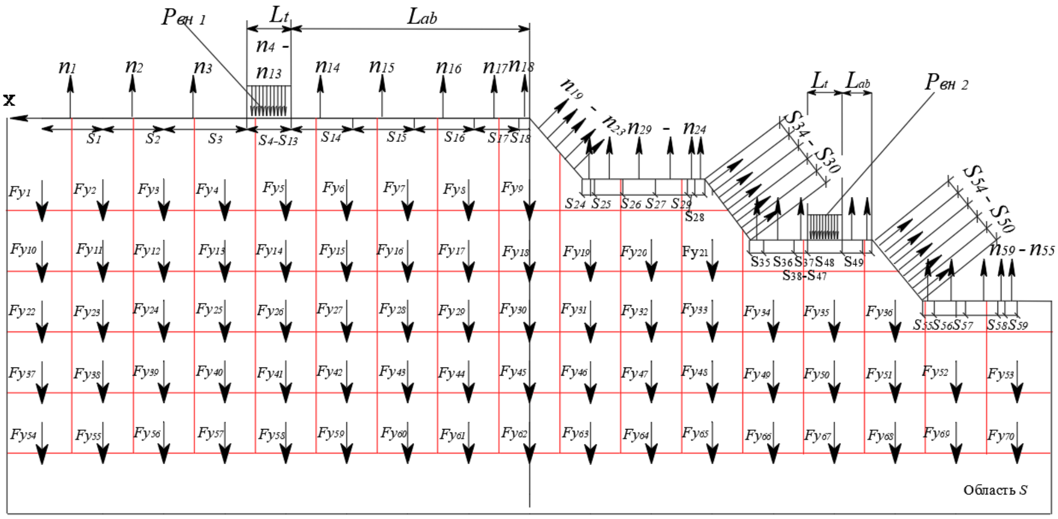

Fig. 1. Loading diagram for the wall design and calculation of stresses in the rock mass of Kokpatas open pit: n_1, n_2, …, n_j are the normals to the boundary of area ∂S; S_1, S_2, …, S_i is the length of the boundary section ∂S; $P_{вн1}$, $P_{вн2}$, …, $P_{внi}$ are external loads from dumps and equipment applied along the length L_t at a distance L_{ab} from the bench edge; F_{y1}, F_{y2}, …, F_{yi} are components of the gravity force in the i-th point of the S area.

$$b_1 = 3 + \frac{\nu}{1-\nu};\ b_2 = 2 - \frac{1}{1-\nu};\ b_3 = 1 + \frac{3\nu}{1-\nu},$$

where ν – is Poisson's ratio.

At $\tau \to \Omega$ expression (2) has a feature, which can be distinguished using the approach consisting in considering a new contour bypassing the boundary point, where $\tau = \Omega$. In this case

$$\int_{\partial S} k_{ij,l}(\Omega,\tau)P_l \cdot n_j(\Omega)dS(\tau) =$$

$$= \lim_{\substack{\Delta\partial S \to 0 \\ a \to 0}} \left[\left(\int_{\partial S - \Delta\partial S} k_{ij,l}(\Omega,\tau)P_l \cdot n_j(\Omega)dS(\tau) \right) + \right.$$

$$\left. + \left(\int_{\partial S} k_{ij,l}(\Omega,\tau)P_l \cdot n_j(\Omega)dS(\tau) \right) \right]. \quad (4)$$

It was shown in [9] that

$$\lim_{a \to 0} \int_{\partial S} k_{ij,l}(\Omega,\tau)P_l \cdot n_j(\Omega)dS(\tau) \to P_x^S/2, P_y^S/2. \quad (5)$$

Then, based on formula (5), equation (2) assumes the final form of:

$$P_i^S/2 + \int_{\partial S} k_{ij,l}(\Omega,\tau)P_l \cdot n_j(\Omega)dS(\tau) = \sigma_i^0(\Omega),$$

$$i = x,y. \quad (6)$$

For numerical implementation of the method, the boundary of the area ∂S is divided into N segments of arbitrary length ∂S_i. Then, at the midpoints of each segment ∂S_i we determine the resulting boundary values

$$P_{xi}^S = \int_{\Delta S_i} p_x^S dS;\ P_{yi}^S = \int_{\Delta S_i} p_y^S dS;\ P_{xi}^{\partial S} = \int_{\Delta S_i} p_x^{\partial S} dS;$$

$$P_{yi}^{\partial S} = \int_{\Delta S_i} p_y^{\partial S} dS, \quad (7)$$

where ΔS_i – are the components of fictitious loads on the boundary of the area ∂S.

Using the numerical trapezoidal method for evaluation of integral and formula (7), we can numerically approximate expression (6):

$$P_{xi}^S/2 - 1/(4\pi)\sum_{\substack{j=1 \\ j\neq i}}^{N} \left[\begin{matrix} \left(b_1 r_{xij}^3 n_{xi} + b_2 r_{xij} r_{yij}^2 n_{xi} + b_1 r_{xij}^2 r_{yij} n_{yi} + b_2 r_{yij}^3 n_{yi}\right) P_{xj}^S + \\ + \left(b_3 r_{xij}^2 r_{yij} n_{xi} - b_2 r_{yij}^3 n_{xi} + b_2 r_{xij}^3 n_{yi} + b_1 r_{xij} r_{yij}^2 n_{yi}\right) \cdot P_{yj}^S \end{matrix} \right] \Delta S_i \left(r_{xij}^2 + r_{yij}^2\right)^{-2} = P_{xi}^{\partial S}; \quad (8)$$

$$P_{yi}^S/2 - 1/(4\pi)\sum_{\substack{j=1 \\ j\neq i}}^{N} \left[\begin{matrix} \left(b_1 r_{xij}^2 r_{yij} n_{xi} + b_2 r_{yij}^3 n_{xi} + b_3 r_{xij} r_{yij}^2 n_{yi} - b_2 r_{xij}^3 n_{yi}\right) P_{xj}^S + \\ + \left(b_2 r_{xij} n_{xi} + b_1 r_{xij} r_{yij}^2 n_{xi} + b_1 r_{yij}^3 n_{yi} + b_2 r_{xij}^2 r_{yij} n_{yi}\right) \cdot P_{yj}^S \end{matrix} \right] \Delta S_i \left(r_{xij}^2 + r_{yij}^2\right)^{-2} = P_{yi}^{\partial S}. \quad (9)$$

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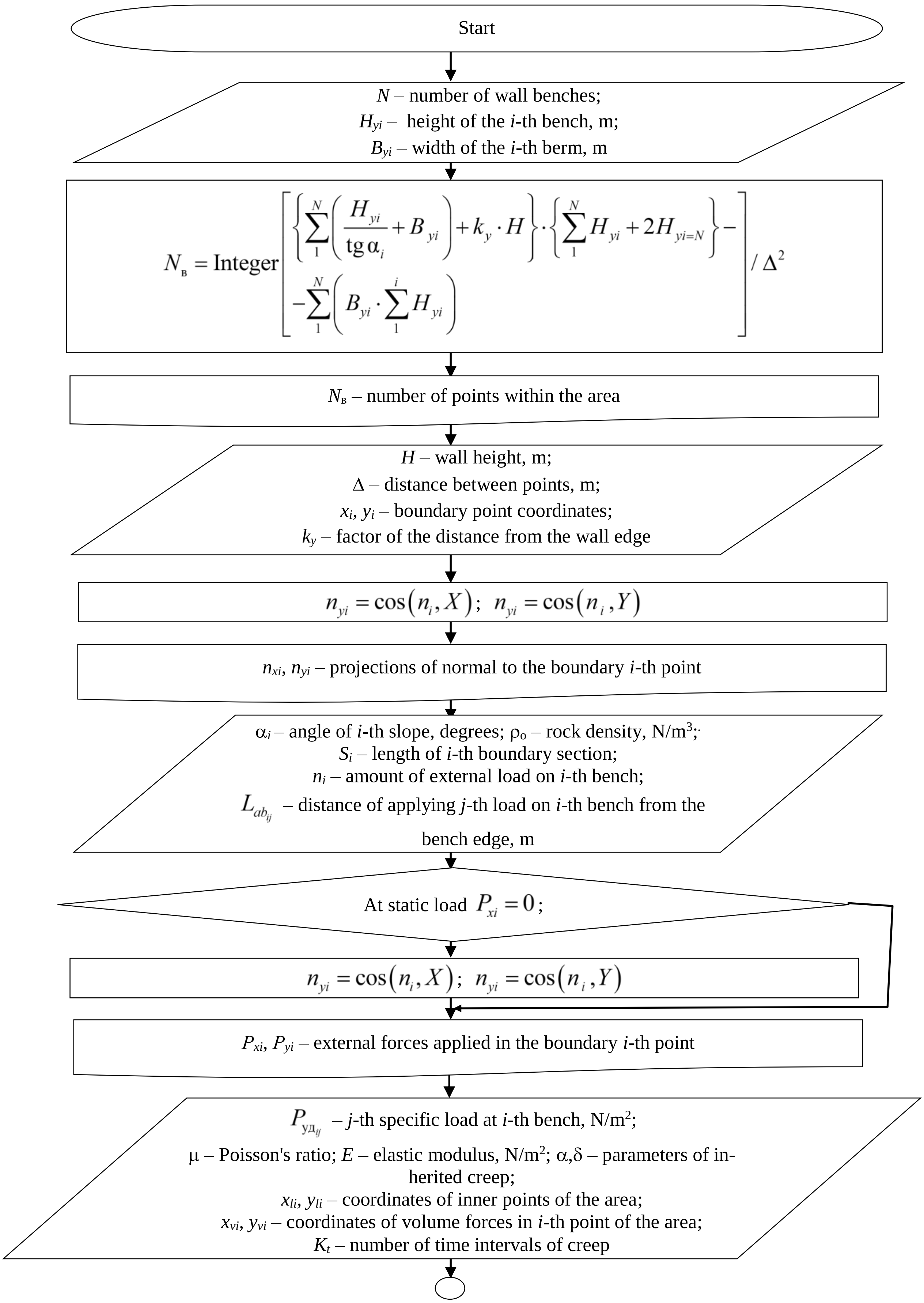
Initial data for the calculation and the formed boundary conditions



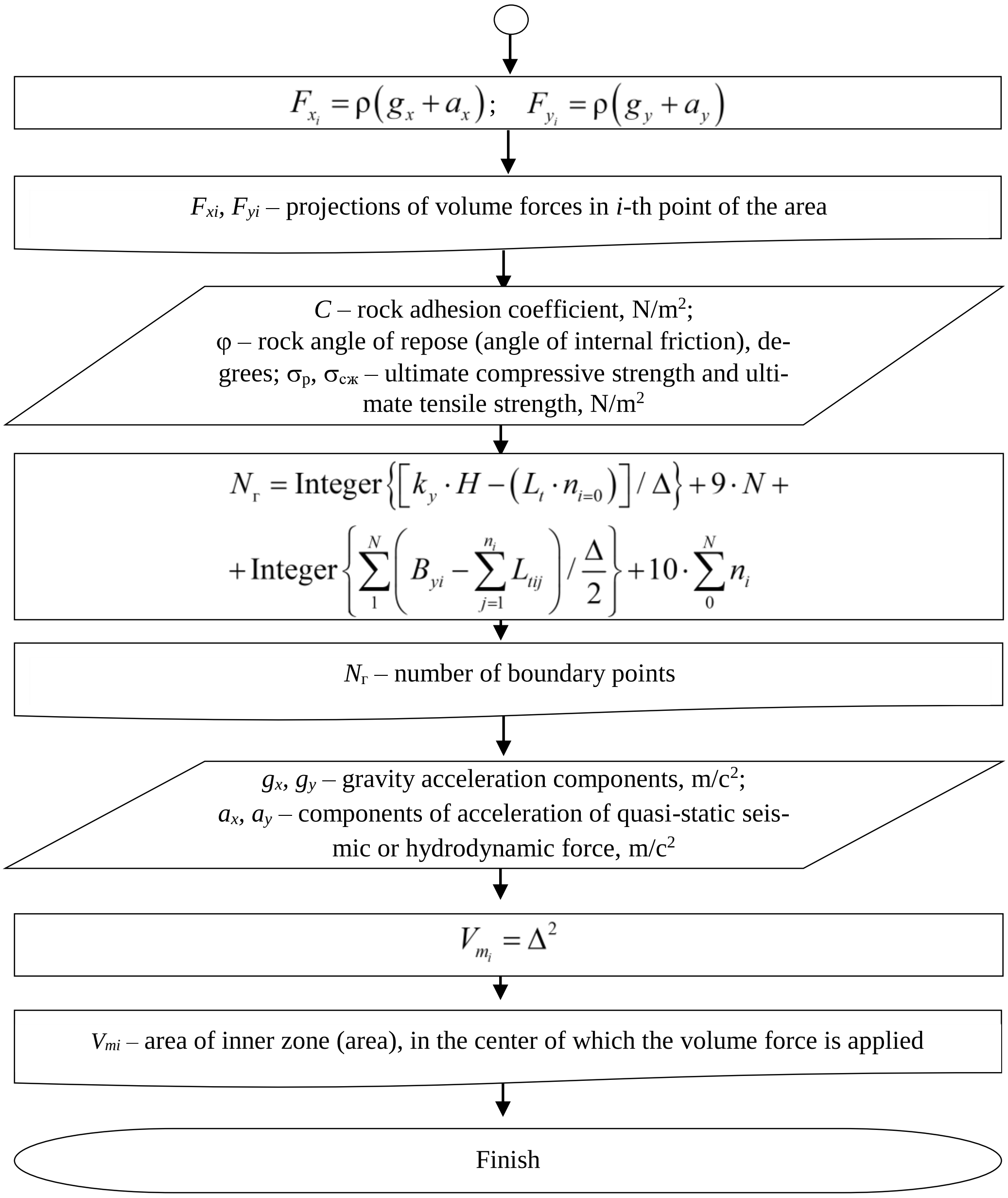


Fig. 2. Block diagram for calculating the pit wall design parameters and stresses in rock mass at Kokpatas open pit

After calculating the fictitious loads P_{xj}^S и $P_{yj}^S\,(j = \overline{1, N})$ the stress components in the area S are found from the following formulas:

$$\sigma_{xi} = -1/(4\pi)\sum_{j=1}^{N}\left[P_{xj}^S r_{xij}\left(b_1 r_{xij}^2 + b_2 r_{yij}^2\right) + P_{yj}^S r_{yij}\left(b_3 r_{xij}^2 - b_2 r_{yij}^2\right)\right]\cdot\left(r_{xij}^2 + r_{yij}^2\right)^{-2}; \tag{10}$$

$$\tau_{xi} = -1/(4\pi)\sum_{j=1}^{N}\left[P_{xj}^S r_{yij}\left(b_1 r_{xij}^2 + b_2 r_{yij}^2\right) + P_{yj}^S r_{xij}\left(b_2 r_{xij}^2 + b_1 r_{yij}^2\right)\right]\cdot\left(r_{xij}^2 + r_{yij}^2\right)^{-2}. \tag{11}$$

Using expressions (10) and (11), it is possible to determine the stresses at all points of the rock mass, which is especially important when studying the areas of its weakening.

In the study of the stress state of the near-wall rock mass and its deformation, this area is modeled by a half-plane, along the boundary of which loads act, and inside the half-plane has block structure.

At the first stage, the area boundary is divided into sections, the number of which depends on the boundary configuration (completeness of information content) and the number of points where the external load is applied to the contour. The sections into which the boundary is divided can be condensed and have an arbitrary length.

To construct isolines of stress components, the internal area is divided by a coordinate grid related to the coordinate system in which the area boundary is described.

After formalization of the area boundary and its internal part in the presence of external forces at the area boundaries, they are also described in the form of distributed or point load in an analytical form.

Then, based on expressions (8) and (9), the boundary integral equations are formed in the form of matrices

$$a_{ij} = \begin{pmatrix} a_{11}...a_{1n}b_{11}...b_{1n} \\ a_{n1}...a_{nn}b_{1n}...b_{nn} \\ c_{11}...c_{1n}d_{11}...d_{1n} \\ c_{n1}...c_{nn}d_{n1}...d_{nn} \end{pmatrix}. \quad (12)$$

The right-hand sides of equations (8) and (9) represent the boundary conditions on the segments ∂S_i. In accordance with [10], to take into account the gravitational field and the loads arising inside the rock mass (for example, blasting or seismic action), the right-hand side of the equations is supplemented with integrals describing these actions. Then expression (6) will assume the form of:

$$P_i^S/2 + \int_{\partial S} k_{ij,l}(\Omega,\tau)P_l \cdot n_j(\Omega)dS(\tau) =$$

$$= \sigma_i^0(\Omega) + \int_v K_{ij}(t) \cdot P_i(t)dv,\ i = x,y, \quad (13)$$

where $K_{ij}(\tau,t)$ is the Green's function; $P_i(t)$ – are efforts arising at the *i*-th point of the area *S*.

To solve the resulting system of linear equations, iteration according to the method of Gauss or Seidel [11] is used.

After calculating, using expressions (10)–(12) the components of the stress tensor σ_x, σ_y and τ_{xy} the principal stresses are determined using the known expressions:

$$\sigma_1 = (\sigma_x + \sigma_y)/2 + \sqrt{(\sigma_x - \sigma_y)^2/4 + \tau_{xy}};$$

$$\sigma_2 = (\sigma_x + \sigma_y)/2 - \sqrt{(\sigma_x - \sigma_y)^2/4 + \tau_{xy}};$$

$$\tau_{\max} = (\sigma_1 - \sigma_2)/2. \quad (14)$$

Justification of process operational diagrams and parameters of blasting operations in an open pit

Analysis of the data on deformation of rocks outside the rock mass contour in the course of applying huge (single) blasts allowed concluding that, when approaching the ultimate pit envelope (contour), it is necessary to change the drilling-and-blasting process [12–15]. One of the criteria for determining the amount of simultaneously blasted explosives was the value of the seismic hazard measure, at which residual deformations of the rocks composing the pit benches and walls were practically excluded.

It is recommended to calculate the amount of simultaneously blasted explosives taking into account K_c, factor, numerical values of which depend on the criterion involving type of displacement, location of the protected object, type of rocks, and type of fracturing (Table 1).

The amount of simultaneously blasted explosives during the approach of blasting to the ultimate envelope (contour) of the open pit is recommended to be determined by the following formula:

$$Q = (r_б/K_c)^3,\ \text{кг}, \quad (15)$$

where $r_б$ is safe distance for the contour benches from the blasting location to the protected object, m; K_c is the averaged value of the factor.

Table 1

K_c factor values for different conditions

Location of the protected object	Average edge size of an elementary block, m	K_c
At the level of blasted bench	Up to 0.1	8.7
	0.1–0.3	6.2
	0.3–0.6	3.76
	0.6–2.0	3.02
	2.0	2.8
At the level above the blasted bench	Up to 0.1	8.22
	0.1–0.3	5.87
	0.3–0.6	3.56
	0.6–2.0	2.85
	2.0	2.65
Two levels above the blasted bench	Up to 0.1	7.89
	0.1–0.3	5.61
	0.3–0.6	3.42
	0.6–2.0	2.74
	2.0	2.54

The regularities of changing the residual deformation zone depending on the amount of simultaneously blasted explosives are shown in Fig. 3.

Thus, the data from Table 1 allow to determine the acceptable amount of simultaneously blasted explosives depending on the rock mass structure and the schematic of non-working benches arrangement in the ultimate contour of the pit wall.

Based on the need to ensure minimizing the zone of severe deformation, the optimal specific consumption and the amount of explosives per 1 m of the work front were determined. Based on the established optimal explosive consumption, allowing preventing peripheral (outside the pit contour) rock mass deformation, the width of the near-contour zone $R_{п.з}$ was determined, which is the distance from the upper edge of the working bench to the points towards the stationary wall:

$$R = A \cdot (w + (n - 1)b)^{1/3}, \text{ м}, \qquad (16)$$

where A is an empirical coefficient (A = 11.5–18.0); w is the width of the mined belt depending on the line of resistance along the bottom, m; n is the number of rows of blastholes, pcs; b is the distance between the blasthole rows, m.

Expression (16) allows, at a specified distance from the blast location to the ultimate pit wall contour, to determine the dimensions of the blasted block along the work face (Fig. 4).

Thus, solutions have been recommended to reduce the width of the residual deformation zone, the parameters of the contouring charges were established to create the shielding gap with an increased protective ability, and the parameters of blasting in the near-contour zone were selected to ensure the creation of the shielding gap with increased protective capacity and corresponding limitation of stresses in the incident compression wave.

A technique has been developed for calculating the effective parameters of drilling-and-blasting in the course of blasting by presplitting method taking into account the physical-mechanical and mining-technological properties of the rock mass [16–18].

It is recommended to determine the presplitting blasthole charge diameter using the formula:

$$\boldsymbol{d_з = 0,55\, \frac{(\rho_0 c^2)^{7/12}}{(\rho_{ВВ} D^2)^{1/3} {\sigma_{сж}}^{1/4}}\, d_c}, \text{ мм}, \qquad (17)$$

where ρ_0 is the rock density, kg/m^3; c is velocity of longitudinal (P) wave in the rock, m/s; $\rho_{ВВ}$ – is density of the explosive, kg/m^3; D is the explosive detonation velocity, m/s; $\sigma_{сж}$ – is ultimate compressive strength, Pa; d_c is the blasthole diameter, mm.

The change in the diameter of the presplitting blasthole depending on the explosive charge density, the rock density, the rock ultimate compressive strength, the longitudinal wave velocity in the blasted rock, and the commercial explosive detonation velocity was established (Fig. 5).

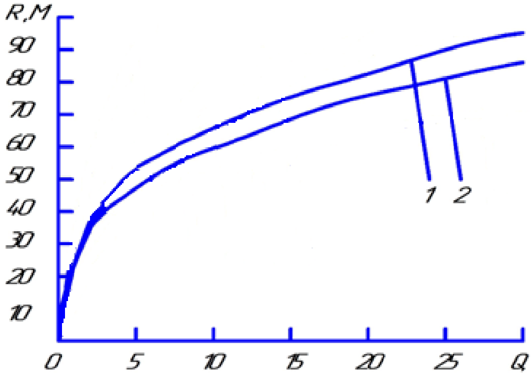


Fig.3. The size of residual deformation zone *R* (m) as a function of the amount of simultaneously blasted explosives *Q (t)*:
1 – vertical displacements; 2 – horizontal displacements

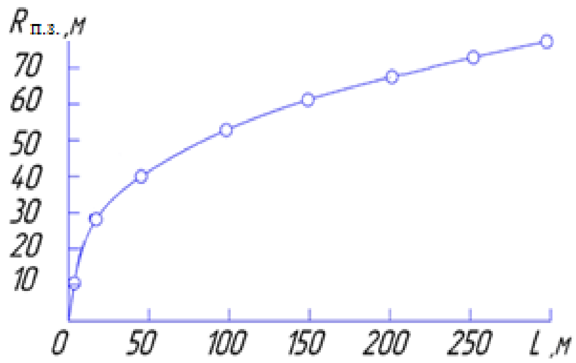


Fig. 4. The size of the near-contour zone width R_{ncz} (m) as a function of the blasted bench length *L* along the work face (m)

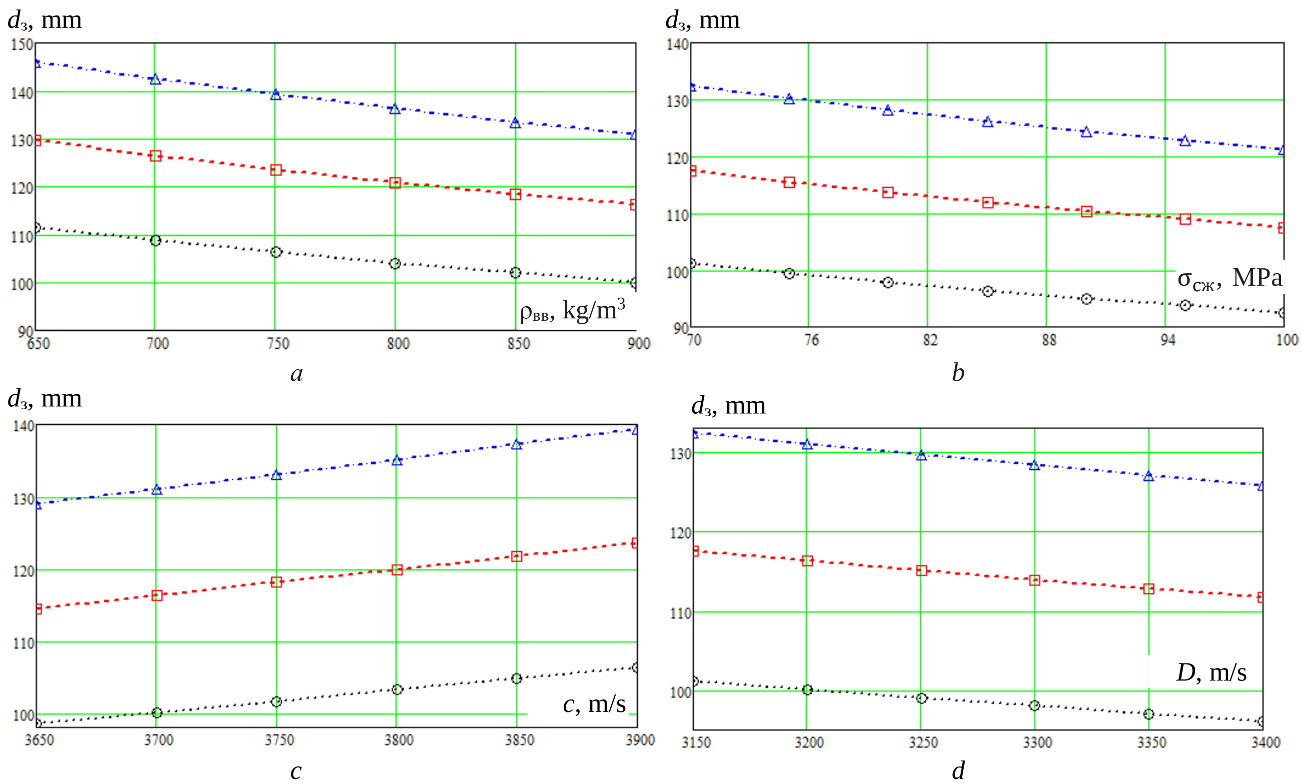


Fig. 5. Dependences of the presplitting blasthole charge diameter$d_з$ on the explosive density ρ_{BB} (*a*), ultimate compressive strength of rocks $\sigma_{сж}$ (*b*), longitudinal wave velocity *c* (*c*), and detonation velocity of explosive *D* (*d*) in different rocks:
o – soft rocks; □ – medium hardness rocks; Δ – hard rocks

Linear mass of the presplitting blasthole charge is recommended to determine by the following formula:

$$\rho = 3{,}8 \cdot 10^{-5} \frac{(\rho_0 c^2)^{7/6}}{(\rho_{BB})^{1/3} D^{4/3} (\sigma_{сж})^{1/2}} r_c^{\,2}, \quad (18)$$

where r_c is the blasthole radius, mm.

The change in the presplitting blasthole charge linear mass depending on the blasted rock density, the longitudinal wave velocity in the blasted rock, the explosive density and detonation velocity, the rock ultimate compressive strength, and the presplitting blasthole radius in different rocks was established (Fig. 6).

It is recommended to determine the distance between the presplitting blastholes for edge presplitting by the following formula:

$$a = 0{,}064\, d_{\text{c}} \left(\frac{\rho_0 c^2 \sigma_{\text{сж}}^3}{5\sigma_{\text{p}}^3} \right)^{1/8}, \text{ м}, \qquad (5)$$

where σ_{p} is the rock ultimate tensile strength, Pa.

The presplitting blasthole spacing was determined depending on longitudinal wave velocity in the blasted rock, the rock ultimate compressive strength and tensile strenght, and the presplitting blasthole radius in different rocks (Fig. 7).

Thus, the action of blasting the presplitting blasthole charges in the near-contour zone of open pits was established by determining the drilling-and-blasting effective parameters taking into account physical-mechanical and mining-technological properties of the rock mass.

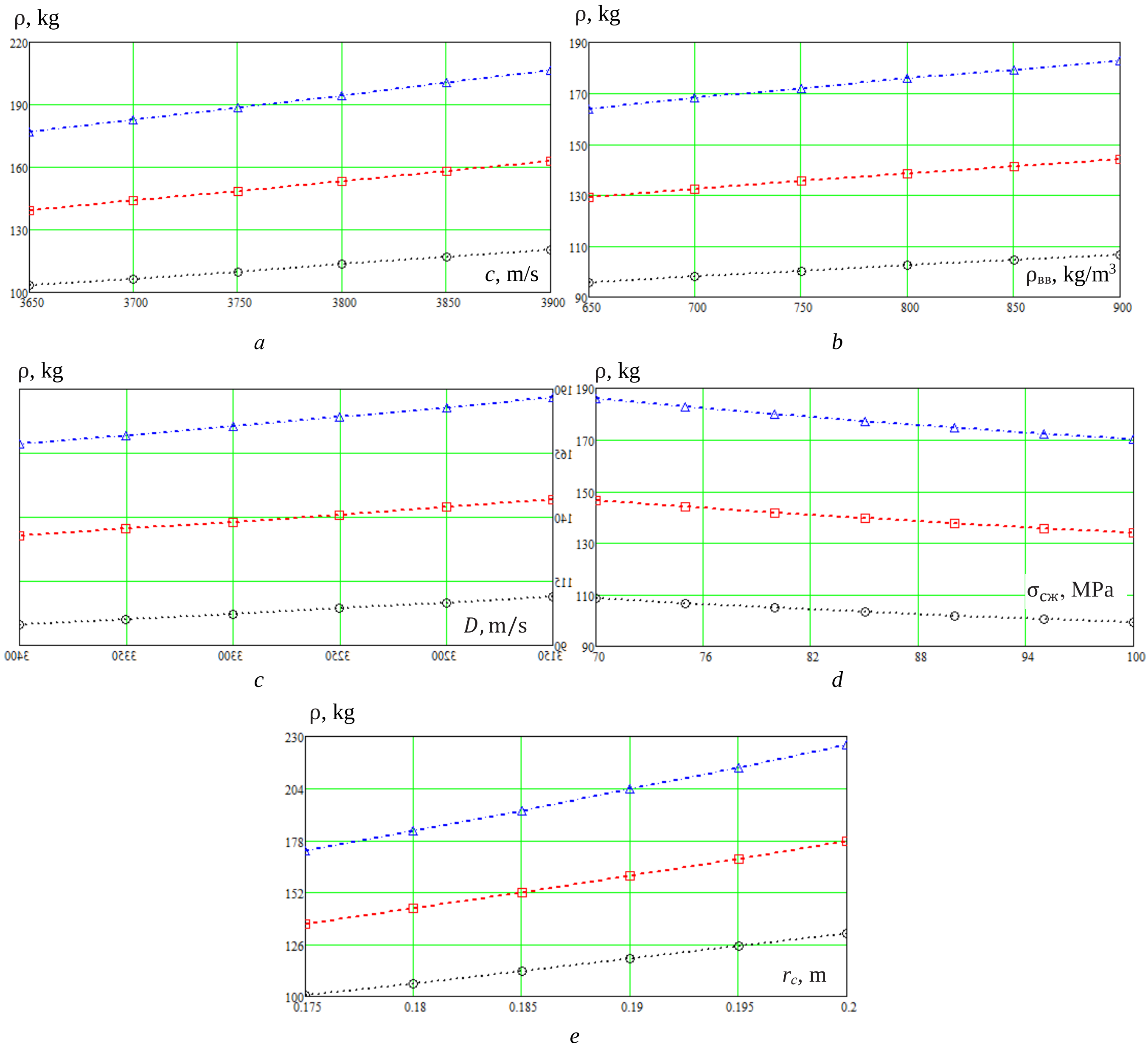


Fig. 6. Dependences of the presplitting blasthole charge linear mass ρ on the longitudinal wave velocity in the blasted rock c (*a*), , the explosive density $\rho_{\text{вв}}$ (*b*), detonation velocity of commercial explosives *D* (*c*), , the rock ultimate compressive strength $\sigma_{\text{сж}}$ (*d*), and the presplitting blasthole radius r_c (*e*) in different rocks:
o – soft rocks; □ – medium hardness rocks; Δ – hard rocks

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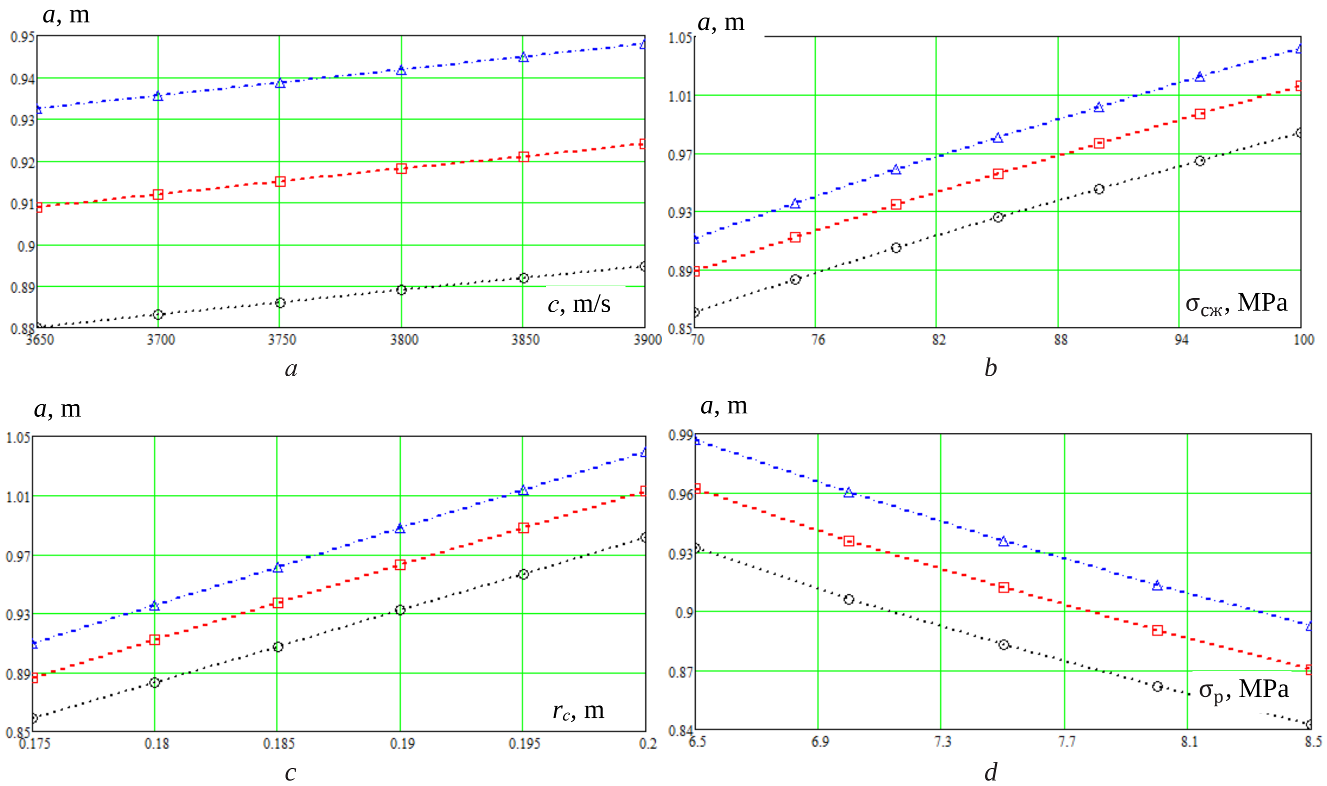

Fig. 7. The presplitting blasthole spacing *a* was determined depending on longitudinal wave velocity in the blasted rock *c* (*a*), the rock ultimate compressive strength $\sigma_{сж}$(*b*), the blasthole radius r_c (c), and the rock ultimate tensile strength σ_t (*d*) in different rocks:
o – soft rocks; □ – medium hardness rocks; Δ – hard rocks

Based on the result of studying various techniques for forming slopes in the ultimate pit walls, it has been established that the best results are achieved when applying edge pre-splitting.

Studies of the mechanism of the peripheral (beyond pit envelope) rock mass when blasting the charge, a part of which is filled with inert material, demonstrated that the blast causes asymmetric fragmentation of the rock mass that decreased the blast impact towards the protected rock mass due to energy absorption when using the inert material.

Experimental research

Based on the result of the theoretical studies, a method for experimental studies of presplitting blasthole charge blasting was developed. Laboratory studies were carried out in the scientific laboratory of the Navoi State Mining Institute.

The experimental studies of the stress wave action in transparent bodies were carried out using the Olympus i-SPEED 2 high-speed video camera and further in rocks by oscillography method using the Rohde & Schwarz RTO1004 digital oscilloscope.

The research also used ZETLAB ZET 048-C seismic station. High-speed video recording allowed to simultaneously record the propagation of waves and fractures in the zone of plastic and elastic deformations without limiting the pressure amplitude in the wave. The wave propagation speed and pulse duration were also recorded.

Instrumental measurements using SV-10Ts sensors and the oscilloscope allowed determining the proportion of energy that is spent for rock rupture. The nature of fracturing, i.e. the presence of cutter breaks (wedges) going into the rock mass or towards the free surface was determined by linear measurements.

The methodology provided for three lines of conducting experiments on models:

– research of fracturing on volumetric models;

– study of wave interaction by the method of high-speed video recording of the blast process in transparent models;

– determination of the parameters of stress waves during the blast in samples of real rocks.

The study of fracturing was carried out on volumetric models made of marble and sandstone. The charge was placed into holes drilled in the rock. The distance between the charges was modeled taking into account the geometric similarity.

The distance between the charges was changed until the optimum one for a given diameter of charges and a given rock was determined. As the criteria for assessing the optimal distance, the quality of the gap formed, the degree of fragmentation of the tested samples, and the presence of cutter breaks (flaws) were taken.

The wave interaction was studied using the data of video filming with the Olympus i-SPEED 2 high-speed camera, which allowed synchronizing the studied process beginning with the record start.

As a first approximation, it was assumed that the model and the rock mass behave like elastic bodies until the moment of rupture/fragmentation.

The filming was carried out at frequency of 2000 frames per second. The process of rock fragmentation, depending on the medium acoustic stiffness, was largely determined by the parameters of the incident and reflected stress waves. To measure the parameters of the interacting charges stress waves in the models, SV-10Ts sensors with recording in Rohde & Schwarz RTO1004 digital storage oscillograph were used.

In the course of interpretation of the oscillograms, nameplate data of the sensors were used.

When modeling, it was required to determine the optimal distances between the charges, allowing obtaining high-quality gap with minimum fragmentation of the tested samples. The smallest possible charge diameter in the models was 2.0–2.5 mm.

To reduce the degree of the sample disintegration, decked charges were modeled. With the use of glass tubes, the charge was distributed along the entire depth of the hole into four parts with three air gaps. The distance between the blastholes varied from 6.5 to 35 diameters of the charge.

Thus, the technique has been developed for experimental studies of blasting contour blasthole charges (blasting by pre-splitting method) using models, allowing to study fracturing on volumetric models and wave interaction by the method of high-speed video recording of the blasting process in transparent models, as well as to determine the parameters of stress waves during blasting in samples of real rocks.

Based on the result of the conducted research, the method of bench pre-splitting in near-contour zone of an open pit has been developed, which provides decreasing the rock mass disturbances (fracturing and fragmentation), as well as minimizes formation of talus and sliding (Fig. 8). According to this method, when mining approaches the ultimate pit contour (envelope) *1*, the 10-m high benches 2 shall be doubled. On the upper bench, at a distance of 1 m from the design contour of the open pit, a number of inclined blastholes *3* of 190 mm in diameter shall be drilled by Driltex-D25KS or URB-2A-2B drilling rig with sub-drilling of 2 m. The pre-splitting inclined blasthole spacing in a row is 2 m.

In the lower bench, three rows of vertical blastholes 250 mm in diameter shall be drilled by 4 drilling rigs SBSh-250MN at 5×5 m grid with sub-drilling of 1 m. At the distance of 3 m from the third row of vertical blastholes, one more row of additional barrier presplitting blastholes *5* with diameter of 190 mm shall be drilled up to the pit design contour using Driltex-D25KS or URB-2A-2B drilling rigs. The vertical barrier presplitting blasthole spacing in the row is 2 m.

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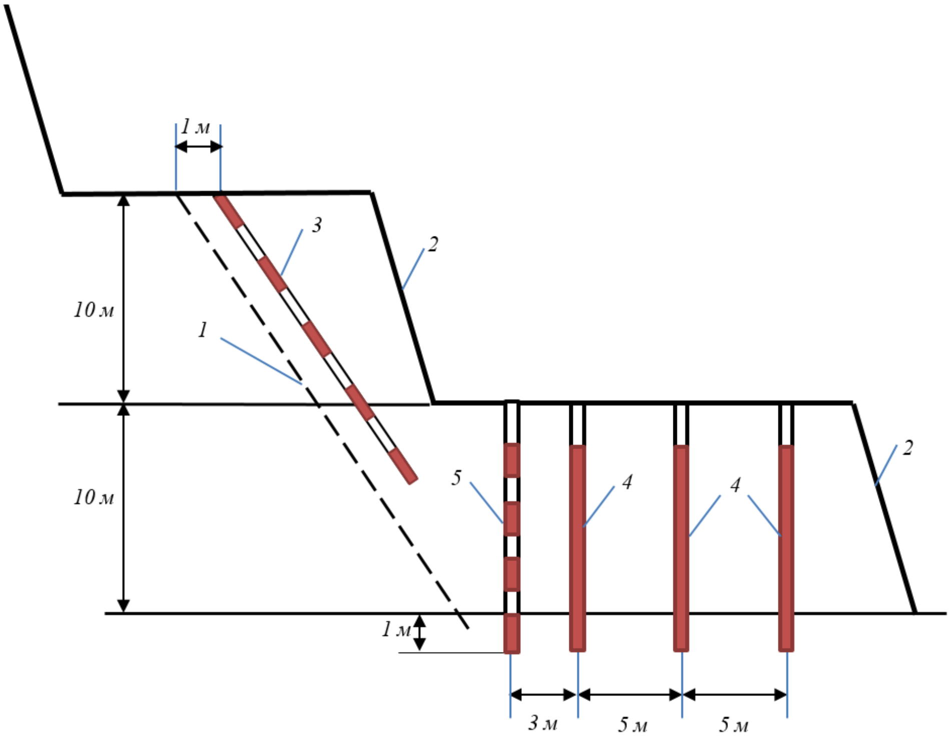

Fig. 8. Schematic of bench pre-splitting in pit near-contour zone:
1 – the pit ultimate contour; *2* – pit working bench; *3* - presplitting inclined blasthole;
4 – vertical break hole; *5* – vertical barrier presplitting blasthole

Charges in the presplitting blastholes at the upper bench and in the vertical barrier presplitting blastholes at the lower bench are formed in the form of garlands of intermediate detonators of Nobelite-216Z brand with diameter of 70 mm, weighing 2 kg, and detonating cord of DSHE-12 brand at specific consumption of 2 kg/r.m.

The vertical barrier presplitting blastholes at the lower bench are filled with continuous charges of commercial explosives with specific consumption of 0.4–0.6 kg/m^3.

The charges in the presplitting blastholes at the upper bench are detonated first, and then the charges at the lower bench are detonated using short-delay sequential blasting after 35 ms interval from the exposed bench surface to the design contour.

In any mining conditions, regardless of the rock mass strength in the near-contour zone, the initial angle of bench slope and its height, a bench slope formed at any angle exceeding the angle of repose will eventually collapse tending to take the angle of repose. The use of natural conditions for natural presplitting of bench slopes was the basis for the development of the method for presplitting of pit wall slopes using mechanical rupture of the rock mass (Fig. 9), which allows ensuring the rock mass stability in the pit design contour and reliable controlling the presplitting parameters for the life of mine, decreasing overburden volume, preserving strength of the peripheral (beyond-contour) rock mass and ensuring mining safety.

According to this method, when setting the upper bench of 15 m high with slope angle of 60^o in the ultimate pit contour, the last row of vertical blastholes is drilled at a distance of 1.5–2.5 m from the design position of the lower edge of the bench. The berm width is 17 m. When blasting two rows of the vertical blasthole charges in the near-contour zone, rock mass ***1*** is formed. The blasted rock mass handling and the slope formation is performed by two excavators in the following order: the lower excavator handles the first cut *1*, the upper excavator handles the upper part of the bench 5 m high, and the rock is handled to

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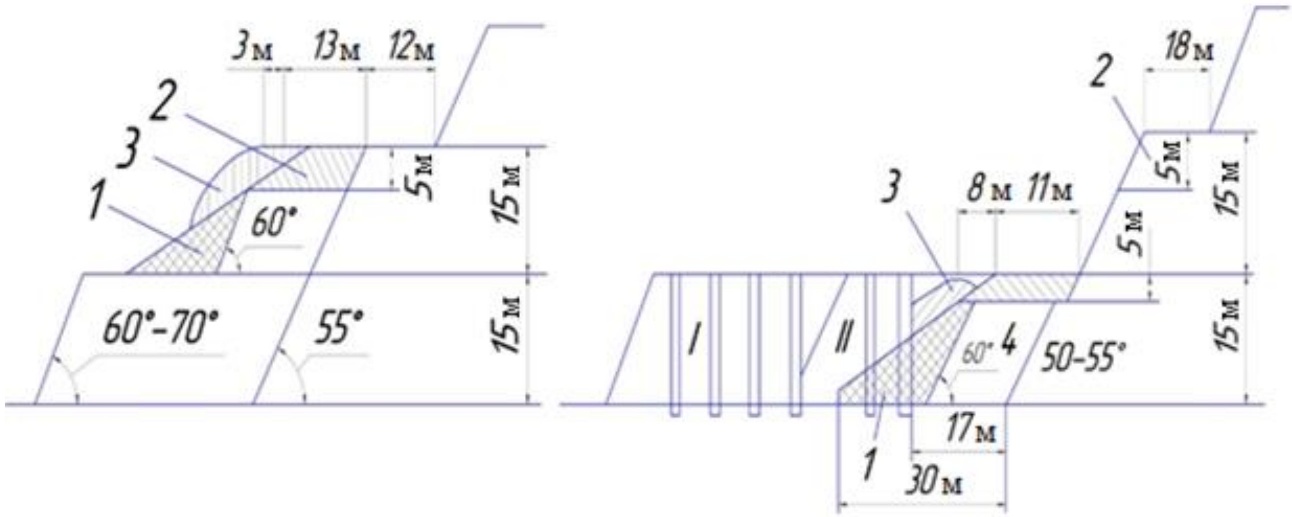
the bench bottom ***2***; the lower excavator handles the rock handled from the upper sub-bench *3*, and then handles the third cur, while forming the lower part of the upper sub-bench *4*.

A method of initiation of blasthole charges in the near-contour zone of an open pit is recommended (Fig. 10), which enables reducing the level of seismic vibrations and increasing integrity (stability) of near-wall rock masses and engineering structures in open pits – protecting them from the blasting seismic actions [19–21].



a *b*

Fig. 9. Schematic of bench formation by excavators in the ultimate pit envelope (contour):
a – intermediate excavator presplitting; *b* – design contour presplitting; *1* – rock mass at the first excavator cut; *2* – rock mass handled to the bench bottom; *3* – rock transshipped from the upper sub-bench; *4* – the lower part of the upper sub-bench; I – rows of vertical blasthole charges for fragmentation; II – rows of vertical blasthole charges in the near-contour zone

According to this method, in the block, where rocks should be fragmented, 10 rows of blastholes of 252 mm in diameter shall be drilled at 5×5 m grid using SBSh-250MN drilling rig. At the bench height of 15 m, the blasthole length is 17 m, the cut length is taken as 5 m, the charge length is 12 m, the lower half of the blasthole is filled with Nobelan 2080 commercial explosive with density of charge of 1.25 g/cm^3, and the upper half with Igdanite commercial explosives with density of charge of 0.85 g/cm^3. Each blasthole charge weighs 618 kg. Downhole blasting caps are installed at the bottom of the blastholes (one blasthole - one blasting cap). Delay intervals between the blasthole rows are taken as 67 ms, and those between blastholes in a row, 42 ms.Sequence of blasting is from the exposed bench surface towards the design contour. The initiation of charges in the SINV system is carried out by ED-8Zh electrical blasting caps and the main wire of DSHE-12 detonating cord. The source of the explosive pulse for the non-electrical SINV initiation system is the SINV-START.

The developed methods for the formation of stable slopes of the pit walls have been implemented at the Kokpatas deposit. Applying the methods, the recommended sequence and parameters of the bench setting ensured the quality of bench presplitting, complete integrity of the peripheral (beyond-contour) rock mass and mining safety.

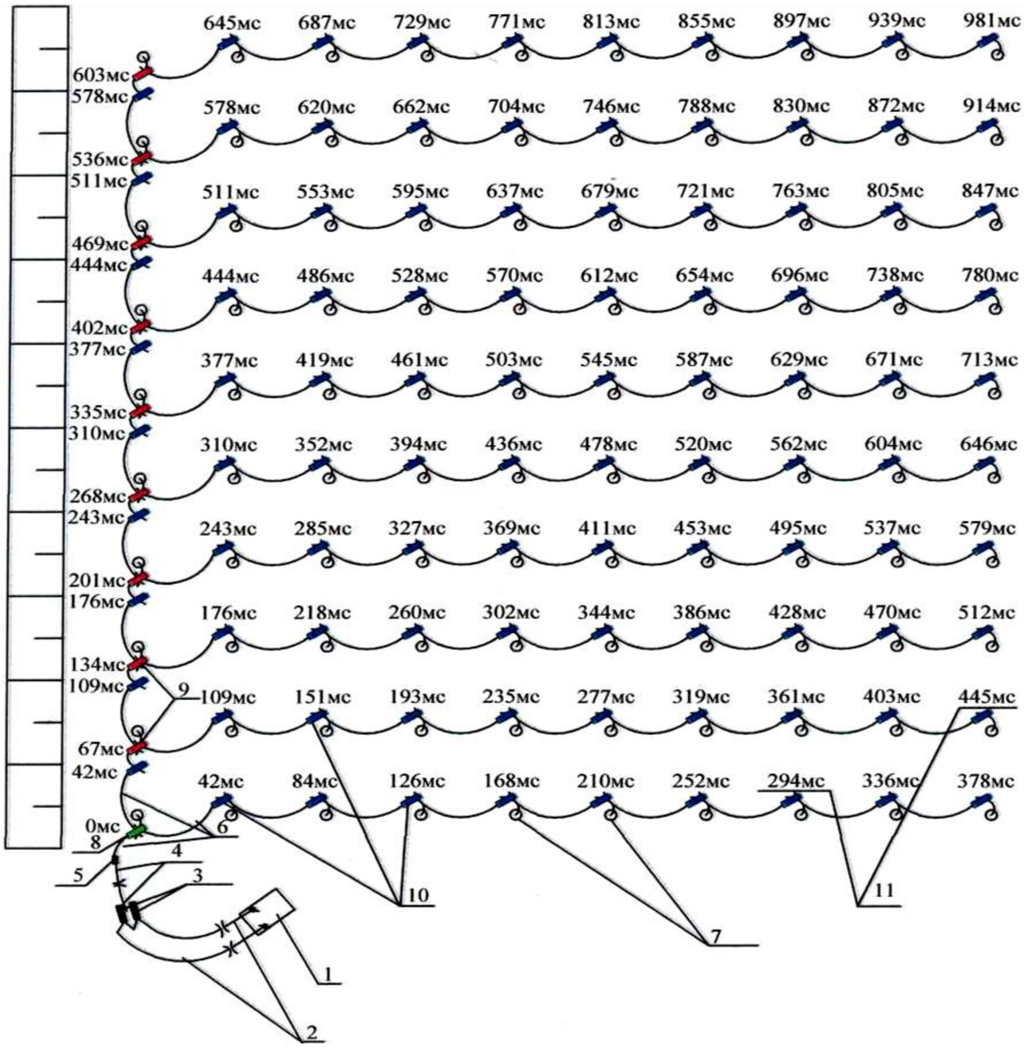


Fig. 10. Blasting schematic in the method of initiation of blasthole charges in the pit near-contour zone:
1 – blasting machine; *2* – electric wires; *3* – instantaneous electric blasting caps; *4* – main wire of detonating cord; *5* – connection of detonating cord with waveguide; *6* – waveguide; *7* – vertical blastholes in plan view; *8* – surface connecting unit, inside which blasting cap with delay interval of 0 ms is positioned; *9* – the same, with delay interval of 25 ms; *10* – the same, with delay interval of 42 ms; *11* – actuation time of the surface connecting units not taking into account wave passing hrough the waveguides, ms

Conclusions.

1. In various geological, mining, and climatic conditions, it is possible to form a pit wall with slope angle of up to 70°. Stability of benches in hard rocks is determined by the rock mechanical-and-physical properties, the length and orientation of fractures relative to the slope, as well as adhesion, the angle of internal friction at the contact, the fracture surface roughness and the filler properties.

2. The pit wall designs and stress-strain state of rocks at Kokpatas deposit have been investigated. The research findings enabled developing a model and method for calculating the stress-strain state of the rock mass.

3. The change in the presplitting blasthole diameter depending on the explosive charge density, the rock density, the rock ultimate compressive strength, the longitudinal wave velocity in the blasted rock, and the commercial explosive detonation velocity was established. With increasing the explosive charge density, the rock ultimate compressive strength, and the explosive detonation velocity, the explosive charge diame-

ter in various rocks decreases, whereas with increasing the longitudinal wave velocity in the blasted rock, the charge diameter increases.

4. The dependences of the presplitting blasthole charge linear mass on the longitudinal wave velocity in the blasted rock, the explosive density, detonation velocity of commercial explosives, and the rock ultimate compressive strength have been established. With increasing the longitudinal wave velocity in the blasted rock, the explosive density, and the presplitting blasthole radius, linear mass of the presplitting blasthole charge increases, and with increasing the commercial explosive detonation velocity and the rock ultimate compressive strength, the charge linear mass decreases.

5. The presplitting blasthole spacing was determined depending on longitudinal wave velocity in the blasted rock, the explosive density, the rock ultimate compressive strength and tensile strenght, and the presplitting blasthole radius in different rocks. With increasing the longitudinal wave velocity in the blasted rock, the rock ultimate compressive strength, and the presplitting blasthole radius, the presplitting blasthole spacing increases, and with increasing the rock tensile strength, the spacing decreases.

6. A technique has been developed for experimental studies of blasting contour blasthole charges (blasting by pre-splitting method) using models, allowing to study fracturing on volumetric models and wave interaction by the method of high-speed video recording of the blasting process in transparent models, as well as to determine the parameters of stress waves during blasting in samples of real rocks.

7. A method for the formation of stable pit wall slopes has been developed and commercially implemented, which enables ensuring high quality of bench presplitting, complete integrity of the peripheral (beyond-contour) rock mass, and mining safety.

8. The excavator method for bench presplitting at ultimate pit contour has been developed and commercially implemented, which enabled increasing the bench slope angle from 60 to 65°, decreasing the overburden volume, ensuring integrity of the peripheral (beyond-contour) rock mass and mining safety.

9. The method of initiation of blasthole charges in the near-contour zone of an open pit, which provided acceptable seismic load on the pit walls and engineering structures without reducing the effect of rock fragmentation and with ensuring a preset size of average fragment of the blasted rock mass.

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