Physicotechnical substantiation of parameters of blasting operations in deep open pit mines

Fizikalnotehniška utemeljitev parametrov miniranja v globokih površinskih kopih

P. A. Shemetov[1], I. P. Bibik[1, *]

[1]Navoi Mining Metallurgical Combine, Central Mining Board, Administrative building, Zarafshan town, Navoi region, Uzbekistan

*Corresponding author. E-mail: I.Bibik@cru.ngmk.uz

Received: March 28, 2012 **Accepted:** August 28, 2012

Abstract: Physicotechnical substantiation of parameters of blasting operations has been developed giving due consideration to the effect of increasingly complicated factors of deep open pit mines of Uzbekistan on efficiency of mining operations, protection of the rock solid and engineering structures from seismic load of bulk blasts, improving safety in production and use of emulsion blasting explosives. There has been developed an algorithm of calculation of rational elements of placing of blasthole BE charge and size composition of broken rocks. Researches have been made to improve formulation of matrix emulsion and emulsification process of feedstock solutions. A procedure of high bench blasting has been developed. A blasting procedure for rocks with different bastards has been developed. A seismically safe for adjacent protected objects method of rock blasting without decreasing rock breaking effect has been recommended. Based on researches and analysis of risk level of EBE plant for the purpose of unconditional provision of safe production of EBE a comprehensive safety. System has been developed for safe operation of the EBE plant.

Izvleček: Fizikalnotehniška utemeljitev parametrov miniranja je bila izvedena ob vsestranskem upoštevanju naraščajoče zapletenih razmer v čedalje globljih površinskih kopih v Uzbekistanu in z ozirom na učinkovitost minerskih operacij ter ohranjanje obstojnosti kamnine in varnosti tehničnih objektov v razmerah seizmičnih vplivov zaradi masovnega miniranja in glede na zagotavljanje varnosti pri izdelavi

Original scientific paper

in uporabi emulzijskih minerskih eksplozivov. Razvit je bil algoritem za izračun elementov pri nameščanju eksplozivnih nabojev v minskih vrtinah z ozirom na željeno zdrobljenost dane kamnine. Opravljene so bile raziskave z namenom izboljšati pripravo matrične emulzije eksplozivov iz surovinskih raztopin. Razvit je bil postopek miniranja na visokih etažah in dalje, postopek miniranja v kamninah z različnimi trdimi vključki. Predlagana je metoda miniranja, ki je seizmično varna za varovane objekte brez zmanjševanja učinkovitosti drobljenja kamnin. Na osnovi raziskav in analize tveganja delovanja proizvodne postaje za varno pripravo emulzijskih eksplozivov je bil izdelan celovit varnostni sistem za varno obratovanje take postaje.

Key words: deep open pit mines, increasingly complicated factors, efficiency of mining operations, bulk blasts, parameters of blasting operations.
Ključne besede: globoki površinski kopi, čedalje bolj zapletene razmere, učinkovitost rudarjenja, masovno miniranje, parametri miniranja

Introduction

Development of mining industry in Uzbekistan is inseparably associated with development of mineral deposits by open-pit method. As is known, the considerable part of open pits for opencast mining of minerals has entered a category of "deep" and the tendency still persists. Parameters of current open pit mines have essentially increased. Large deep Muruntau and Kalmakyr open pits located on the territory of Uzbekistan are in the world mining practice unique for complexity and novelty of the solved in the course of their creation and operation scientific and technical problems defined by peculiarities of mining-and-geological and mining-technical conditions of deposits development. There are predesign options of profitable mining and further development of Muruntau and Kalmakyr open pits to the depth of 900–1000 m.

Meanwhile, it is known that with increasing depth of open-cast mining of ore deposits up to the utmost economically expedient level there become more sophisticated mining-and-geological and mining-technical conditions of operations, water content and fissuring of rocks increase, influence of mine depth on ore resistibility to explosive rupture grows, and requirements to safety of engineering structures installed on deep horizons are raised and etc. In whole, design of parameters of drilling and blasting operations (DBO) in deep open pit mines must take into consideration physical-mechanical and physicotechnical properties of rocks changing with mining depth, applicable blasting explosives (BE), seismic load of bulk blasts on safety of pit walls, engineering structures and services, integrity of the ore-hosting solid and mine openings with involvement into combined open-and-underground mining of ores deposited beyond the final pit boundary near the pit edges and under the pit bottom, when the openings (adits, inclined shafts and etc.) may be driven from the exhausted area of the pit.

Meanwhile, having analyzed the current status of drilling and blasting system allowing for regularities of change in the rock solid blastability as excavation depth of Muruntau and Kalmakyr open pits increases, it has been determined that [1]:

- known methodology for calculation of blasting parameters for the rock solid in deep open pits does not fully ensure the required quality of rock breaking;
- there are no criteria of effectiveness of applicable emulsion BE using low-price components manufactured in the Republic of Uzbekistan;
- known procedures of blasting and breaking of rock mass are insufficient to increase completeness of mineral extraction;
- known DBO parameters design methods do not fully meet specified criteria and limitations to ensure protection of the pit near-edge solid and engineering structures from seismic load of blasting;
- methodology used for calculation of parameters of safe protective pillar between open pit bottom and underground mine openings underestimates seismicity of the near zone being of great importance during concurrent open-and-underground mining of deposits;
- existing safety systems in production and application of BE with controlled energy in deep open pits do not ensure safety of operations to the full.

Thus, assessment of DBO for opencast mining of deposits up to the utmost economically expedient level shows that improvement of effectiveness of drilling and blasting system and optimization of parameters of DBO processes is feasible through:

- effectiveness of explosive breaking and displacement of rock mass;
- development of formulation with cheaper and more high-energy emulsion BE using ingredients manufactured in Uzbekistan, which characteristics suit the broken rock to the full;
- development of blasting procedures enabling to improve quality of rock mass breaking and to enhance completeness of commercial mineral extraction;

- development of efficient parameters of DBO ensuring minimization of seismic load of blasting on the near-edge solid and engineering structures;
- development of comprehensive safety system in production and use of emulsion BE with controlled energy.

MATERIALS AND METHODS

Determination of the basic mechanism of how parameters of blasting and physico-mechanical properties of the solid have effect on the intensity of breaking and displacement of rock masses. The basis for solving the problem of determination of basic mechanism of how the parameters of blasting and physico-mechanical properties of the solid have effect on the intensity of breaking and displacement of rock mass is the information about extent of breaking, which is based on taking into account both physical-mechanical and mining-technological properties of rocks and ores in light of block structure and fissuring of the solid.

Experimental dependences developed for conditions of Muruntau and Kalmakyr deep open pits demonstrate that change in size of natural jointing in the rock solid with increase of depth is expressed by the following correlating equation:

$$D_i = 0.306 + 0.0036H \qquad (1)$$

where H – depth of occurrence of the stratum under study (m).

Results of researches have shown that rocks of Muruntau and Kalmakyr open pits can be grouped and classified by four categories of rocks in terms of block structure and extent of fissuring: small-block, medium-block, large-block, highly-large-block rocks, coinciding with categories of blastability. Given that in calculations of DBO parameters the block structure is characterized by the average (weighted average) size of the jointing (block) in the solid, there have been developed a methodology and algorithm of calculation of size composition of broken rocks based on the economic-mathematical description of block structure of the solid and on IBWC classification. In accordance with the classification the curves have been defined, i.e. the lines delimiting the solids (massifs) of rocks of deep open pits by fissuring categories which are generally described by the equation [2]:

$$R_{i_\text{M}} = 100 \cdot \exp\left[-3 \cdot k^2 \cdot D_i^{\lambda}\right], \% \qquad (2)$$

where R_{i_M} – content of blocks with size of more than D_i; k – empirical factor characterizing the average size of the jointing $D_{\text{ave(s)}}$ in the solid; λ – empirical coefficient characterizing fissuring (block structure) of the solid:

$$\lambda = \frac{0.667}{(k + 0.05) + 2} \quad (3)$$

Between the empirical factor *k* and average value of the jointing in the solid there is the following dependence:

$$k = \frac{0.5}{D_{ave(s)}^{1.05 + D_{ave(s)}}} \quad (4)$$

Using dependences (2) (3) (4) it is possible to determine a distribution of jointings for various rocks in wide range of fissuring, particularly for Muruntau open pit with correlation coefficient r_c = 0.71–0.84:

$$\lambda = 1 + \frac{0.667}{k + 0.05}, \quad k = \frac{0.35}{D_{ave(s)}^{1,05 + D_{ave(s)}}} \quad (5)$$

Given the foregoing, there has been developed an algorithm of calculation of rational elements of placing of blasthole BE charge and size composition of broken rocks. Use of the algorithm has enabled to determine a mechanism of effect of DBO parameters, physicotechnical properties of rocks and BE explosive characteristics on efficiency of breaking and displacement of rocks during blasting in deep open pit mines. At the same time, improvement of blasting efficiency and enhancement of mining safety and decrease in expenses by 7–8 % are being achieved by application of emulsion BE using inexpensive ingredients manufactured in the Republic of Uzbekistan, which energy and detonation characteristics suit physicotechnical properties of hard rocks to the fuller extent.

There has been developed a procedure for determination of throw coefficient during rock displacement by explosion of blasthole charges under conditions of deep open pit mines depending on specific BE consumption, blasthole angle towards the sky line, stope width and bench height. As a result of statistical manipulation of pilot work findings, by this procedure the following formula for calculation of throw coefficient during displacement of rock by explosion of blasthole BE charges with correlation coefficient 0.94 ± 0.012 has been derived:

$$K_{thr} = \frac{q(0.4\sin 2\alpha_c + 0.65)(0.5 - 0.016m)}{0.01\sqrt[3]{Ah} + q} \quad (6)$$

where α_c - blasthole angle towards the sky line (°); *m* – thickness of hard seams (m); *A* – stope width (m); *h* – bench height (m); *q* – specific BE consumption (kg/m³).

Specific BE consumption can be regarded as an integral characteristic reflecting compressive resistance of rock and size of an average fragment (lump) of rock in a shotpile of broken rock mass. As a result of processing of statistical data and experimental and industrial blasts the following design formula has been derived:

$q/(\mathrm{kg/m^3}) = (0.01 - K_a \sigma_{com} \ln d_{ave})$ (7)

where K_a – coefficient of adaptation to certain open pit conditions, $K_a = 0.0034$ for Muruntau open pit and $K_a = 0.0036$ for Kalmakyr open pit; σ_{com} - compressive resistance of rock (MPa); d_{ave} - average size of rock fragment (lump) in a shotpile (m).

Achieving the required quality of rock loosening by blasting under conditions of reducing width of blocks blasted in deep open pits solely by adjusting the specific consumption of BE is practically impossible, since spatial placing of blasthole charges have a considerable effect on results of blast. Moreover, the main initial parameter is a blasthole diameter determining the zone of controlled breaking of rocks by blasting. As a result, design formulas have been determined for a blasthole diameter and for interrelated with it other parameters of charge placing: lines of the least resistance of rock to blast, size of blasthole BE charge above the bench toe, extent of sub-drilling and uncharged part of blasthole (plug), spacing between blastholes in a row and between the rows of blastholes:

$$D = \frac{H_y Ctg\alpha + C_b}{K_f \sqrt[3]{K_e}}, \mathrm{m} \quad (8)$$

where D – blasthole diameter (m); H_y – bench height (m); α – bench slope (°); C_b – distance between blasthole axis and bench crest (m); K_f - coefficient of adaptation of blasthole BE charge diameter to mining and technological characteristics of broken rock and conditions of charge initiation by non-electric triggering system (NTS),

$K_f = \dfrac{55}{\sqrt[4]{0.1\sigma_{com}}}$, $K_e = K_{BE} \cdot K_\Delta$ - BE relative energy concentration factor, for power industrial BE with high charging density $K_e > 1.0$, for reduced-power industrial BE with low charging density $K_e < 1.0$; $\dfrac{Q_V}{Q_{V_0}} = K_{BE}$ – energy conversion factor of the applied BE; Q_V, Q_{Vo} - energy of the applied BE and the reference BE (kJ/kg); $K_\Delta = \dfrac{\Delta_a}{\Delta_r}$ - conversion factor of charging density; Δ_a, Δ_r - charging density of the applied BE and the reference BE.

There have been determined changes of resistance along the bench toe during explosion of blasthole BE charge from its effective length above bench toe:

$W/\mathrm{m} = 0.387e^{0.3} l_{ef}$ (9)

where l_{ef} – effective length of blasthole charge above bench toe (m).

Adjustment of spatial arrangement of blasthole charges with specific BE consumption is made by alignment of spacing between blastholes that at $a = b$ is determined by the formula:

$$a_d = \sqrt{\frac{Q}{qH_y}}, \mathrm{m} \tag{10}$$

where a_d – adjusted spacing between blastholes (m); Q – BE weight in blasthole (kg).

Blasting operations conducted in Muruntau deep open pit in accordance with the developed procedure in view of changing physico-mechanical properties of the solid with increase in the pit excavation depth have shown evener breaking of the solid at reducing BE costs by 12 %.

Substantiation of DBO parameters using emulsion BE providing improvement of effectiveness and safety of blasting operations. Due to the fact that in Muruntau and Kalmakyr open pits only in-house produced blasting explosives are being used, all regulations, normative documents and analytical dependences characterizing design values of specific BE consumption and DBO parameters have been amended so as to take into consideration relative explosive force of the applied BE in relation to the reference BE (Grammonit 79/21), which analysis has shown that at emulsion BE efficiency factors equal to 1.05 (Nobelit 2030) and 1.07 (Nobelan 2080) the charge column length in a blasthole shortens by 32 % and 27 %, specific energy consumption decreases by 26–30 %. In this regard, abatement of extent of effective utilization of explosive energy of blasthole charge for breaking and expansion of zone of uncontrolled breaking is apparent. For the purpose of neutralizing these negative factors it has been proposed to compensate the shortening of charge column length by using a combination of blasthole emulsion BE charge +ANFO and by inverse initiation by blasthole detonator of non-electric triggering system (NTS), during which the primary pre-breaking of the solid mass is done by high brisant emulsion BE, and final phase of breaking - by low brisant BE, i.e. by ANFO. In such case, later ignition of ANFO locks up explosion products thus extending their duration in charge chamber and accordingly facilitating the improvement of effective utilization of explosion energy for breaking and working of bench toe. Pilot works have recorded that application of combined charge structure improves its explosive force as compared to single charge structure by 15 %, at the same time displacement of ore boundary and host rock decreases as much as 1.5–2.0 times, rock breaking quality improves with simultaneous reduction of BE costs by 20 %.

For the purpose of improvement of EBE and WEBE application efficiency and reduction of cost price of explosive rupture of rocks in deep open-pit mines, researches have been made to improve formulation of matrix emulsion and emulsification process of feedstock solutions aimed at possibility to use inexpensive raw ingredients made in Uzbekistan. Assessment of emulsion physical stability using various samples of emulsifiers has demonstrated that cost of ingredients, their market availability and acceptance analytical test of raw ingredients or manufacturers' guarantee of conformity with quality standards and technical requirements of process procedure must be the criteria for selection of ingredients for oil fraction used for production of emulsion at the EBE plant.

With the view of reducing expenditures for purchase of raw ingredients and expendable materials some researches have been conducted to optimize formulation and manufacturing method of emulsion explosive formulations (EEF) which made it possible to reduce percentage of imported raw ingredients from 5.2 % to 4.5–4.0 % and to modernize equipment and instrumentation in manufacturing scheme of the EBE plant, thus enhancing efficiency and safety of EBE production.

As for priorities for further work on improvement of formulation for EBE production, there have been offered EEF where calcium nitrate is used to increase explosion energy, and carbamide inhibiting interaction of EEF and sulphide rock is added into oxidants solution, in case of lack of sulphides in rock it can be replaced with ammonium nitrate. It is recommended to avoid application of wax, paraffin, acetic acid, sodium nitrite, sodium lye solution, thiourea, Swedish porous nitre which should be replaced with petrolatum, microspheres or foamed polystyrene. According to economic estimation, the EBE produced by the recommended formulation costs by 10–25 % less as compared to similar energy characteristics. Cost of 1 t of matrix in terms of raw ingredients is reduced by USD 130 for machine-aided charging and by USD 78–96 for cartridged EBE. For improvement of efficiency and safety of EBE production the sections such as liquid ammonium nitrate intake section, cartridge cooling section, oil solution makeup section and etc. have been modernized and reconstructed based on performed pilot works.

As a result of research works the DBO parameters have been determined to provide optimal placement of EBE charge in the solid mass, basic values of blasthole network size depending on used drilling tool and potential of their applicability. Thus, for hard-to-blast ore zones in Muruntau open pit: the network size of 6.0 m × 6.0 m is for 250 mm diameter blastholes; specific BE consumption in dry blastholes is 1.10–1.25 kg/m^3 (Nobelan 2080), in watered blastholes it is 1.15–1.30 kg/m^3 (Nobilit 2030); stemming space length is 5–6 m. In the rock area of the open pit mine it is possible to apply the 250 mm diam. blasthole network having size of 6.5 m × 6.5 m at specific BE consumption of 0.98–1.05 kg/m^3 and stemming space length of 5.5–6.5 m. The developed parameters of DBO adapted to EBE have enabled to reduce expenditures for blasting by 15 % with the best handling of the rock solid along the whole height of the blasted bench.

Development of effective blasting techniques enabling to improve quality of rock mass breaking and to enhance completeness of extraction of commercial mineral. Intensification of geomechanical effects with increasing mining depth has impact on achieving the target quality of breaking. This imposes further requirements to explosive rupture of rock which are implemented by means of new methods and parameters.

Researches on quality of rock mass breaking have demonstrated that even observing the determined optimal parameters of charge placing in the solid and at increased specific EBE consumption (q): on average for Nobelan 1.11 kg/m^3 (maximum value is 1.27 kg/m^3); and for Nobelit 1.26 kg/m^3 (maximum value is 1.59 kg/m^3) it is impossible to avoid yield of oversize and coarse size. Moreover, at $q \geq$ 1.15–1.20 kg/m^3 as a result of predominance of throwing (propellent) effect of an explosion over rending effect a considerable portion of rock mass is thrown from the side of mined-out area (10–20 %) on lower horizons or formation of shotpiles as wide as up to 40 m takes place. In the first case this results in decrease of quality and amount of ore, and in the second case this causes drop in intensity of mining operations.

In this regard, researches have been conducted to increase duration of explosion effect on the upper part of bench and to enhance stemming efficiency. Specifically, to increase duration of explosion effect on the solid mass and to enhance locking effect it is recommended to use dynamic stemming consisting in explosion of BE charge in an additional short blasthole drilled at 2 m distance off the main blasthole or explosion of locking charge in stemming space of the main blasthole. NTS was used to initiate BE charges: in blastholes by nonelectric explosion initiating device-B (blasthole) with delay of 500 ms; the surface network was arranged by nonelectric explosion initiating device-S (surface) with delays of (25, 42 and 67) ms. On experimental areas the delays were selected in such a manner that the locking charge was the first to explode and then the main (primary) charge exploded. Retardation of the main charge relative to locking charge with placing the latter in the main blasthole was maintained by various length of shock-wave tubes (SWT) for lower and upper boosters with their concurrent initiating by dentonator of surface network and was about 7 ms at 2000 m/s detonation speed in SWT. In order to delay the primary charge relative to locking charge in aditional blasthole a combination of retardants rating (25, 42 and 67) ms in surface network was used. As a result, an interval between explosion of the primary charge and locking charge was between 17–25 ms. It has been established that application of locking charge in additional blasthole has increased extent of rock solid breaking by 8–10 % and reduced specific BE consumption by 3 %.

Application of emulsion explosive formulations (EEF) with controlled volume concentration of explosion energy applicable for charging dry and watered blastholes, together with combined structure of charges makes it possible to meet in practice any technological challenges of DBO owing to available park of drilling rigs. This circumstance promotes increasing height of benches that allows of reducing quantity and length of transport horizons, increasing open pit wall slope, enhancing intensity of mining in deep open pits. Moreover, engineering and economical performance of open pit mining is increasing.

A procedure of high bench blasting has been developed, which main point consist in the following. Rocks are broken by a pair of divergent blasthole charges. Specifically, one blasthole in each pair is drilled perpendicularly to bench toe, and the second one – towards bench slope with incline to the toe. Vertical and inclined blastholes are arranged in parallel vertical planes spaced apart by distance equal to 1–2 diameters of blasthole equivalent in terms of charge energy to total charge in a pair of divergent blastholes. Blasthole angle and limit bench height are determined by the following formulas:

$$\beta = arctg\frac{1.13H}{d_e}\sqrt{\frac{q}{\gamma}}, \deg ree;$$

$$H = \frac{2d_e\sqrt{\frac{\pi\cdot\gamma}{q}} - 2c}{2\cdot ctg\alpha}, \mathrm{m} \qquad (11)$$

where α, β – angles of bench slope and inclination of blastholes in cluster; H – bench height; d_e – diameter of charge equivalent in terms of charge energy to total charge in a cluster of blastholes; q – specific BE consumption; γ – density of charging BE in a blasthole; c – safety berm.

Effectiveness of the developed procedure of high bench blasting has been tested experimentally during reactivation of western wall of Muruntau open pit mine. An area of the pit wall to be blasted included four 15 m benches, which were combined in two 30 m benches during the experiment. Diameter of blastholes was 250 mm. Blastholes network was 7 m × 7 m. Specific BE consumption was 0.95 kg/m^3. Charge structure was combined (EBE and ANFO). In a pair of divergent blastholes the BE charge was placed in vertical and inclined ($\beta = 65^\circ$) blastholes.

Study of size composition of the blasted rock mass by photo-planimetric method has shown that average size of fragments of rock mass does not exceed 22–25 cm (under similar conditions on benches of 15 m height it was 27–30 cm.)

A wide range of changing mining-and-geological properties of rocks in deep open pits requires an individual approach during blasting of rocks with different bastards occurred in less hard hosting rocks. When blasting the solid with hard interlayers on narrow operating benches in a large deep open pit it is necessary to ensure their proper blasting and formation of compact shotpile that is uniform by fractional composition. This can be achieved by placing BE in areas of occurrence of bastards or by increasing duration of explosion effect. These two approaches are combined in the developed technology of blasthole BE air-cushioned charge with concavity in bottom part of the charge. In order to raise the charge to the level of bastards, in the bottom part of the blasthole an air cushion of 1–2 m length is formed, and to blast the lower part of the solid a concavity is made in the bottom part of BE charge. BE charge is initiated at height of 2/3–3/4 of the charge length. An air gap made with concavity forms a jet which creates a chock wave directed downwards and sidewards off a blasthole. In turn, stress waves from cumulative (cavity) charge directed upwards produce secondary breaking (fragmentation) of upper rock. Directing energy of a part of a BE charge towards bottom of blasthole increases duration of explosion effect on the solid and forms two stress waves having impact on the whole solid. Conducted researches based on physical simulation and pilot blasting have established that more efficient placement of BE charge is at the depth off the bastard up to 1/2 its thickness, at the seam thickness of 1.5 m; and at 1/3 its thickness at the seam thickness of less than 1/4 bench; immediately under the seam or partially in the seam at the seam thickness of more than 1/4 bench height.

A blasting procedure for rocks with different bastards has been developed, as a result of which the effectiveness of breaking different bastards (hard inclusions) has increased due to taking into consideration the main properties of hosting rocks, bastards and used BE and is achieved by drilling additional blastholes to main holes inside bastard contour with placing BE charges in them inside bastards. Selection of BE for charging additional blastholes is made by value of BE detonation velocity determined by the following correlation:

$$D_{\partial} = D_0 \sqrt{2\frac{\sigma_t^{bast}}{\sigma_t^{hr}} - 2\sqrt{\frac{\sigma_t^{bast}}{\sigma_t^{hr}}} + 1} \quad (12)$$

where D_{∂} − BE detonation velocity for charging additional blastholes (m/s); D_0 – BE detonation velocity for charging main blastholes (m/s); σ_t^{bast} – ultimate tensile strength of bastard rock (Pa); σ_t^{hr} – ultimate tensile strength of hosting rock (Pa).

Additional blastholes are drilled to the depth of:

$$l_{\partial} = \frac{\sum_{i=1}^{n} l_{on_i}}{n} - (1.5...5.0)d_{adhole_{\partial}} \text{, m} \quad (13)$$

where l_{on_i} − mark of bastard bottom by depth of main blasholes, between which a respective additional blasthole is arranged (m); n − number of main blasholes, between which a respective additional blasthole is arranged; $d_{adhole_{\partial}}$ − diameter of additional blastholes (m).

Both incomplete drilling of additional blastholes up to bastard bottom to length of 1.5–5 diameters of blastholes and incomplete charging to the same length up to bastard ceiling eliminates orientation of explosion action towards hosting rock having lesser resistibility to blasting.

Experimental blasting of rock in Muruntau open pit by bastards using the developed technology of blasthole BE air-cushioned charge with concavity in its bottom part and method of blasting of rocks with different bastards have shown effectiveness of the solid breaking with formation of compact and uniform by fractional composition shotpile with decreasing specific consumption of BE by 7–8 %.

Determination of blasting operations parameters providing safety of the near-wall solid and engineering structures from seismic load of blasting. It is known that during blast loosening of rock there are seismic waves of huge intensity having impact on safety of the near-wall solid and engineering structures [3].

Generally accepted criterion of estimation of blast seismic effect is medium displacement velocity (U):

$$U = K[\sqrt[3]{Q} / R]^n \quad (14)$$

where K – soil conditions coefficient; n – factor of extent of seismic wave attenuation characterized by the dependency: $n = 2 - \frac{\mu}{1-\mu}$; μ - Poisson ratio.

Safety of the near-wall solid and engineering structures will be ensured if deformations of the wall solid and foundations of protected structures caused by explosion effect do not extend the limits of horizontal component of vibration velocity $U_{adm.} \geq U_x$, whose dependence on horizontal distance off the blast point (R) is described by the equality $U_x/(\text{cm/s}) = 450 \cdot R^{-1,85}$, and the estimate of tolerable vibration velocity will be:

$$\sigma_{adm.} = \rho \cdot C_p \cdot U_{\text{adm.}} \tag{15}$$

where $\sigma_{\text{adm.}}$ - stress at which a sample of the open pit rock is not ruptured by multiple dynamic impacts; ρ - average rock density; C_p – mean value of velocity of medium displacement along the horizontal component on longitudinal wave front; $U_{\text{adm.}}$ - mass velocity of the solid vibrations (displacements) taken as admissible criterion of an estimate of seismic explosion wave impact on the solid.

Having set values of distance between blast point and protected object it is possible to determine seismically safe weight of charges for instant detonations. In addition, to avoid interferences of seismic waves it is required that delay intervals (t_3) exceed lifetime of plus phase of seismic waves:

$$t_3 \geq K_t \cdot \lg R, c \tag{16}$$

where: K_t – coefficient allowing for rock hardness, $K_t = 0.01–0.03$.

A seismically safe for adjacent protected objects method of rock blasting without decreasing rock breaking effect has been recommended. As an example let's consider an implementation of seismically safe technology for conducting DBO in Muruntau open pit, where a task of prevention of destruction of five objects located on the pit wall at the distance of 120 m; 750 m; 720 m; 480 m; 80 m off centre of the blast block was solved. For this purpose, a block to be blasted has been chosen on horizon +375 m and a project for conducting the DBO within the zone of adjacent objects with achieving the design performance of seismic loads in different directions off the centre of the blasted block has been plotted graphically on computer. The block to be blasted is prolate shaped with narrowed ends characterized by two parameters – length and width. Given that the block to be blasted is surrounded by several protected objects, a distance R_i (i = 1…. n) to protected objects is measured by connecting centre of the block to be blasted with protected objects. Amongst R_i distances the shortest one is chosen. The prolate shaped block is oriented with the long side in the line of the nearest protected object $R_{\min}$ (Figure 1).

After the drilled block has been oriented accordingly, rows of blastholes in plan view are inclined at an angle within the range from 45° to 135° to

where R1 = 120 m; R2 = 750 m; R3 = 720 m; R4 = 480 m, Rmin = 80 m: blasted block-protected objects distance; Перегрузочный пункт - Dumping station; Борт карьера - Pit wall; КНК-270 – High-angle conveyor-270; ЦПТ – Cyclic-and-continuous process; ДПП – Crushing-and-conveying plant

Figure 1. Orientation of an area of the blasted block relative to some protected objects

Table 1. Results of mathematical modeling of seismic load on protected objects of the open pit

Configuration of blasted block	Seismic load rate					Extent of stability of an object				
	R_1	R_2	R_3	R_4	R_{min}	R_1	R_2	R_3	R_4	R_{min}
Trapeze	37.1	21.8	20.7	29.2	37.8	–	+	+	+	–
Rhomboid (α = 90°)	20.5	12.9	16.1	18.9	26.4	+	+	+	+	+
Rhomboid (α = 120°)	18.3	10.5	14.2	16.4	24.1	+	+	+	+	+

the bee-line (the shortest distance) to the object. Wiring-up of the explosive circuit is made so that it could lead to successive detonation of charges in the direction from the chosen protected object to centre of the blasted block.

Arrangement of blastholes on the chosen block within the prolate-shaped area oriented in plan view with narrowed ends along the bee-line to the chosen protected object enables, due to optimal shape of the area and angles of inclination of blasthole rows in the range of 45–135°, to achieve redistribution of released seismic energy by width and length of explosion field.

Calculated by mathematical modeling rates of seismic load and findings on objects safety in case of trapeze- and rhomboid-shaped blast block at an angle of blasthole rows inclination $\alpha = 90^\circ$, $\alpha = 120^\circ$ are presented in Table 1.

The analysis of the findings shows that when a trapeze configured block is blasted, three protected objects stand seismic load and two are destroyed, and when an elongated rhomb configured block is blasted, the whole group of protected objects stand seismic load.

It has been established that blasting method, including drilling of blasthole rows in a block of definite configuration, their charging and short-delayed blasting, in which case the blastholes rows are commuted in a block having an elongated configuration view in plan with narrowed opposite ends at an angle of $45^\circ \leq \alpha \leq 135^\circ$ to a beeline to one of the chosen protected objects, provides reducing seismic load on protected objects:

- in the direction of commencement of blasting the rows – twice;
- to the left and to the right of the explosion field in 120° sectors - four times.

Successive detonation of blasthole rows with delay from the nearest hole in the direction from the protected object results in redistribution of released seismic energy in such a manner that the maximum flow of seismic energy is directed towards the side opposite to the protected object. This fact has been determined experimentally. When length of rows in the blasthole block is reduced, a number of blasthole rows are increased to maintain the total target weight of blasted BE and a total number of rows, that leads to decrease in extent of seismic load and concurrent increase in seismic load duration. The two said factors allow of adjusting flow of seismic energy on protected group of objects so that the shock effect on the nearest object could be minimal and the other nearby objects could be practically safe.

Using the open pit space for mining of ore deposits outside the pit contour enables to implement advantages of combined open-and-underground mining method to the full. At the same time, conventional approach to design and operation of deep open pits does not take into consideration changing mining methods in view of development of underground mine. In this regard, researches have been conducted to determine safe parameters of arch pillar separating open and underground mining operations. In calculations it is deemed that rock is the elastoplastic solid medium. To ensure stability of the arch pillar solid it is required that seismic load does not cause irreversible deformations. The condition of maintenance of rock stability is written as follows:

$$U \leq U_o \quad (17)$$

where U – velocity of rock displacement caused by explosion (cm/s); U_o – safe velocity of rock displacement based on conditions of their elastic deformation (cm/s).

Velocity of rock displacement U depending on charge size, distances to protected object and elastic impedance of rock can be expressed by the following formula:

$$U = \frac{3.72 \cdot 10^6 \cdot Q^{0,4}}{R^{1.75} \cdot \sqrt{C_p \cdot \rho}} \quad (18)$$

where Q – weight of simultaneously blasted charge (kg); R – distance to protected objected (m); $C_p \cdot \rho$ – elastic impedance of rock (m/s·kg/m³).

Safe velocity of rock displacement can be determined from general conditions of their deformation. If in the course of deformation of volume V_o to volume V_1 the specific energy of volume V_o increases by some final and quite definite for certain medium value F, this forms necessary and sufficient conditions for rupture. Then, general equation of energy conditions defining potential rupture of medium can be presented as follows:

$$\frac{W_1}{V_1} - \frac{W_0}{V_0} \geq F \quad (19)$$

where W_0 and W_1 – energy of medium before and after rupture (kg m); V_0 and V_1 – volume of medium before and after rupture (m³).

Rupture (breaking) stress can be defined as follows:

$$\sigma_p = -\frac{3}{8} \cdot K \left[\left(\frac{V_0}{V_1} \right)^{4/3} - \left(\frac{V_0}{V_1} \right)^{4} \right] \quad (20)$$

where K – coefficient depending on properties of massif.

Stress on elementary surface inside the rigid body acts on both normal and tangent. The body referred to three perpendicular axes Q_x, Q_y, Q_z is affected by at least three stress components σ_1, σ_2, σ_3 causing 6 more components σ_{xy}, σ_{yz}. σ_{xz}, σ_{yx} and etc. Action of these very components of stress causes deformation of volume V_0 to V_1 during its change from ρ_0 to ρ_1.

Given that $\frac{\sigma_1 + \sigma_2 + \sigma_3}{E}$ in whole defines volumetric strain (relative volume deformation) (ε) of the medium under action of total stress tensor (σ_1, σ_2, σ_3), we obtain:

$$\sigma_p = -\frac{3}{8} \cdot K \cdot \frac{\left\{ \left[1 + (1 - 2\mu) \cdot \varepsilon \right]^{8/3} - 1 \right\}}{\left[1 + (1 - 2\mu) \cdot \varepsilon \right]^{4}} \quad (21)$$

where E – dynamic modulus of elasticity; σ_1, σ_2, σ_3 – total stress tensor; negative values of σ_p conform to compression stress as in this case deformations are positive, and positive values of σ_p conform to tensile stress as deformations are negative here.

Use of dependence (21) is of great importance for practice as besides general qualitative dependences in determination of parameters of rupture process, we take into account the effective (working) stress σ_p related to the known dependence from theory of elasticity:

$$\sigma = \frac{U \cdot C_p \rho}{q} \tag{22}$$

that enables to determine safe velocity of rock displacement U_o:

$$U_o = \frac{12{,}5 \cdot C_p \cdot \left\{\left[1 + (1-2\mu)\cdot \varepsilon_o\right]^{8/3} - 1\right\} \cdot (1+\mu)}{\left[1 + (1-2\mu)\cdot \varepsilon_o\right]^4 \cdot (1-\mu)} \tag{23}$$

where C_p – velocity of P-wave propagation (m/s), C_p = 4000 cm/s; μ - Poisson ratio, μ = 0.25–0.30; ε_o – admissible relative deformation of rocks within elasticity, for conditions of the mine horizons +128 m, +78 m and ±0 value ε_o is accepted within the range of 0.0003–0.0004.

Subject to dependences (17), (18) and (23) we derive an equation for determination of required thickness of protective pillar over underground mine workings at different values of BE charge weight (Q) per one retardation:

$$R = \frac{1380 \cdot Q^{0.228} \cdot \left[1 + (1-2\mu)\cdot \varepsilon_o\right]^{2.28}}{C_p^{0.856} \rho^{0.286} \left\{\left[1 + (1-2\mu)\cdot \varepsilon_o\right]^{8/3} - 1\right\}^{0.57}} \cdot \left(\frac{1-\mu}{1+\mu}\right)^{0.57} \tag{24}$$

Results of calculation of ceiling thickness, pillar (protected pillar) are determined subject to seismic explosion effect of bulk blasting conducted in the open pit based on BE charge weight per one retardation and are 50 m for Muruntau open pit (Figure 2).

Researches have been made on seismic load of huge blasting to determine critically admissible deformations and velocities of displacements for surface engineering structures and underground mine workings. It is recommended to set an admissible relative deformation of rock within the elasticity limits in accordance with classification of protective structures by their responsibility and lifetime.

Development of comprehensive safety system in production and use of blasting explosives in deep open pit mines. It is known that main technological and operational properties of emulsion BE (EBE) depend on correct selection of formulation of oxidant, emulsifier, method of emulsion production, nature of sensibilizing additive and EBE manufacturing equipment.

It has been established that having a number of advantages over other types of industrial BE, EBE composition has some specific features: there is a number of critical parameters of external effects, which increasing leads to generation of initial site (of explosion); EBE decomposition runs at molecular level and requires no additional reagents; EBE is characterized by high energy concentration and velocity of energy output, which, as a rule, ends in explosion having effect on the environment (equipment, buildings and structures, personnel). Therefore, production and use of EBE require strict regulation as per safety requirements [4].

Level of average technical (theoretical) risk for EBE manufacturing plant has been calculated as a sum of criteria of probability (theoretical frequency) of occurrence (B) and importance of consequences of undesirable events (P) in terms of more serious events (n). The risk level was:

$$\frac{\Sigma(B_n + P_n)}{n} = 6{,}5 \text{ points.} \qquad (25)$$

Based on results of the assessment it can be concluded that the total risk level during operation of EBE plant under current conditions is acceptable like for a hazardous industrial object subject to observance of technical and organizational measures (safety system) aimed at risk mitigation. And at the same time, a source of uncertainty of risk assessment is the human factor and equipment failure.

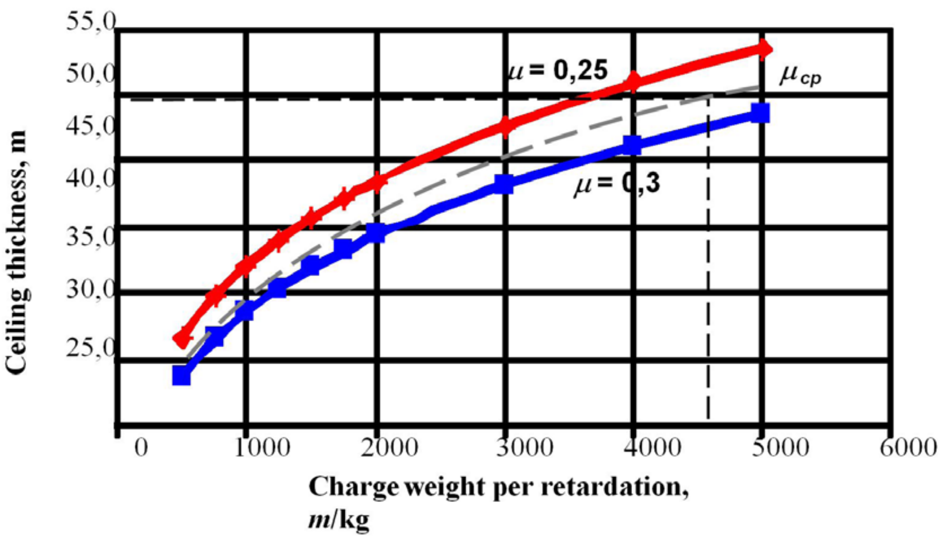

Figure 2. Ceiling thickness changing from the level of seismic explosion effect for conditions of Muruntau open pit

Based on researches and analysis of risk level of EBE plant for the purpose of unconditional provision of safe production of EBE a comprehensive safety system has been developed for safe operation of the EBE plant (Figure 3).

In addition to the active (instrumentation) safety system existing at the EBE plant, in order to maintain safe and steady operation of processing equipment, it has been recommended to include into the modernization program the following elements: backup emergency stop push button in operator's room; staff evacuation audio signal (alarm sirens and beacons) at actuation of emergency shutdown of operation; reducing maximum temperature settings on mixers and pumps from 150 ˚C to 140 ˚C; additional installation of protective systems on Netzsch stators; Ametist IP type 329-5 01 "Flame Announcer" sensors respond-

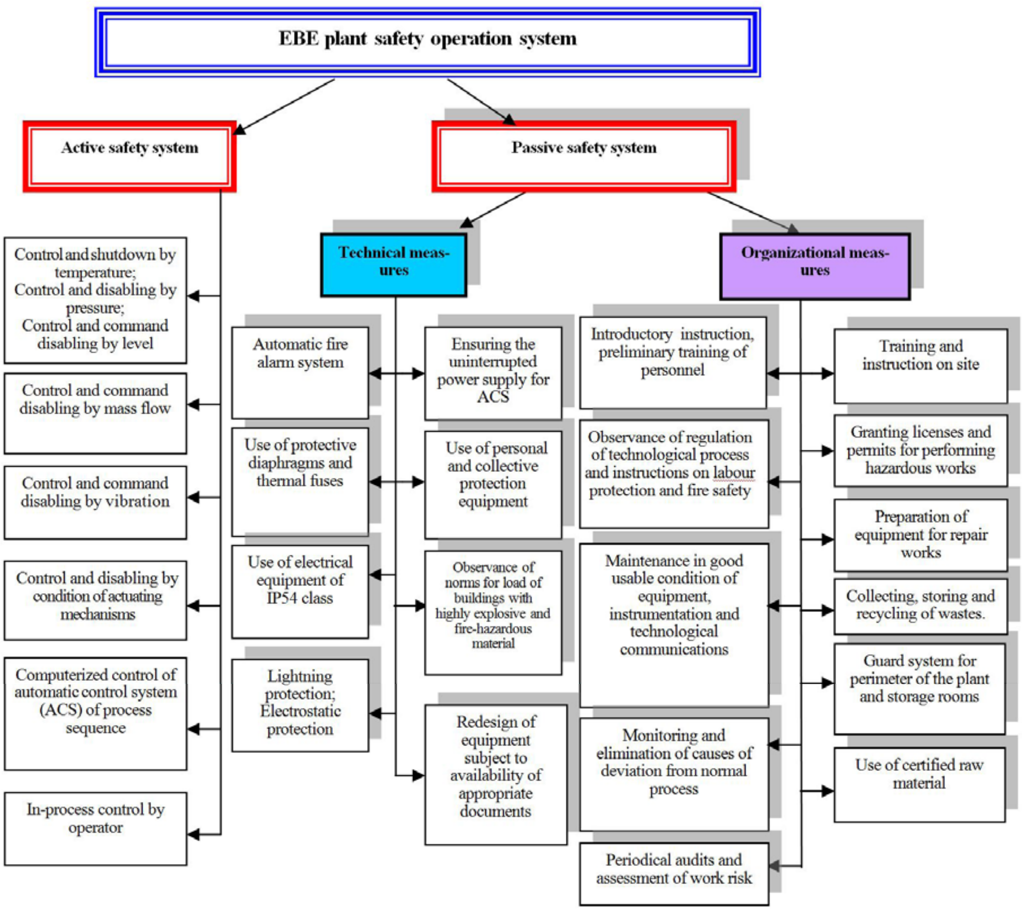

Figure 3. Recommended comprehensive safety operation system of the EBE plant

ing to open flame emergence with outputting the signal from sensors to staff evacuation audio signal (alarm sirens and beacons) should be installed in rooms for preparation of oxidizing and oil solutions, packaging, packing and cartridging equipment in addition to fire-control warning system. The developed comprehensive safety operation system made it possible to ensure safe production of EBE at the plant and accordingly reduce the existing risk level of the plant from 6.9 points to 4.0 points.

In view of the fact that parameters of formula for calculation of range of rock movement during explosion of blasthole charges at open mining operations, as set forth in effective "Unified Safety Regulations For Blasting Operations" of 1992 edition, are relative and nondimensional values, substantiation of predictive assessment of blasting operation conducted in proximity to mining equipment and engineering facilities under conditions of deep open pits in terms of damaging action of the blasted rock fragments has been conducted:

$$R_0 = R_{MOV} \times K_{STEM} + \Delta R \qquad (33)$$

where R_0 – maximum range of rock movement; R_{MOV} - predictive assessment of maximum range of rock movement; K_{STEM} - coefficient of stemming value influence on range of rock movement, $K_{STEM} = 1/(0.8 + 0.002\ L_{STEM}^{-1,8})$; $L_{STEM}^{-1,8}$ - length of stemming expressed in diameters of blasthole charge, ΔR – wind drift of blasted rock fragments to leeward side, $\Delta R = 5\ V$; V - wind velocity (m/s).

Comparative calculation of maximum range of blasted rock movement demonstrates that range of rock movement as per the developed methodology subject to use of stemming and wind drift is lower respectively by 71.5 m and 46.5 m. Change in routine of bulk blasting in Muruntau deep open pit has made it possible to enhance effectiveness and safety of blasting operations, to reduce downtime of mining transport equipment and annual expenditures for blasting operations by USD 105 thousand.

Conclusion

Thus, there have been determined theoretical generalization and solution of scientific problem of determination of regularities of explosive rupture and substantiation of DBO efficient parameters suitable for the rock to be broken in deep open pits that is of great economic importance and ensures reducing expenditures for blasting operations, increasing their effectiveness subject to application of EBE with controlled energy using inexpensive ingredients manufactured in the Republic of Uzbekistan, minimization of seismic load of blasting on the near-wall rock solid and engineering structures, improvement of breaking quality and completeness of extraction of commercial minerals from the earth. A strategy of effective development of blasting operations has been developed for mining enterprises that is aimed at enhancement of safety level of blasting operations, technology of EBE production and blastholes charging.

References

[1] Tolstov, E. A., Sytenkov, V. N., Phillipov, S. A. (1999): Processes of open-cast mining of ore deposits in rocky files. The monograph.Publishing house «Fan», 276 p., Tashkent.

[2] Rubtsov, S. K., Shemetov, P. A. (2011): Management of explosive influence on a hills at open-cast mining deposits. The monograph.Publishing house «Fan», 400 p., Tashkent.

[3] Kucherskiy, N. I. (2007): Modern of technology at development of radical deposits of gold. The Publishing house «Ore and Metals», 696 p., Moscow.

[4] Norov, JU. D., Shemetov, P. A., Bibik, I. P. (2007):Collection of practical works in a subject new technologies and safety at prosecution of explosive works. The Publishing house «Bukhara», 166 p., Bukhara.