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Simulation of the enclosing rock displacements around the development mine workings

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Abstract

Purpose is to develop and substantiate a mathematical model for forecasting of rock displacements around the development mine workings and optimization of techniques for their reinforcement.

Methods. A comprehensive approach has been applied including numerical simulation; theoretical analysis; and experiments. Special attention has been paid to rock displacements based upon stress-strain state of the rock mass. The developed KMS-Ш software was used for the calculation helping model displacements and analyze the obtained data.

Findings. Recommendations have been proposed to decrease a rock dilatancy coefficient achieved through correction of support parameters; mine working geometry; and control of rock deformation rate. It has been demonstrated that rock bolting use lowers significantly the intensity of the displacements. It has been identified that a decline in rock strength results in the increased failure zones; at the same time, the improved plastic properties minimize elastic energy accumulation reducing the displacement probability of the opposite crack surfaces.

Originality. An algorithm has been developed forecasting displacements of the mine working peripheries taking into consideration the mining, geological, and engineering factors. A mathematical model has been represented to identify both elastic and non-elastic deformations in a border zone. The dependencies between border rock mass displacements, mine working depth, and rock strength have been defined.

Practical implications. Determination of optimum parameters of rock strengthening helps minimize failure zones; better stability of mine workings; and reduce the possibility of dangerous geomechanical phenomena. Use of the proposed model makes it possible to improve mining efficiency owing to more accurate forecasting of displacements.

Keywords: *underground mining, mine workings, supports, geomechanical processes, rock bolting, stress-strain state, rock pressure, rock displacement*

1. Introduction

Kazakhstan mining industry is among the leading economic branches of the country [1]. Intensive use of raw-materials base poses challenges to mining enterprises as for ensuring stability and safety of mine workings [2]. The problem of rock displacements around workings becomes particularly relevant since it is critical while mining deep deposits where rock pressure increases significantly [3], [4].

Geological conditions of Kazakhstan differ in considerable diversity of rock masses including coal, iron-ore, polymetallic, and other deposits [5]-[7]. Karaganda coal basin, being one of the largest in the country, stands out with the complexity of seam composition; significant occurrence depth; and high loads. The abovementioned needs use of the current simulation techniques to assess stress-strain state of the rocks [8]-[10]. The problems become extremely important against huge demands for mining safety and the necessity to minimize risks connected with border rock failure.

Under constant loads, border rocks at large mining depths transit from elastic loading conditions to a stage of non-elastic deformation when the rock mass integrity disturbs, and microdefects arise growing into direct fractures in future. At the expense of the mentioned deformations (dilatancy), increase in rock volume takes place which value is an order of magnitude more than displacements resulting from elastic deformations [11], [12].

Breaking of rock mass around a mine working is one of the most prevailing forms of rock pressure manifestation [13], [14]. The breaking may include large areas of rock mass, and result in the certain share of rock caving in a mine working under gravity. If the breaking area is smaller, then rock pressure is manifested in falling of separate rock fragments. The deformation of mine working periphery observed during mining operations is another manifestation form. Mostly, the both manifestation forms take place simultaneously; in this regard, such dynamic manifestations as outburst of rocks fracturing in a brittle manner; sudden bumps; and blows are the most dangerous [15]-[18]. Rock mass

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breaking in the neighbourhood of underground structure takes place when the certain combination of strain-stress parameters achieves its critical point [19]-[22].

Eventually, stresses of enclosing rock mass around a mine working exceeding their creep limit [23]-[26] decrease owing to plastic rock deformation. Along with the stress drop, rock deformation velocity decelerates; in turn, displacement velocity of the mine working periphery also slows down. In this context, increase in stresses provokes new displacement wave. Following algorithm should be followed for accurate analytical definition of the displacements:

– setting of mining, geological, and engineering conditions for a mine working construction including characteristics of floor and roof rocks;

– analysis of mechanical and strength characteristics of rock layers;

– identification of stresses in the neighbourhood of the mine working, i.e. in front of a stope, in the undermining zone, and in a zone of residual support resistance; calculation of a zone of non-elastic rock deformations as well as displacement velocities based upon the experimental data;

– determination of displacement velocities throughout a mine working operation taking into consideration the identified displacements; and

– definition of the periphery displacements within different areas of the mine working depending upon time (the stope advance).

The purpose of research is the development of a mathematical model to forecast rock displacements around the development mine workings taking into consideration mining, geological, and engineering conditions. Such a model is required to optimize support parameters; select correct geometry of mine workings; and develop procedures for minimization of breaking zones. The paper proposes theoretical approaches and experimental substantiations which formed the basis of KMS-Ш software for numerical simulation. The program helps assess rock displacements and develop recommendations as for border rock mass stability improvement.

The carried-out research relies upon the data obtained while mining coal seams of Saranskaya mine (Karaganda coal basin) which emphasizes its applied nature. The modelling and experimental results may be used to develop scientifically grounded solutions concerning rock displacement decrease under the conditions of large mining enterprises in Kazakhstan as well as in other regions having similar mining and geological environment.

2. Mathematical model

Displacements of border mass rocks are rock pressure manifestations observed from the mined working. Finite values of rock displacements in mine workings U_{sum} consist of elastic U_1 and non-elastic U_2 deformations:

$$U_{sum} = U_1 + U_2 . \tag{1}$$

Elastic deformations are defined through [11] and [12] Formulas:

$$\frac{\partial X_x}{\partial x} + \frac{\partial X_y}{\partial y} + X = 0 \; ; \; \frac{\partial X_y}{\partial x} + \frac{\partial Y_y}{\partial y} + Y = 0 \, , \tag{2}$$

where:

$$X_x = \lambda\theta + 2\mu \frac{\partial u}{\partial x} ;$$

$$Y_y = \lambda\theta + 2\mu \frac{\partial v}{\partial y} ;$$

$$X_y = \mu \left(\frac{\partial v}{\partial x} + \frac{\partial u}{\partial y} \right) ;$$

X, Y, X_x, and Y_y – vectors of active displacements and their projections on coordinate axes;

μ – Poisson ratio;

ν – kinematic viscosity, N/m^2;

u – deformation velocity, m/day;

λ – horizontal stress coefficient, r;

θ – polar point coordinates.

Non-elastic rock deformations within the final condition zone are defined using Formula [23]:

$$U_2 = \frac{S_z}{P_w} \left(K_{def} - 1 \right) , \tag{3}$$

where:

S_z – area of conditional zone of non-elastic deformations, m^2;

P_w – mine working perimeter, m;

K_{def} – rock fragmentation degree within the out-of-limit deformation area.

The area of conditional zone of non-elastic deformations depends upon physicomechanical rock characteristics as well as stress-strain state of rock mass; in the neighbourhood of a mine working, it is defined using following Formulas:

– stresses in the undisturbed rock mass (4);

– extra stresses generated by constructed mine working (5);

– the total of stresses acting in the rock mass (8) [12]:

$$\begin{cases} \sigma_r^{(0)} = \sigma_1 \left(\frac{1+\lambda}{2} + \frac{1-\lambda}{2} \cos(2\theta) \right) \\ \sigma_\theta^{(0)} = \sigma_1 \left(\frac{1+\lambda}{2} + \frac{1-\lambda}{2} \cos(2\theta) \right) ; \\ \tau_{r\theta}^{(0)} = -\sigma_1 \frac{1-\lambda}{2} \cos(2\theta) \end{cases} \tag{4}$$

$$\begin{cases} \sigma_r^{(1)} + \sigma_\theta^{(1)} = 2\left[\Phi(z) + \Phi(z) \right] \\ \sigma_\theta^{(1)} - \sigma_r^{(1)} + 2i\tau_{r\theta}^{(1)} = 2\left(z\Phi'(z) + \Psi(z) \right) \end{cases} , \tag{5}$$

where:

σ_1 – principal vertical stress, MPa;

λ – horizontal stress coefficient, r;

θ – polar point coordinates.

$$z = \omega(\xi) = c_0\xi + \frac{c_1}{\xi} + \frac{c_2}{\xi^2} + \frac{c_3}{\xi^3} + \frac{c_4}{\xi^4} \, , \tag{6}$$

where:

$\omega'(\xi) \neq 0, |\xi| \geq 1$ – section form function [25].

In (5), Φ (z), Ψ (z) functions are:

$$\begin{cases} \Phi(z) = \frac{1}{4}(\sigma_1 + \sigma_2) - \frac{X+iY}{8\pi(1-\nu)} \cdot \frac{1}{\omega(\xi)} + \Phi_* \xi \\ \Psi(z) = \frac{1}{2}(\sigma_2 - \sigma_1) e^{-2ia} + (3-4\nu) \frac{X+iY}{8\pi(1-\nu)} \cdot \frac{1}{\omega(\xi)} + \Psi_* \xi \end{cases} , \tag{7}$$

where:

$X + iY$ – resultant vector of the forces acting in the rock mass (the final value);

$F_x + iF_y$, σ_1, σ_2 – principal stresses, MPa;

α – principal direction angle with Ox axis.

$$\Phi_*(\xi) = \frac{n(\xi)}{m(\xi)}\sigma^p + \frac{\xi^3}{m(\xi)}\sigma^\alpha - \frac{\xi^5}{m(\xi)} p_{n-1}\left(\frac{1}{\xi}\right);$$

$$\Psi_*(\xi) = \frac{\xi^3}{m(\xi)}\left[\begin{array}{l}\left(c_0 - c_1\xi^2 - c_2\xi^3 - c_3\xi^4 - c_4\xi^5\right) \\ \Phi_*(\xi) - p_{n-1}\left(\frac{1}{\xi}\right) + c_0\sigma^p - \\ -\left(c_0\xi + c_1\xi^3 + c_2\xi^4 + c_3\xi^5 + c_4\xi^6\right)\times \\ \times\Phi_*'(\xi) + \left(c_1 + \frac{2c_2}{\xi} + \frac{3c_3}{\xi^2} + \frac{4c_4}{\xi^3}\right)\sigma^\alpha\end{array}\right];$$

$$\sigma^p = P + \frac{1}{2}(\sigma_1 + \sigma_2);$$

$$\sigma^\alpha = \frac{1}{2}(\sigma_2 - \sigma_1)e^{2i\alpha};$$

$$p_{n-1}\left(\frac{1}{\xi}\right) = -c_3\frac{\overline{a_2}}{\xi^2} - c_4\left(\frac{2\overline{a_2}}{\xi^3} + \frac{\overline{a_3}}{\xi^2}\right);$$

$$n(\xi) = c_1\xi^3 + 2c_2\xi^2 + 3c_3\xi + 4c_4;$$

$$m(\xi) = c_0\xi^5 - c_1\xi^3 - 2c_2\xi^2 - 3c_3\xi - 4c_4,$$

where:

ξ – relative deformations, m;

c_0, c_1, c_2, c_3, and c_4 – constant characterizing conditions of a mine working operation, i.e. development pattern of mining, shape and geometry of the mine working, arrangement towards the elements of the seam occurrence, and physicomechanical characteristics of enclosing coal mass;

a_1, a_2, and a_3 – constants under algebraic manipulation;

P – hydrostatic pressure in the mass, MPa;

p_{n-1} – series of changes in displacements;

$m(\xi)$ – coal seam thickness impact on a displacement value.

Composite stresses are determined while summing up initial substance state and stresses generated during mining operations:

$$\begin{aligned}\sigma_r &= \sigma_r^{(0)} + \sigma_r^{(1)} \\ \sigma_\theta &= \sigma_\theta^{(0)} + \sigma_\theta^{(1)}. \\ \tau_{r\theta} &= \tau_{r\theta}^{(0)} + \tau_{r\theta}^{(1)}\end{aligned} \quad (8)$$

While identifying stresses in a random rock mass point, we will obtain matrix of points with their values. A zone of non-elastic deformations (creeping [23]) is defined; within the mentioned area, it is defined through rock fragmentation and displacement at the expense of dilatancy.

The strength condition, developed by L.Ya. Parchevsky and A.N. Shashenko, has been applied to determine the area of rocks being under non-elastic condition [23]:

$$\sigma_e = \frac{(\psi - 1)(\sigma_1 + \sigma_3) + \sqrt{\begin{array}{l}(\sigma_1 + \sigma_3)^2(\psi - 1)^2 + \\ +4\psi(\sigma_1 - \sigma_3)^2\end{array}}}{2\psi} \le R_c, \quad (9)$$

where:

$\sigma_1 > \sigma_3$ – principal stresses, MPa;

$\psi = R_p / R_c$ – rock tensile strength-uniaxial compression strength ratio, MPa.

Rock mass integrity is disturbed within the non-elastic deformation area; microdefects arise growing into macrocracks. Expansion of the deformations results in the increased rock volume, i.e. dilatancy which level is an order of magnitude more than displacements caused by elastic deformations [23], [24]. The abovementioned is the key reason of rock displacements in the border part of mine workings within the area of acting rock pressure [26].

According to research [25], fragmentation degree is 1.1-1.18 within the unsupported mine working periphery. Timbering decreases significantly the displacements involved by decrease in rock fragmentation degree:

$$K_{def} = K \cdot \frac{2}{3}\left(\frac{S_w}{S_z}\right), \quad (10)$$

where:

S_w – rock section area of a mine working, m²;

S_z – supported mine-working area in the clear, m²;

K – coefficient taking into consideration the support availability, $K = 0.18 - 0.08 \cdot (P/100)$;

P – bearing capacity of the support, t-f/m².

Fragmentation degree decreases along with the distancing from a mine working periphery owing to the rock pressure decline. If the supposed non-elastic deformation zone is less than 0.5 m, then a fragmentation degree drops by half [23].

A mine working periphery displacement is approximated by means of a logarithmic function [27]:

$$U_t = b_0 \ln(t + 1); \quad (11)$$

$$b_0 = \frac{U_t}{\ln(t + 1)}, \quad (12)$$

where:

b_0 – coefficient characterizing displacement intensity, mm/day defined relying upon the results of field studies performed during the first month;

t – time, days.

To define b_0, observation stations are installed on the first day for observation of deformation onset; however, rather often such a task turns out to be not always possible from engineering viewpoint. Hence, to exclude inaccuracy, no less than three observation series should be implemented:

$$U_t + U_0 = b_0 \ln(t_0 + 1 + t), \quad (13)$$

where:

U_0 – displacements before observation stations are installed, m;

t_0 – period from the moment of the mine working cavity formation, days.

The combined solution of equations resulting from results of the three measurements factors into the transcendental function as for t_0:

$$\frac{U_2 - U_1}{U_3 - U_2} = A;$$

$$\left(\frac{t_3 + t_0 + 1}{t_2 + t_0 + 1}\right)^A = \frac{t_2 + t_0 + 1}{t_1 + t_0 + 1};$$

if $A = 1$ then:

$$t_0 = \frac{t_2^2 - t_1 t_3}{t_1 + t_3 - 2t_2} - 1. \tag{14}$$

Coefficient b_0 value depends upon the ultimate dimensions of a limit state zone; it may be identified based upon generalization of the available experimental observation materials concerning displacements of mine working peripheries. In this regard, to compare with permanent workings, analytical definition of rock mass displacements in the neighbourhood of development workings is complicated due to following factors:

– insufficient accuracy of stress concentration determination within a bearing pressure zone;

– time limitation of a mine working being in the maximum stress zone within the mine working neighbourhood preventing from the use of the finite maximum displacements.

Displacement velocities of the peripheries of a mine working depend upon stress degree of rock mass; its neighbourhood; and physicomechanical characteristics of the rock mass. Calculation of the development mine working periphery displacements should involve their velocities resulting from changes in mining conditions.

3. Methodology

When a mine working is driven through a coal seam with a stable roof and floor, displacements of the working walls are to be considerably superior to the floor and roof displacements, which depend only upon elastic deformations of the floor and roof as well as upon general shift without fall. Mine working wall displacements include following components of changes in rock volume within the mine working walls:

– U_1 component at the expense of change in the floor and roof convergence level within a zone where acting stresses are less than natural ones defined through the Formula:

$$\sigma_y = \gamma H \cos^2 \alpha + \lambda \gamma H \sin^2 \alpha, \tag{15}$$

where:

α – formation dip, degrees;

λ – horizontal stress coefficient;

– U_2 component stipulated by rock dilatation within non-elastic deformation zone.

In the context of a mining operation schedule (on one side, a mine working is with rock mass; on the other side, it borders on the mined-out area), displacements from floor and roof are defined on the line with the caved rocks. Displacements of the periphery of a mine working from rock mass are identified using the Formula:

$$U = \frac{V_{znd} \cdot K_{def}}{m}, \tag{16}$$

where:

V_{znd} – volume of non-elastic deformation zone per a meter of the mine working, m^2;

m – seam thickness, m.

The considered approach is correct for the case when only a coal seam through which a mine working is constructed is under deformation. Most commonly, both coal seam and enclosing rocks around development mine workings experience non-elastic deformations. Under the conditions, displacements resulting from the main roof and floor convergence (if only they do not transit to out-of-limit deformation stage) are considerably less than displacements resulting from dilatation of rocks in the area of non-elastic deformations.

Displacements of the mine working peripheries are calculated using the maximum stresses generated within area of their location during a stope advance. In this regard, the hardest stage is to take into consideration the period of the maximum stresses connected with advance velocity of the stope, and distance from the seam being mined.

The most general principle to calculate displacements of development working periphery is definition of the two previously described components of changes in rock volume within an area of mine working influence: ΔV_1 component stipulated by elastic rock extension in a zone of the decreased average stress value $\sigma = \frac{1}{3}(\sigma_1 + \sigma_2 + \sigma_3)$, and ΔV_2 component stipulated by rock dilatation in a zone of non-elastic deformations. Average value of peripheral displacements is a partial of the rock volume increment division in a zone of a mine working influence by its perimeter L [25]-[27]:

$$U = \frac{\Delta V_1 - \Delta V_2}{L}. \tag{17}$$

The finite displacements for different levels of stress-strain state of rock mass are defined using the Formula [23]:

$$U_0 = U_1 + U_2 = \frac{R_L^2}{R_0} \left[\begin{array}{l} \alpha \dfrac{1+\nu}{E} (k\gamma H - \sigma) + \\ + \dfrac{K_{def}}{2} - \dfrac{R_0^2}{2R_L^2} K_{def} \end{array} \right]. \tag{18}$$

Rock displacement within the periphery share of mine workings may depend upon following factors: rock dilatancy and increase in its volume during breakage; stratification; and bend of the formed rock consoles. Rock convergence decrease or exclusion in a mine working roof for the last two reasons may be achieved through correct selection of means and timbering parameters. It is more complex to identify and decrease rock displacements in the neighbourhood of a mine working connected with rock dilatancy.

Rock dilatancy depends upon the two reasons: formation, accumulation, and consolidation of microcracks; and transition of adjoining surfaces of integrity macrocracks relative to each other. Moreover, the latter mechanism of rock dilatancy prevails. In the context of practical calculations to define displacements of a border rock mass in the limit area, fragmentation degree may be equal to 1.001-1.005 if rock transition into out-of-limit condition is excluded; otherwise, K_p = 1.04-1.1 [28], [29].

Strain is considered as a two-stage process. Stage one (preparatory) is characterized by joint and dislocational phenomena identifying translational (preliminary) deformation which varies material structure. It defines the environment when microcracks originate, and their clusterization up to critical size cracks. The area limited by a rectangle in front of a crack tip (Fig. 1), define the possibility of viscous mode of its propagation. Stage two (brittle) penetration of a crack takes place along directions 2 and 3 of maximal shear stresses. In this case, concurrent deformations depend mainly upon instability of dislocations.

As a result, crack growth is the most probable because dislocations from sliding surfaces of neighbouring grains inflow into it; however, other mechanisms of a crack extension cannot be excluded.

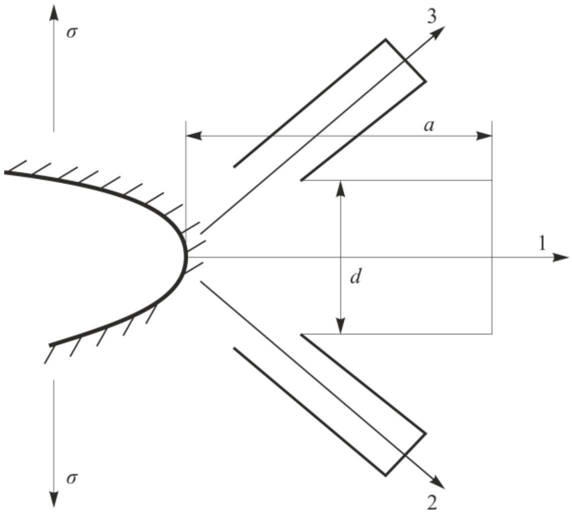

Figure 1.** **Two modes of crack propagation: 1 – viscous; 2, 3 – brittle; a – depth; d – crack geometry

In general, fracturing process at the stage of accelerated crack growth (preliminary fracturing stage) can be represented as follows (Fig. 2). Dislocation breakaway and movement at an initial stage factor into structural transformation of the material at mesoscopic and structural levels in terms of defects being available in it before the loading. Dislocations converge to the chambers (Fig. 2a); their blockage due to obstacles results in elastic energy accumulation within the zone. The abovementioned favours formation of critical dilatons within the top of defects being available in the body (Fig. 2b).

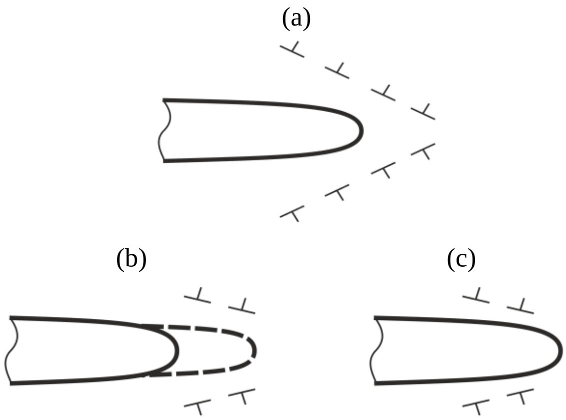

Figure 2. Scheme of microcrack growth at a preliminary fracturing stage: (a) accumulation of dislocations near the defect top; (b) critical dilaton formation; (c) crack extension after the dilaton disintegration

At the moment, the crack periphery is defined like a contour of a structural defect continued in the undisturbed part of the material in the form of the critical dilaton. Under the effect of thermal fluctuations, the critical dilatons decompose explosively forming initial cracks; merging with previously existing defects, they stretch them out (Fig. 2b). While exploding, dilaton provokes release of dislocations from impurities; in such a way, stress relaxation takes place.

Consolidation of border rocks (i.e. grouting, chemical strengthening etc.) gives rise to increase in their strength, and factors into decrease or elimination of breaking zones near a mine working. If the prepared rock mass strength is higher than stresses acting in border mass then no disintegration takes place, and dilatancy is excluded.

The described above mathematical tool to forecast the expected displacements has become the basis for software program KMS-Ш (displacement simulation complex for mines) (Fig. 3).

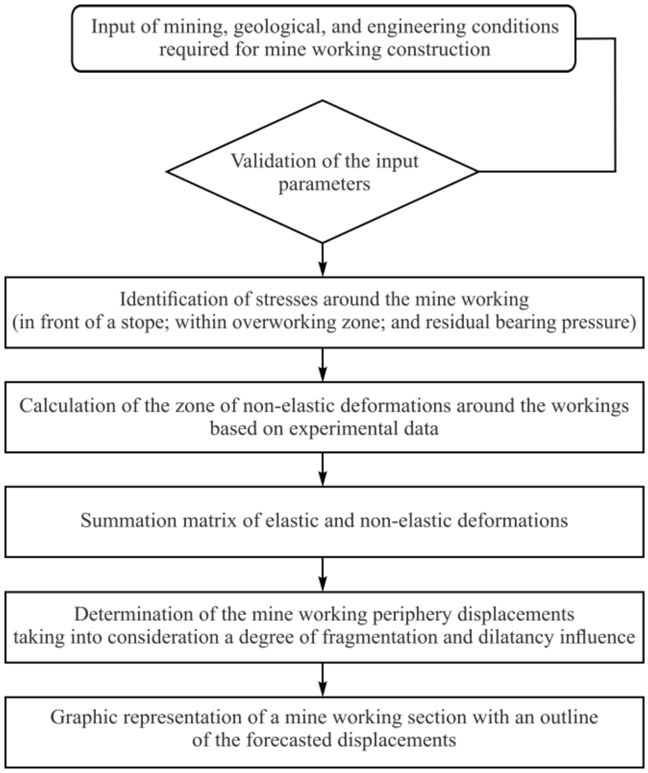

Figure 3. Functional diagram of KMS-Ш software program

The program performance starts from the input of coefficient parameters and constants applied for computation, and initialization of interface part of the software. Next step is interactive data input with control of allowable values and data correctness.

Following criteria are introduced as the initial data:

– mining depth, m;

– volume rock weight, kN/m^3;

– geological section of the working being constructed including thickness of layers, and physicomechanical characteristics of corresponding layer (i.e. compression strength, tensile strength, adhesion factor etc.);

– inclination of rock layers, degrees;

– sectional shape of a mine working and its geometry, m.

Simulation of strain state of rocks within the heterogeneous mass using rheological model and identifying support parameters in stopes, development mine workings, permanent mine openings etc. is only possible if geomechanical conditions of mine working driving and timbering are taken into consideration as well as influence by mining, geological, engineering, and technological environment [30], [31]. Mining schedule being depth; vertical and horizontal loads; and Poisson ratio is also important [32], [33]. Such mining and geological conditions of rock mass as a layer characteristic; ordinate of the layer roof; adhesion factor of corresponding rock layer; internal friction angle; uniaxial compression strength; tensile strength; and inclination of layers to the horizon were taken from scientific sources [34], [35]. Mining depth is 800 m; vertical load is 10 kN/m^2; horizontal load is 10 kN/m^2; section of a mine working is rectangular; top (bottom) width is 6.0 m; and height of the mine working is 4.0 m.

4. Results and discussion

Numerical simulation of the expected displacement was considered in terms of the eastern ventilation slope 50 k_{10} of Saranskaya mine ArcelorMittal Temirtau UD JSC. The mine working is at 428-554-m depth; angle is 10°; extension is 630 m. The total seam thickness in the construction place is 4.65 m. k_{10} seam is of a complex structure. It consists of 9 coal benches with 0.05-1.17-m thickness; they are separated by interlayers of carbonaceous argillite and argillite with 0.01-0.04-m thickness. k_{10} seam belongs to a class of seams prone to sudden coal and gas outbursts from 300-m depth. The seam is gas-and dust-hazardous; it is prone to self-ignition.

Sandstone (m = 23.7-29.56 m; f = 60 MPa) occurs in the main roof. Immediate roof is represented by argillite which thickness is 1.24-2.09 m (f = 25 MPa). False roof consists of carbonaceous argillite and argillite which thickness is 0.45 m (f = 15 MPa).

Argillite with 5.25-6.35-m thickness (f = 20-25 MPa) occurs in the seam floor; the mineral is unstable and prone to heaving. The expected water inflow will be up to 5 m^3/hour.

Bolting with 0.8-m interval is applied to support a mine working. The number of rockbolts per 1 m of the mine working, i.e. 12 are in roof and 6 are in walls.

Following expected displacements of the border rock mass have been obtained with the use of KMS-Ш software:

– 200-300 mm in the roof;
– 500-650 mm in the floor;
– 150-200 mm in the walls.

To compare the simulation results and actual displacements, observation stations were installed on survey marks ПК10, 18, 21, 32, 52, and 59 along with a stope advance.

Analysis of displacements from a right side (Fig. 4) shows that intensive deformation stage takes place during the first month from the moment of observational survey point installation. For the first month, displacement value was 7 mm. Following months did not demonstrate any displacements. Within a right side of the mine working, intensive displacements of border mass rocks were observed during the first month. The maximum displacement values were 3 mm. No displacements were observed for following months.

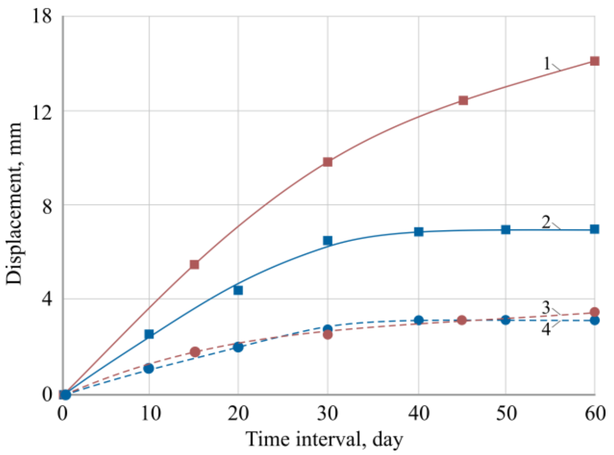

Figure 4. Displacements around the mine working: 1 – right side; 2 – left side; 3 – floor rocks; 4 – roof rocks

Displacement dynamics of the roof rocks of a ventilation slope (Fig. 4) shows that all the displacements took place during the first month. Following months demonstrated decrease in the displacement intensity down to a complete stop. The maximum displacement values were not more than 3 mm; hence, the selected support parameters are efficient. Soil heaving analysis showed that for the first month, the maximum displacement values did not exceed 10 mm. Following month demonstrated decrease in the intensity; the maximum displacements were not more than 4 mm.

The obtained displacement values can be explained by the availability of unstable and prone to heaving argillite in the immediate floor; the mineral thickness is 5.25-6.35 m (f = 20-25 MPa). On the whole, analysis of the displacements across all survey marks more or less demonstrate their general nature; in addition, they are similar to the considered pattern.

Displacement calculation of a border rock mass where rock bolting was not applied, showed coincidence of the obtained results with the observed ones; standard error was ±7.2%. Consideration of the calculation results in comparison with actual ones for a site where rock bolting has been installed, helps understand that values of the forecasted displacements are an order of magnitude more than actual amounts. Hence, rock bolting availability reduces the expected displacement by several times.

If rook bolting is installed then roof displacement intensity is 8 mm, i.e. b_0 = 2.33 mm/day. If 300 mm are assumed as the maximum expected monthly displacement then the displacement intensity should be 87.36 mm/day. Consequently, rock bolting operation decreases displacement intensity by almost 40 times.

Taking into consideration the number of rockbolts as well as their bearing capacity, it is possible to derive dependence between displacement values of bolted and unbolted mine working:

$$U_{\Phi} = \Delta b_0 U_{pc}, \tag{19}$$

where:

$\Delta b_0 = b_0^{\Phi} / b_0^{PAC}$ – ratio between actual and calculated displacement intensities or decrease in displacement intensity owing to rock bolting availability.

In turn, decrease in displacement intensity can be represented criterially as follows:

$$\Delta b_0 = f\left(P_{bolt}, N_{bolt}\right), \tag{20}$$

where:

P_{bolt} – bearing capacity of rockbolts, t;
N_{bolt} – number of rockbolts, pieces.

To identify the dependence between displacement intensity and bearing capacity of a support, further research should be carried out where bearing capacity of a support will vary, and displacements will be defined for each case.

Comparison of analytical values with data obtained numerically, it is possible to draw a conclusion on their repeatability:

– error of displacement calculation in a roof between full-scale measurements and analytical ones is about 2%;
– error of displacement calculation in walls between experimental numerical and analytical data is about 6 %;
– displacement in floor obtained analytically are 2.8 more than displacement obtained under full-scale conditions.

The results of numerical simulation concerning displacement and failure of mine working periphery are represented both graphically and textually (Fig. 5).

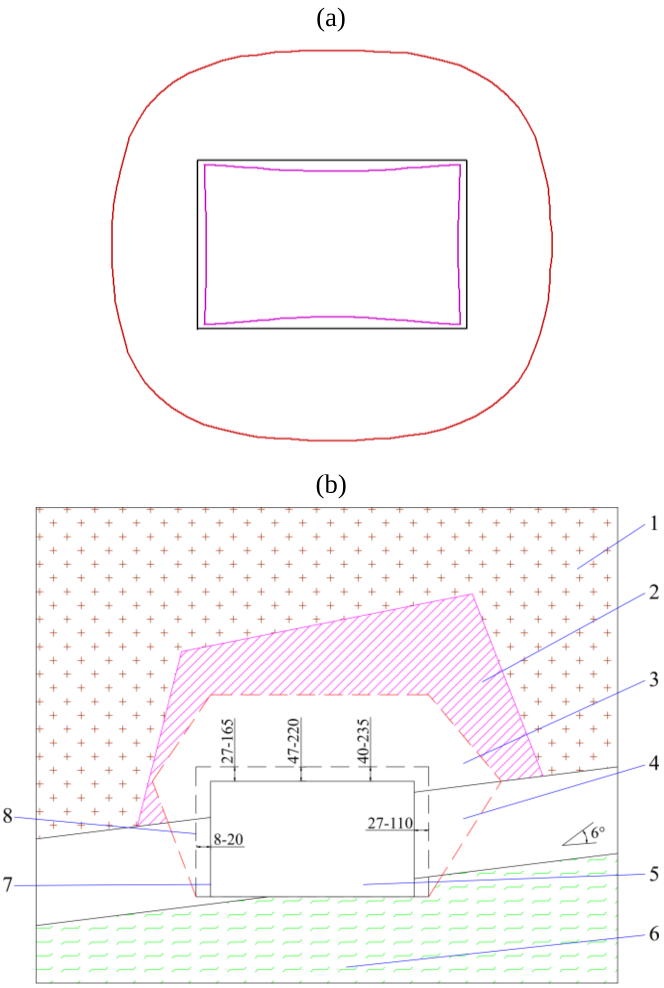

Figure 5. Sample report on the calculation results: (a) scheme of failure and displacement peripheries; (b) explanations of the calculations; 1 – roof rocks; 2 – elastic deformation zone; 3, 4 – non-elastic deformation zones in the mine working roof and walls; 5 – mine working cavity; 6 – floor rocks; 7, 8 – operational and initial peripheries of the mine working

The obtained analytical displacements are: 0.272 m for roof; 0.276 m for floor; and 0.236 m for walls.

To identify the influence of various factors on the progress of rock layer displacements of a border rock mass around development mine working, it is possible to apply a numerical experiment with the help of software for displacement modelling.

Comparative analytical and experimental assessment of the rock mass stress-strain parameters near mine workings involved an analysis of displacement progress of a border formation depending upon the mining and geological conditions.

The key factor influencing displacement progress is rock pressure, which depends on mining depth. The calculations were performed for a single mine working driven in homogeneous rock mass where uniaxial rock compression strength is 24 MPa being average for Karaganda coal basin. The mine working is of arch cross-section; its width is 5.57; height is 3.55 m. Its depth varied from 400 down to 800 m (the depth interval is for Karaganda coal basin mines). Figure 6, a shows analytical dependence of the peripheral formation displacements upon the mining depth. Horizontal stress coefficient value has been assumed as 1.0.

Linear dependence has been defined for the considered depth interval between the expected displacements of peripheral formation from the unbolted mine working roof and mining depth (Fig. 6, a) expressed through the Formula:

$$U_{exp} = 0.3093H - 21.322 \text{ if } r = 0.97. \quad (21)$$

Rock strength, which varied within the values being typical for Karaganda coal basin (i.e. 10-40 MPa), was an extremely important factor influencing displacements of peripheral formation of a mine working.

Exponential dependence of the peripheral rock mass upon strength has been derived:

$$U_{exp} = 1294.4e^{-0.1003\sigma_c} \text{ if } r = 0.97. \quad (22)$$

Figure 6 demonstrates the obtained dependencies of rock displacements of the peripheral rock mass from a roof.

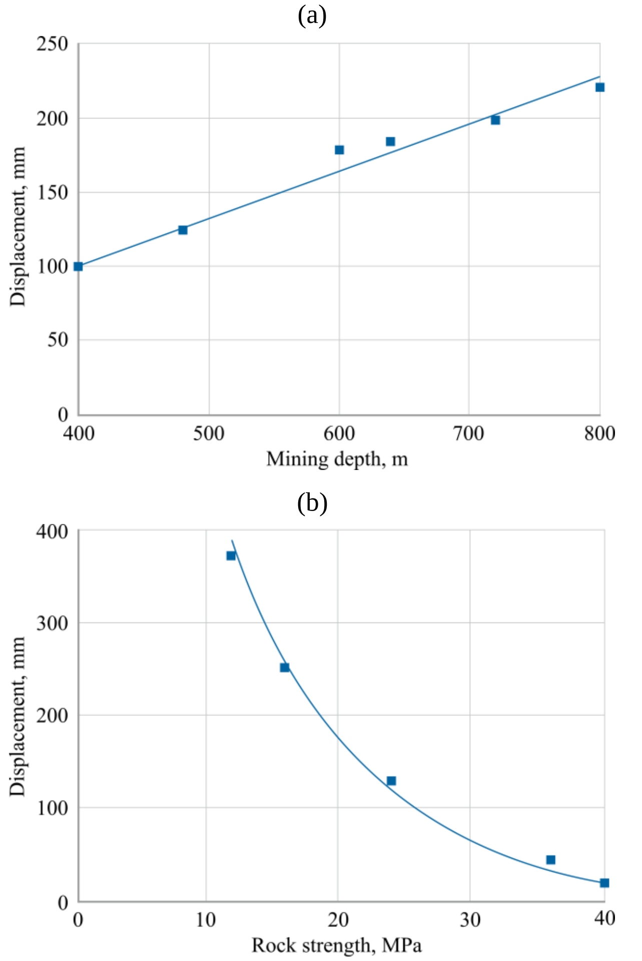

Figure 6. Dependence of the peripheral formation rock displacement from a roof: (a) upon mining depth; (b) upon uniaxial compression strength

The simulation results also emphasize the importance of mining depth as the key factor influencing displacements. Linear dependence between the mining depth and peripheral formation displacements (Fig. 6a) supports the idea that displacements increase along with depth deepening which is stipulated by rock pressure rise. In practice, it needs a specific approach to a support design for deep mine workings where rock pressure impact is particularly important. Rock strength characteristics have a significant effect on displacements too (Fig. 6b).

The derived dependence shows that the minimum displacements are achieved if rock strength is ultimate stressing the necessity of rock mass strengthening.

Relying upon the research, it is recommended to optimize support while increasing the number of rockbolts in the floor; using support differing in high bearing capacity for the stressed zones; strengthening peripheral formation by means of cementation and chemical stabilization methods to prevent heaving; and optimizing mine working geometry applying arch cross-section and decreasing dimensions under high pressure conditions. In addition, it is recommended to implement systems of the automated monitoring of displacements, and forecasting of the maximum deformation zones with following use of local strengthening.

5. Conclusions

The research has helped conclude that deformation acceleration results in the increased rock strength as well as accumulated elastic energy. The abovementioned is favourable from the viewpoint of reducing the probability of failure zone formation in the neighbourhood of a mine working; however, it may factor into dilatation if analytical stresses surpass the rock strength. Even if failure zones originate (which depends upon rock characteristics), it is required to decelerate deformation to reduce dilatancy. It can be achieved through the support installation right after outcrop.

Stress increase is defined by means of a mine working depth: along with distancing from its periphery into the depth, dilatancy probability drops owing to the reduced displacement of adjoining crack surfaces. Moreover, it is possible to decrease dilatancy while selecting such cross-section shape of mine workings, which will exclude concentration of stresses and minimize their values.

Structural characteristics influence differently: strength degradation increases failure zones and strengthening decreases the accumulated elastic energy as well as dilatancy probability. Rock strength decline factors into the increase in failure zones. In turn, plastic property improvement drops elastic energy accumulation reducing the potential for opposite crack surfaces to be displaced.

Strengthening of the peripheral formation rocks, using cementation, chemical stabilization or other methods, increases rock hardness resulting in minimization of exclusion of disturbance zones near mine working. If strength of the treated rock mass surpasses stresses acting in the formation then failure is excluded, and dilatancy does not manifest itself.

Author contributions

Conceptualization: AZ, NM, SS; Data curation: TD; Formal analysis: DM, ZA; Funding acquisition: SS; Investigation: TD; Methodology: TD, AG; Project administration: AZ, NM, ZA; Resources: DM; Software: TD, SS; Supervision: AG; Validation: AZ, NM, SS; Visualization: TD, ZA; Writing – original draft: DM; Writing – review & editing: TD, AG. All authors have read and agreed to the published version of the manuscript.

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Conflicts of interests

The authors declare no conflict of interest.

Data availability statement

The original contributions presented in the study are included in the article, further inquiries can be directed to the corresponding author.

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Моделювання зміщень вміщуючих порід навколо підготовчих гірничих виробок

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Мета. Розробка та обґрунтування математичної моделі для прогнозування зміщень гірських порід навколо підготовчих гірничих виробок та оптимізація способів їх зміцнення.

Методика. Використано комплексний підхід, що включає чисельне моделювання, теоретичний аналіз та експериментальні дослідження. Основну увагу приділено визначенню зсувів гірських порід на підставі напружено-деформованого стану масиву. Для розрахунків використано розроблену програму "KMS-Ш", що дозволяє моделювати зміщення та аналізувати отримані дані.

Результати. Запропоновано рекомендації щодо зниження коефіцієнта розпушення (дилатансії) гірських порід, що досягається за рахунок коригування параметрів кріплення, геометрії виробки та управління швидкістю деформування порід. Показано, що використання анкерного кріплення значно знижує інтенсивність зміщень. Виявлено, що зменшення межі міцності порід призводить до збільшення зон руйнування, тоді як підвищення пластичних властивостей зменшує накопичення пружної енергії, що знижує ймовірність переміщення протилежних поверхонь тріщин.

Наукова новизна. Розроблено алгоритм прогнозування зсувів контурів гірничих виробок з урахуванням гірничо-геологічних та гірничо-технічних факторів. Представлено математичну модель для визначення пружних та непружних деформацій у приконтурній зоні. Встановлено залежності між зміщеннями приконтурного масиву та глибиною виробки, а також міцністю порід.

Практична значимість. Визначення оптимальних параметрів зміцнення порід дозволяє мінімізувати зони руйнування, підвищити стійкість гірничих виробок та знизити ймовірність небезпечних геомеханічних явищ Застосування запропонованої моделі забезпечує підвищення ефективності гірничих робіт за рахунок більш точного прогнозування зміщень.

Ключові слова: *підземна розробка, гірничі виробки, кріплення, геомеханічні процеси, анкерне кріплення, напружено-деформований стан, гірський тиск, зміщення порід*

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