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EFFECT OF INCREASING MINING RATE ON LONGWALL COAL MINING – WESTERN DONBASS CASE STUDY

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Abstract: This paper presents the problems associated with the rapid change of the rock stress-strain state in terms of increasing the rate of coal mining. Parameters of the roof collapse are determined depending on the rate of a longwall advancing under conditions of poor rocks. Statistical data are processed to obtain a general trend concerning the mining rate impact on the roof collapse. The statistical strength theory is applied to explain the increase in mined-out space and the size of hanging roof behind a coal face. Numerical simulation is carried out to determine a critical size of mined-out space that provokes a roof collapse. The area of yielded rocks is outlined using the criterion developed taking into account the rate of longwall advancing. A general regularity is obtained to determine the roof collapse parameters. The developed technics gives a possibility to predict the moment of general roof collapse at the initial stage of longwalling to prevent the negative effect of the rapid stress redistribution provoking joints propagation and intensive gas release. The estimation of the rock stress-strain state considering the rate of mining operations can be useful for tasks related to a new technology implementation. The statistical strength theory and failure criterion applied together provides adequate planning of mining activities and the assessment of natural hazards.

Key words: *rock, stress-strain state, mining rate, failure criterion, roof collapse*

1. INTRODUCTION

Western Donbass coal deposit is located in Dnipropetrovsk region of Ukraine and covers an area of 12000 km^2. As the thickness of the coal seams varies from 0.1 to 1.5 m, they can be classified as "thin seams" (Nagorniy, Nagorniy and Prihodchenko, 2005). A distance between the seams varies from 4.6 m up to 40–60 m. The depth of coal seam below ground surface varies from 50 m up to 900 m.

The coal seam roof and floor are mudstones and siltstones (75–80%) and sandstones (20%), respectively. Sandstones have thickness of 5 to 50 m and are aquifers. A distinctive feature of Western Donbass geological structure is that coal is strong and ductile with compressive strength of 30–35 MPa, while both floor and roof are composed of weak jointed rocks with compressive strength of 10…25 MPa. Mudstones and siltstones of Western Donbass are classified as poor and unstable rocks according to the classification adopted in Ukraine (Standard, 2007). When exposed to water they loose 50–80% of their strength. Floor heaving often occurs during excavation (Khalymendyk, 2011).

Numerous faults are located in the western part of the deposit. Zones of heavily jointed rocks surround the faults. The coal formation is characterized by monoclinal bedding with an angle of inclination between 1 to 5°.

Coal mining in Western Donbass is often accompanied by falling wedges formed in the roof by intersecting structural features, such as bedding planes and joints, which separate the rock mass into discrete but interlocked pieces. The height of opening created varies from 1.0 ... 1.5 m up to 8 ... 10 m. Mining the rocks where faults are located can be accompanied with methane emissions. The natural methane capacity of a coal seam is about 8 ... 14 m^3/t. Coal seams are not subject to sudden outbursts and rock bursts, but the dust can cause dangerous situations.

In spite of rather complicated geological conditions a coal production is growing in the region due to the introduction of new techniques and technologies. The implementation of effective coal mining technology (in particular, plow longwalling) facilitates high rates of operations (Pivnyak et al., 2013). Increasing the rate of excavation alters the mechanical processes in rocks. Therefore, the prediction of possible changes in rock state is essential to ensure a safe operation of coal mines.

While a longwall retreats a mined-out space (a goaf) increases behind a coalface and primary roof collapse occurs, causing an abrupt redistribution of stresses in the rock mass. This initiates the opening up and propagation of natural joints and creates new planes of weakness. The support of roadways and longwall shields gets loaded additionally and can be destroyed (Prusek and Lubosik, 2006). Sometimes a dramatic floor heaving occurs, resulting in disruption of transport routes. Roof caving provokes intensive gas release and other phenomena related to crack opening and propagation. This negative effect depends on the size of mined-out space. So, forecasting the roof collapse in a goaf behind the longwall face is a great challenge.

The research goal is to determine the effect of mining rate on the roof collapse and other processes connected with the stress redistribution in the rock mass.

2. STATISTICAL DATA PROCESSING

Data related to a roof caving have been gathered at 10 coal mines of DTEK Pavlogradugol Company. The critical size of mined-out space provoking the roof collapse has been studied depending on the rate of a coalface advance. The sites having the similar length of a mining front and longwall up to 300 m have been observed. The analysis showed a general tendency of increasing the critical mined-out space while increasing the rate of mining. A sufficient degree of correlation is shown by data related to the sites with similar geological conditions. These conditions can be indicated by a "mining" factor (Shashenko, Solodyankin and Martovitskiy, 2012)

$$\Theta = \frac{R_c}{\gamma H}, \tag{1}$$

where R_c is an average compressive strength, MPa, γ is a weight density, MN\m^3, H is a depth of mining, m. The average compressive strength is calculated involving the roof thickness of at least 20 m (Standard, 2007). Data collected in the sites indicated by the factor Θ in the range between 0.7 and 1.1 are represented in Fig. 1. In this case the trend line can be approximated by a function increasing monotonically. In particular, a logarithmic function gives the biggest coefficient of determination (R^2 = 0.87) for the sample obtained.

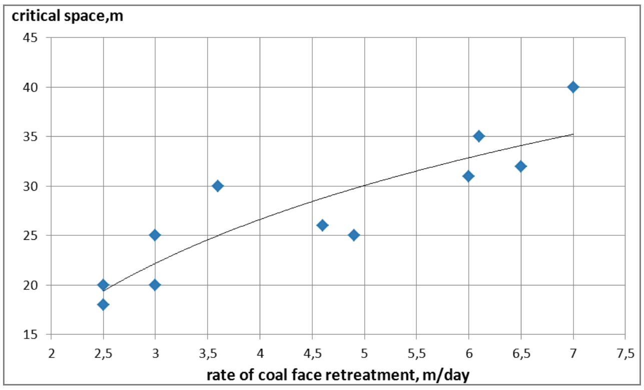

Fig. 1. Critical mined-out space provoking a roof caving depending on a mining rate

It should be noted that because of a small scope of data, the statistics are used only to show a general tendency and define further research direction. The increase in critical size of mined-out space provoking the roof collapse can be explained by rock strengthening under effect of mining rate. So, a theoretical concept concerning the effect of loading rate on the rock strength should be discussed.

It can be assumed hypothetically that mechanical processes in a structural element of the rock mass at rapid excavating are similar to the processes in a rock specimen at rapid loading in a test machine. Therefore, many research teams have studied the failure of rock samples at various loading rates (Mansurov, 1982; Chong et al., 1987; Olsson, 1991). A representative scope of compression testing the samples of sandstone, siltstone, mudstone and coal has been carried out at Geotechnical Institute of Ukrainian Academy of Sciences (Skipochka and Usachenko, 2006). The research resulted in increasing the compressive strength depending upon the loading rate within the speed range 10^{-3} ... 10^{2} MPa/sec. Many of the authors mentioned above approximated the experimental rate-strength relationship using a logarithmic function.

The phenomena of rock strengthening can be explained in terms of statistical theory of strength.

3. ROCK STRENGTHENING IN TERMS OF STATISTICAL THEORY

Destruction of a rock specimen can be represented as a consecutive failure of links in the material (Ivanov, Sdvyzhkova and Rubets, 2007). The strength of any link (Fig. 2) can be considered as a random variable distributed in accordance with one of such asymptotic probability laws as normal, log-normal, Weibull, etc. (Shashenko and Sdvyzhkova, 2010). Destruction of the weakest link under action of the external load *P* initiates a redistribution of link strength. In the case of total failure of *n* links including the strongest one, the specimen resistance runs out during the time t_f. Probability of this event is described by a function $F(r)$ of a random variable *R* which is a maximum value among all possible values of the link strength. Maximum value *R* of any sample is a random value distributed according to Gumbel' law with a probability function (Hahn and Shapiro, 1994)

$$F(r) = \exp\left(-\exp\left(-\frac{r-\mu}{\delta}\right)\right) \qquad (2)$$

Here μ, δ are parameters of the shape and scale. If the statistical sample is representative (a sample volume *m* equals 30–50 elements) these parameters are determined with the use of the sample mean $\overline{R}$ and standard deviation *D* according to Gumbel' distribution (Hahn and Shapiro, 1994)

$$\mu = \overline{R} - 0.58\delta, \qquad \delta = 1.29D. \qquad (3)$$

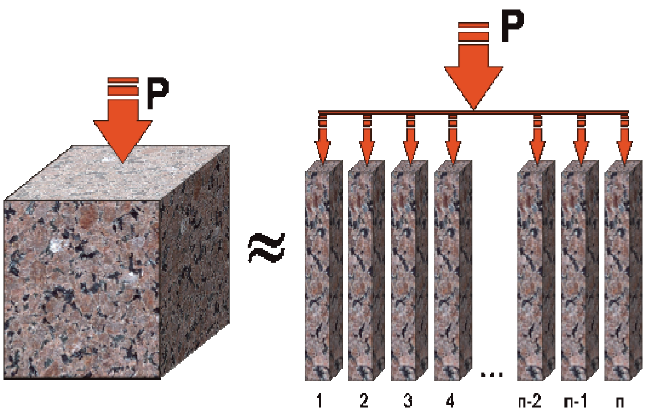

Fig. 2. A rock specimen under the load *P* shown as a system of "links" with different strengths

Let the external load acts over time $t_{fn} = {}^{t_f}\!/_{n}$. This means that loading rate increased by *n* times from a value v_0 up to $v_n = nv_0$. Then the probability of failure of all *n* links can be defined as a probability of intersection of *n* independent events. The probability function takes the form

$$F^n(r) = \left[\exp\left(-\exp\left(-\frac{r-\mu}{\delta}\right)\right)\right]^n$$

or

$$F^n(r) = \exp\left(-n\exp\left(-\frac{r-\mu}{\delta}\right)\right).$$

After transformation one can obtain

$$F^n(r) = \exp\left(-\exp(\ln n)\exp\left(-\frac{r-\mu}{\delta}\right)\right)$$
$$= \exp\left(-\exp\left(\ln n - \frac{r-\mu}{\delta}\right)\right).$$

Finally, the probability function at loading rate v_n looks like

$$F^n(r) = \exp\left(-\exp\left(-\frac{r-(\mu+\delta\ln n)}{\delta}\right)\right) \qquad (4)$$

where

$$n = {}^{v_n}\!/_{v_0}. \qquad (5)$$

The function $F^n(r)$ is of the same type as $F(r)$-function and differs only in parameters which are given by the formula

$$\mu_n = \mu + \delta\cdot\ln(n), \quad \delta_n = \delta. \qquad (6)$$

Considering that according to (3) $\mu_n = R_n - 0.58\delta_n$ and $\delta_n = \delta$, we obtain a formula to calculate the mean of strength at increasing the loading rate by *n* times

$$R_n = \overline{R} + 1.29\cdot D\cdot\ln\frac{v_n}{v_0}. \qquad (7)$$

Thus, the probability-statistical solution also tends to a logarithmic association of strength with a loading rate, and equation parameters can be obtained by standard processing statistical data. Equation (7) can be led to an aspect

$$R_n = \overline{R}\left(1 + 1.29\eta\ln\frac{v_n}{v_0}\right), \qquad (8)$$

where $\eta = D/\overline{R}$ is a relative variation of strength values. The mean value $\overline{R}$ and relative variation η characterize a statistical sample obtained from the standard testing at the loading rate v_0. These parameters can be obtained by standard processing of the statistical data. Equation (8) gives the average strength in the case of increasing loading rate by n times up to the value v_n. Apparently from the graph (Fig. 3) illustrating equation (8) increasing the load rate results in strengthening (more than 2 times) those rocks for which a relative variation is considerable ($\eta > 0.3$), i.e., for the structurally inhomogeneous rocks. Putting $v_n = v_0$ in equation (8) (this means that $n = 1$) we obtain the strength value without rate effect which coincides with mean $\overline{R}$ of the standard testing.

The rate of rock excavation and, hence, the velocity of stress redistribution in the rock mass are comparable to the velocity of a specimen loading in a test machine. An infinitesimal structural element of the rock mass is compressed by the rock pressure like a rock specimen in a test cell. The resistance of the rock structural element also depends upon loading rate like the resistance of the rock specimen. Hence, applying equation (8) gives the possibility to take into account the real condition of mining and correct the strength obtained in a laboratory.

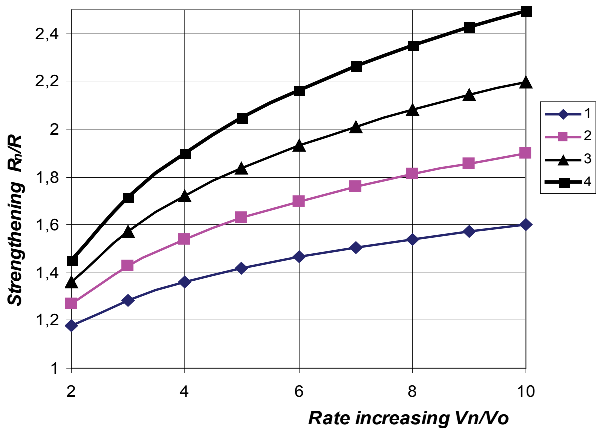

Fig. 3. Rock strength increase depending on affixed load rate according to (8): 1 – $\eta = 0.2$; 2 – $\eta = 0.3$; 3 – $\eta = 0.4$; 4 – $\eta = 0.5$

To estimate the real failure in rock mass equation (8) should be used together with one of the failure criteria.

4. SIMULATION OF THE ROCK STRESS STATE CONSIDERING THE LOADING RATE

The area of broken rocks around any opening can be defined as a set of points at which the condition takes place

$$\sigma_e \geq \sigma_{\lim}, \tag{9}$$

where $\sigma_{\lim}$ is a limit value of the rock strength and σ_e is an equivalent stress defined according to available strength theory. In particular, a confine stress state can be reduced to an equivalent uniaxial compressive state according to Balandin's strength theory (Shashenko, Sdvyzkova and Gapeiev, 2008)

$$\sigma_e = \frac{(\psi-1)(\sigma_1+\sigma_2+\sigma_3)}{2\psi} + \frac{\sqrt{(\psi-1)^2(\sigma_1+\sigma_2+\sigma_3)^2 + 2\psi[(\sigma_1-\sigma_2)^2+(\sigma_2-\sigma_3)^2+(\sigma_3-\sigma_1)^2]}}{2\psi} \geq R_c \tag{10}$$

where σ_1, σ_2, σ_3 are principal stresses, $\psi = \dfrac{R_t}{R_c}$ is a factor of brittleness, R_t, R_c are tensile and compressive strength, respectively. Balandin's failure criterion is derived analytically (Karkashadze, 2014) based on constitutive equations of solid mechanics. It represents a solid failure under the simultaneous action of normal and shear stresses. The concept of equivalent uniaxial state is applicable to a lot of analytical failure criteria and allows such simple mechanical characteristics as compressive and tensile strength being involved.

The criterion (9) can be transformed considering (8) and (10)

$$\sigma_e \geq R_c\left(1 + 1.29\eta \ln\frac{v_n}{v_0}\right). \tag{11}$$

The criterion (11) describes the rock strengthening at a rapid loading. It can be used to estimate rock yielding around an excavation considering an alteration of mining rate, that is, corresponding change in loading a structural element of the rock mass.

The effect of mining rate has been simulated in terms of coal mines of Western Donbass. Physical and mechanical rock properties are shown in Table 1. The depth of mining at the site investigated is 350 m. The 3D-state of rock mass around intersection of the maingate

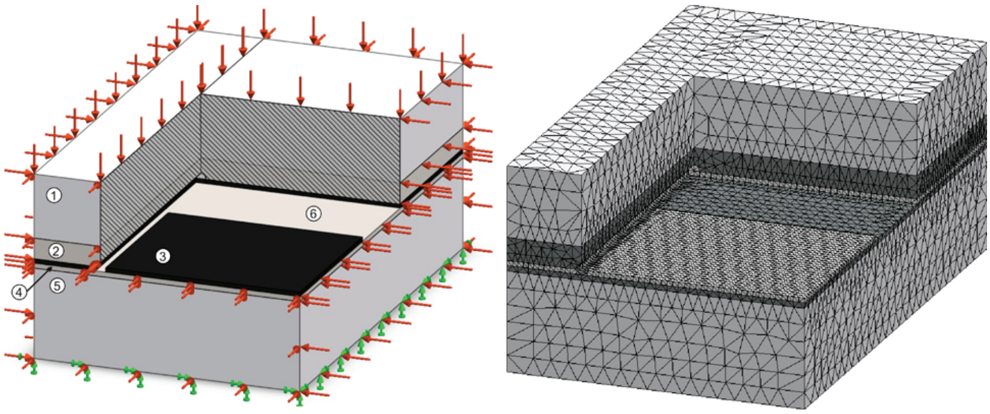

Fig. 4. Design scheme and finite element approximation of area investigated:
1 – main roof, 2 – immediate roof, 3 – coal seam, 4 – immediate floor, 5 – main seam floor, 6 – mine-out space (goaf)

roadway and coalface (Fig. 4) is simulated by the finite element method. For that case the software based on SOLIDWORKS and additional processing modules implementing Balandin's failure criterion in the form (11) are applied. Input data for simulation are considered in terms of real conditions of the mine "Stepnaya" located in Western Donbass. Physical and mechanical properties of the rocks are provided by the mine geological service (Table 1).

Table 1. Physical and mechanical properties of the rocks

Characteristics	Argillite	Siltstone	Coal
Young's modulus, MPa	3193.0	2981.7	11755.2
Poisson' ratio	0.3	0.3	0.3
Compressive strength of intact rock, MPa	32.0	43.0	37.5
Tensile strength of intact rock, MPa	6.5	4.1	3.2
Relative variation of compressive strength	0.35	0.3	0.3

A cross-section which is perpendicular to the coal face and located at 2 m from the maingate roadway is studied in detail.

Different stages of the longwall retreat are studied as well. Stage 1 represents a formation of an arched set-up room to assembly the longwall equipment in initial rock mass (Fig. 5). Stage 2 corresponds to the space increase behind the coalface at a distance of 5 m (excavating the set-up entry). Design schemes are relevant to the technology of mining equipment assembly. To remove the shields of powered support from the arched set-up room to the set-up entry, the coal seam floor is excavated. Each subsequent stage corresponds to an increment of mined-out space in 5 m. Thus, a quasi-static process of longwall retreat is simulated.

The area of failed rocks (area of yielding) is determined according to the criterion (11) at each stage of simulation to fix the critical size of a mined-out space at which a primary roof-caving occurs (Ivanov, 2009).

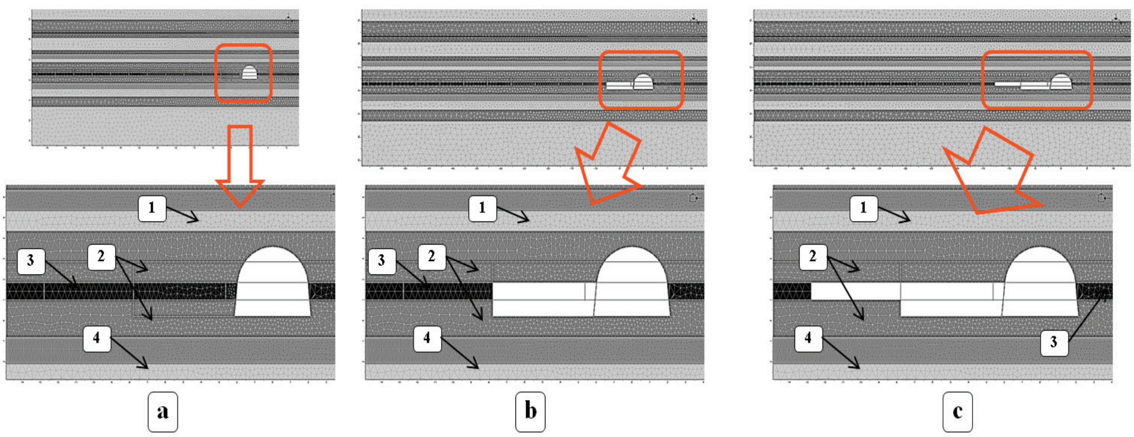

Fig. 5. Design scheme for determining the stress-strain state of the rock mass:
a, b, c – stages 1, 2, …, *n*: 1 – argillite; 2 – siltstone; 3 – coal; 4 – sandstone

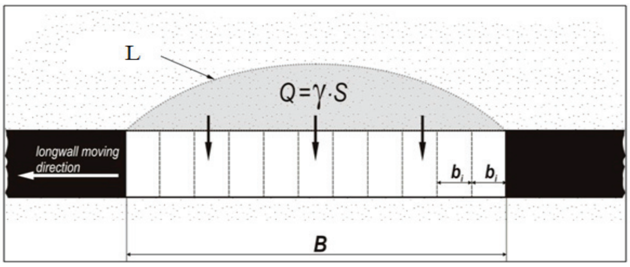

Fig. 6. Condition of a roof collapse

A simplified scheme representing the condition of a roof collapse in the longwall cross-section is shown in Fig. 6. Each simulation step is incremented by a certain amount of space b_i. The area of failure in the roof of the excavation extends at each step as well. Let the mined-out space B be created at step $i = k$ and area of failed rocks is S at this step. The roof caving takes place when a weight of failed rocks Q exceeds a confining force R

$$Q \geq R, \tag{12}$$

where $Q = \gamma \cdot S$. Confining force distributed along the failure area border depends on the rock tensile strength (Shashenko, Surgay and Parchevskij, 1994) and equals: $R = \sigma_t L$. Here, L is the length of a curve bordering the area S.

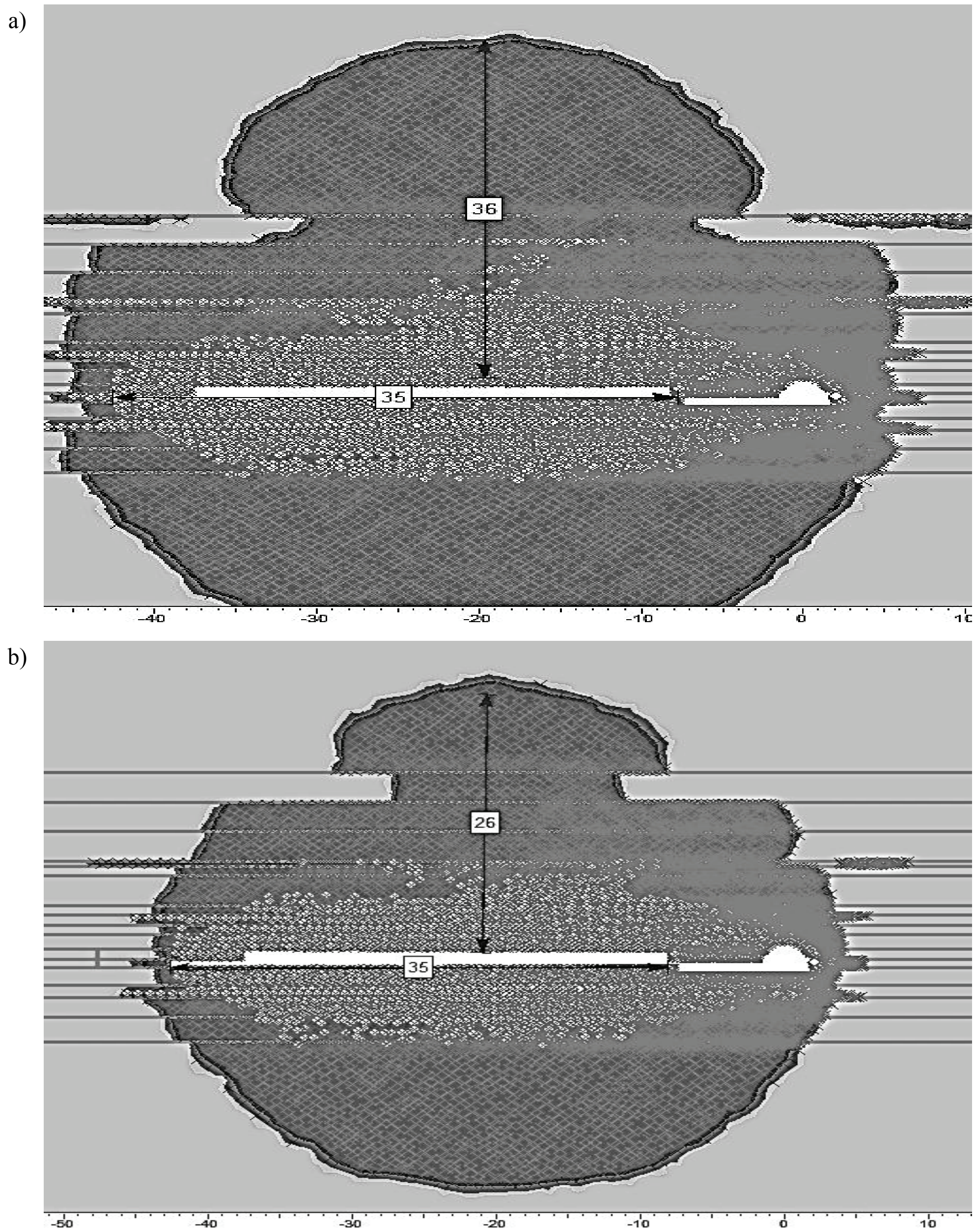

Fig. 7. Area of yielding in the roof of excavation at a different rate of longwall retreatment:
(a) $v_n = v_0$; (b) $v_n = 2v_0$ ($R_n = 1.5\ R_c$)

5. RESULTS AND DISCUSSION

The criterion (12) is verified at each stage of simulation. An initial calculation is carried out at the value $n = v_n/v_0$ which equals 1.0. In this case, the roof collapse occurs according to the criterion (12) if the mined-out space reaches up to 35 m. This result has been confirmed by in-situ measurements in terms of "Stepnaya" mine and registered by the mine surveying service. According to the simulation results the failure zone height reaches up to 36 m in this case (Fig. 7a). Such a size of failure zone should be considered as a critical one indicating the roof collapse under given condition.

Next calculation is carried out on the assumption that a mining rate is to be increased twice ($n = 2$). Then the rock strength increases by 1.5 times according to (8). Respectively, the failure zone does not extend significantly. It makes only 26 m in height at the same size of the goaf (35 m). The weight of rocks Q within the failure zone is not sufficient to overcome the confining force R in this case. Thus, the situation resulting in the roof collapse does not occur (Fig. 7b). At a doubled mining rate the goaf has to be increased up to 50 m to provoke rock yielding within the area of 36 m in height.

Hence, at the mining rate increased twice the critical size of the mined-out space increases by 1.4 times under given conditions. This could have some negative effect. When the roof hangs up for long distances it tends to fall dramatically. The falling roof acts as a piston, displacing the air at sufficiently high velocities to result in damaging the ventilation stoppings and conveyor belts and exposing possible injury to people (Merwe and Madden, 2002).

Under certain condition the rock burst can be affected by a great size of the excavation. Sizable opening in the rock mass provides the facilities for fracture opening and intensive gas release. A risk of the fire becomes more relevant.

Generalization of statistical and simulation results gives a formula related to the size of mined-out space (a goaf) at which the roof collapse occurs under geological conditions of Western Donbass

$$B = \frac{11 \cdot 10^2 k_v}{\gamma} \sqrt[3]{\frac{R_c^2}{H}}. \tag{13}$$

Here, the rate of the longwall retreat is considered with a factor k_v

$$\begin{aligned} k_v &= 1 && \text{at } V \le V_0, \\ k_v &= 1 + 0.8 \ln \frac{V}{V_0} && \text{at } V > V_0. \end{aligned} \tag{14}$$

The value of V_0 can be justified by a statistical data processing concerning the rate of the longwall retreat. The histogram (Fig. 8) shows that the value of 3–4 m/day has the highest frequency under conditions of coal mines observed. The average value of this range (3.5 m per day) can be considered as a "limit" value of a coal face advance.

If the coal face rate is less than the "limit" value, a factor k_v should be taken equal to 1.0. When the speed exceeds the "limit" value, the factor k_v should be calculated accordingly to formula (14) or the diagram in Fig. 9.

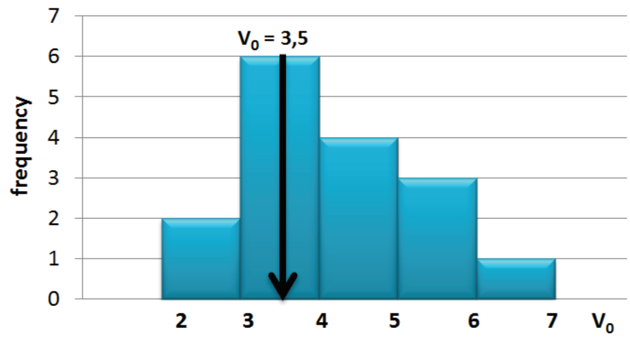

Fig. 8. Histogram of retreatment rate values

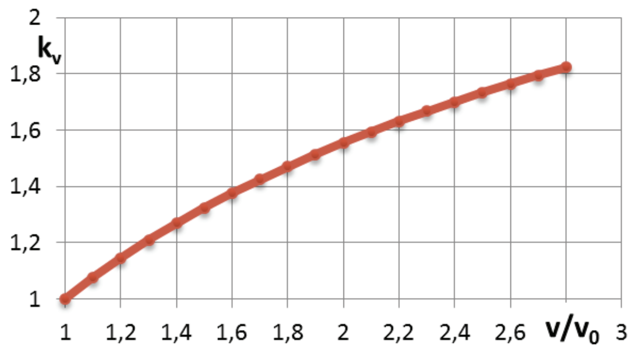

Fig. 9. Factor k_v depending on a rate of a coal face advance

6. CONCLUSIONS

Implementation of an effective coal mining technology (in particular, plow longwalling) facilitates a high rate of mining operations. As a result, the quick excavation causes the rapid change of the rock stress-strain state. This has negative consequences at the initial stage of longwalling. In situ observations show that mining rate increase causes a dramatic roof collapse. The moment of primary roof collapse is one of the basic parameters involved in planning mining activity and it should be determined while considering mining rate.

Statistical data processing shows a general trend stating that mining rate growth causes the increase in the critical mined-out space, thus provoking a primary roof collapse. The correlations can be described approximately by a logarithmic function and this outlines a direction of further study of the phenomena.

Increase in critical mined-out space can be explained using the statistical strength theory. It has clarified the rock strengthening under the action of a rapid load based on the Gumbel probability law and the representation of a rock element failure as a consecutive failure of micro-level links in the material. The developed strength-rate relationship is described by a logarithmic function similar to the relationship proven experimentally (Skipochka and Usachenko, 2006). This strength-rate function describing the strength growth under increasing loading can be used in different calculations associated with the rate of rock stress state alteration and failure.

Numerical simulation gave the possibility to determine an area of yielded rocks taking into account the rock strengthening at rapid excavation and stress redistribution. This allowed us to define a critical size of mined-out space (a goaf) at which the weight of failed rocks exceeds a confining force and the roof collapse occurs. In this way the moment of primary roof collapse can be predicted considering the rate of a longwall retreat.

Generalization of statistical and simulation results permits us to develop a simple formula related to the critical size of the mined-out space at which the roof collapse occurs under geological conditions of the Western Donbass. This technique allows predicting the increase of the critical mined-out space depending on the excavation rate and providing an adequate planning of mining activities to avoid the hazards associated with the join propagation, intensive gas release, dramatic floor heaving and other negative phenomena.

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