ENGINEERING SCIENCES

A COMPREHENSIVE STUDY OF THE STABILITY OF LATERAL ROCKS WITH A SUPPLE SUPPORT

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ABSTRACT

A complex of analytical studies of the stability of lateral rocks with a supple support, where the coal seam roof was a beam model, some laboratory experiments on samples made from optical and equivalent materials, as well as some mine experiments were was carried out. As a result of the studies, it was found that the stability of lateral rocks with a supple support of crushed rock depended on the compaction of the backfill array on which the roof rocks were based. It was experimentally proved that with external force, the coefficient of compaction of the backfill array changes according to the hyperbolic dependence, the maximum values of which were determined as a result of compaction of the supple support consisting of heterogeneous fractions of the starting material of a certain bulk density. The change in the stress-strain state of the lateral rocks in the coal array with a roadway depended on the bending stiffness of the stratified rock mass and the parameters of flexible structures used to support the roadway. When supporting the roadways with supple support structures, the convergence of the lateral rocks on the contour of the stoop roadway was observed exponentially until the support was completely compressed, while supporting the roadways with coal pillars, the displacement of the rocks on the contour of the roadway increased having linear dependence due to the destruction of the pillar. The stability of the lateral rocks, which determines the operational state of the excavation roadways, was ensured by the use of supple supports or stowing the mined-out space, taking into account the reasonable granulometric content of crushed rock, which ensured the maximum values of the compaction coefficient of the backfill array when the roof and coal seat got deformed.

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Introduction. Mining steep coal seams of Donbas is characterized by a relatively low level of technical and economic indicators. To a large extent, this is due to lack of reliable ways to ensure the stability of lateral rocks and roadways. If we look at deep mines, the deeper a mine is, the more vivid such natural factor as a stratification of lateral rocks is, which leads to a deterioration in the stability of roadways.

With traditional methods of controlling the rock pressure in the longwall by a complete collapse or holding the roof on hogs and supporting roadways with wooden structures (a bunch of posts) or coal pillars of a limited size, it is impossible to resist deformation of the disturbed rock in the mine array. Therefore, the development of measures aimed at maintaining the stability of lateral rocks while mining very deep steep coal seams will not only help to maintain the roadways in an operational condition, but also increase the safety of miners.

Analysis of recent achievements and publications. The efficiency of mining steep coal seams and the safety of mining operations largely depend on a method of controlling the roof in the longwall. The study of the process of displacements and deformations of sedimentary rocks showed [1, 2] that while mining coal seams in a disturbed rock, some peculiar zones of displacement are observed, the size of which depend on the method used to control the roof in the longwall.

It is known [3,4] that a method of stowing the mined-out space is most favorable one for the condition of lateral rocks in a mine array with roadways when the roof rocks along the entire length of the lava (of a floor height), are supported by the stowing mass. The studies previously conducted in DonUGI and DonNTU found that a backfill array prevents the development of intense cracking in the zone of the roadways and creates zones of stable rocks behind the longwall [5, 6, 7]. When applying this method, sudden lowering of the main roof is eliminated, which means that dynamic loads are minimized.

The experience shows that the loss of stability of any systems is one of the main causes of many modern catastrophes and accidents, including in mining, which means, ceteris paribus, the likelihood of natural hazards in deep mines will always be greater if there is no supple support in the mined-out space, which is a stowing array [6, 7].

Research objectives. The research objectives are to determine the conditions for the stability of lateral rocks with a supple support and its effect on the operational state of roadways while mining steep coal seams.

Theoretical models. It is known [8, 9, 10] that when studying the rock pressure in roadways, it is necessary to establish the nature of the limited state of roof rocks. Assessing the stability of the lateral rocks in the coal array with roadways, one should take into account a certain state of the geomechanical system, when a certain form of equilibrium is maintained in the bottomhole space of the longwall and behind it, and the deformation of the lateral rocks does not affect the performance conditions of the roadways.

According to the hypothesis of beams and articulated-block displacement of the stratified rock stratum [4,10], we assume that the roof of the coal seam is deformed and collapses in the longwall like a beam with supporting connections. Based on such a model, the roof of the mined coal seam is studied as a beam with length of L_b (m), height h (m) and width b (m). The beam has a fixed articulated support at point A, and a movable articulated support at point B, the distance between which is a, (m). The section of the BC beam is a console, the length of which is l_k (m) (Fig. 1a, b).

The beam has a constant cross-section and bending stiffness. It is in equilibrium. Influenced by the load, the system is deformed and some loads are observed. In one case, an evenly distributed load q, (N / m) influences a limited zone of the beam (zone AB), and towards the end of the console, i.e. point C some external force P (H) is applied (Fig. 1, a). In other case, a uniformly distributed load q, (N / m), acts along the entire length of the beam (Fig. 1b). The calculations consider a statically indeterminate system. For such systems, the method of forces is used [11,12,13,30].

Let us study a static problem of transverse bending of the beam.

In relation to the beams with pinching, it is recommended to use a hinge in the tab for the formation of the main system [13]. We obtain a canonical equation of the force method as

$$\delta_{11} X_1 + \Delta_{1p} = 0 \tag{1}$$

and its coefficients

$$\delta_{11} = \frac{a}{3EI} \tag{2}$$

and

$$\Delta_{1з} = \frac{qa^3}{24} - \frac{ql_k^2 l}{12}, \tag{3}$$

where δ_{11} is a displacement from the action of a single force applied to the beam, m.

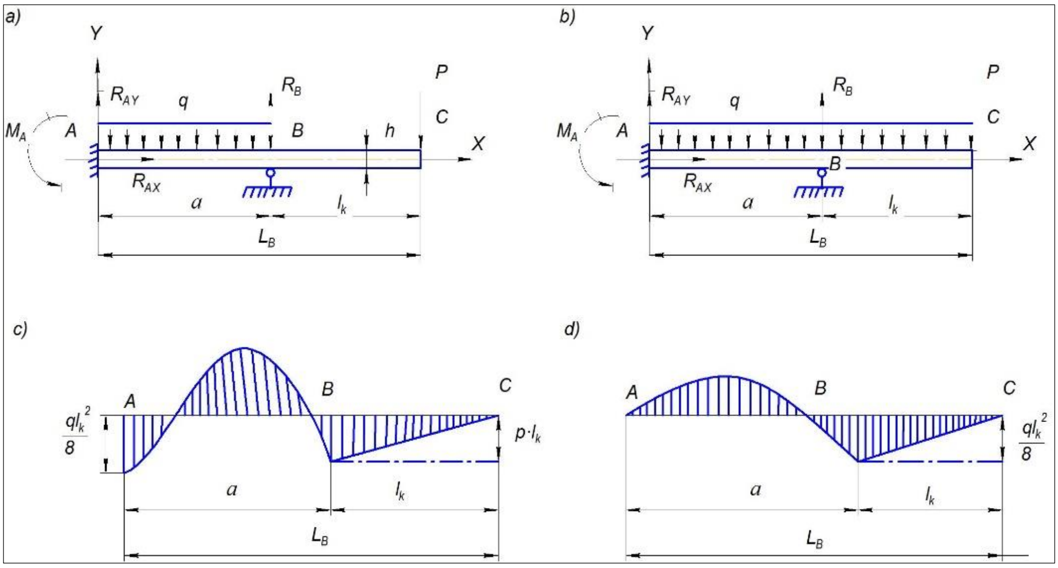

Fig. 1. Calculation scheme for determining the stability of the coal seam roof (a, b) and the diagram of bending moments on the compressed grain of the simulated beam (c, d): a, b) with an uniformly distributed load q, (N / m) within the bottomhole space (section AB) and the concentrated load P (N) at the end of the console; c, d) with an uniformly distributed load q (N / m) along the entire length of the beam; h is the thickness of the beam, (m); a - beam span, (m); l_κ - console length, (m); L_b- beam length (m), RAX, RAY- reaction in support A along the X and Y axes (N), RB- reaction of support B, (N)

When calculating the load coefficient Δ_{1p}, we plot the diagram of bending moments because of the load as two diagrams: a triangular one, from the reference moment $\frac{ql_k^2}{12}$ and a parabolic one from the load q, whose area is equal to $\frac{qa^3}{12}$.

From the canonical equation (1) we determine the moment on the left support (point A)

$$X_1 = -\frac{qa^2}{8} - \frac{ql_k^2}{4}. \tag{4}$$

In the case when a concentrated load is present on the edge of the console (Fig. 1, a), the moment on the left support is

$$X_1 = \frac{qa^2}{8}, \tag{5}$$

and on the support B it is

$$X_1 = P \cdot l_k, \tag{6}$$

as it can be seen from the diagram of bending moments (Fig. 1c).

In relation to the left support, when an uniformly distributed load q, (N / m) acts on the entire support of the beam and $l_k \geq a \cdot \frac{\sqrt{2}}{2n}$ (Fig. 1b), X1 = 0, acts on the support B

$$X_1 = \frac{ql_k^2}{8}, \tag{7}$$

which is confirmed by the diagram depicted in Fig. 1d.

The analysis of the bending moment diagrams on the compressed fiber of the simulated beam (Fig. 1 c, d) enables us to state that in all the cases there is a compression of the rocks of the immediate roof over the roadway (section AB). That happens because the array gets different pressures from the action of an uniformly distributed load q, (N / m), concentrated force P, (N) and a cantilever beam (section BC) (Fig. 1 a, b, c, d).

It was experimentally determined that when the studied model was loaded on a limited area with a uniformly distributed load q, (N / m), and a force P, (N) was applied to the edge of the console, the beam underwent a kink by the array (Fig. 1c). When a uniformly distributed load q, (N / m), acts along the entire length of the beam, the moment on the left support is always zero. Replacing support B with a flexible one will contribute to a lateral pressure in the bedding plane and the formation of a stable arch above the roadway where compressive stresses are observed, the magnitude of which depends on the load and the flexibility of the support B (Fig. 1 g).

It is known [14, 15, 16, 17, 29] that the natural frequency of beam vibrations in the system under study is determined by the expression

$$p = \sqrt{\frac{1}{m\delta_{11}}}, \tag{8}$$

where m is the mass of the beam, kg.

Using the method of forces [13], we determine the reaction at point B when P = 1. we derive a canonical equation as

$$\delta_{22} X_B + \Delta_{2p} = 0, \tag{9}$$

wherein

$$\delta_{22} = \frac{a^3}{3EI}, \tag{10}$$

and

$$\Delta_{2p} = -\frac{a^2(3l_k + 2a)}{6EI}. \tag{11}$$

Taking into account the expressions (10) and (11), we have

$$\frac{a^3}{3EI} X_B - \frac{a^2(3l_k + 2a)}{6EI} = 0, \tag{12}$$

and the value X_B is determined by the expression

$$X_B = \frac{3l_k + 2a}{2a}. \tag{13}$$

We determine the value δ_{11} using the Mohr integral and the Vereshchagin rule [13], according to the expression

$$\delta_{11} = \int \frac{\overline{M_1} \cdot M}{EI} dx = \frac{l_k^2(4l_k + 3a)}{12EI}. \tag{14}$$

Knowing the value δ_{11}, you can determine the stiffness of the beam

$$C_B = \frac{1}{\delta_{11}} = \frac{12EI_x}{l_k^2(4l_k + 3a)}. \tag{15}$$

In this study, the stiffness of the geomechanical system C_{sys}, (N / m), should be determined taking into account the stiffness of the support $C_{\sup}$, (N / m), i.e.

$$C_{sys.} = C_B + C_{\sup}. \tag{16}$$

Then, the expression (8) is as

$$p = \sqrt{\frac{12EI_x}{ml_k^2(4l_k + 3a)} + C_{\sup}}, \tag{17}$$

from which it can be concluded that with the stiffer the support is $C_{\sup}$, (N / m), the stiffer the system is, which means that the frequency of natural vibrations of the beam increases.

Figure 2 shows the graphs of the change in the frequency of natural vibrations of the beam *p* (1 / s) from the lengths of the consoles l_k (m), taking into account the rigidity of the supports. In all cases, the length of the section AB, i.e. the parameter *a*, (m), is equal to *a* = 5.0 m.

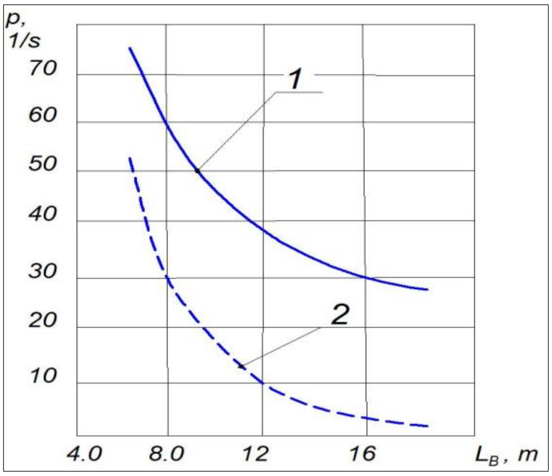

Fig. 2. Graphs of the changes in the natural vibration frequency p (1 / s) of the beam depending on its length LБ, (m): 1- when leaning on a support whose rigidity is equal to S_{sup} = SB; 2- when resting on a support whose rigidity is S_{sup} = 0.25SB; $L_B = a + l_k$, where a = 5 m.

From Fig. 2 it is seen that with an increase in the lengths of the consoles, for the given stiffness of the support, the frequency of natural vibrations decreases. It was determined that with a rigid support, in the case when $C_{sup.} = C_B$, the natural frequency of the beam decreases from p = 72.0 1 / s to p = 36.0 1 / s, with an increase in the length of the beam from L = 7 to L = 15 m (Fig.2, dependence 1). With a compliant support $C_{sup} = 0.25C_B$, the rigidity of which is equal, the frequency of natural oscillations in the system is reduced from *p = 52.0* 1 / s to *p = 8.0* 1 / s (Fig. 2, dependence 2). It is obvious that in order to improve the stability of the roof, as well as to decrease the amount of displacement of the lateral rocks under the action of a static load behind the longwall and mitigate the effects of dynamic loads, especially with caving the main roof, it is advisable to use supple supports.

Experimental models. In order to assess the effect of the compliant support on the stability of roof rocks of the coal seam under dynamic loads, the studies were carried out on models of equivalent materials, when the roof rock was a beam with a length of L_b = 0.6 m, a thickness of h = 0.02 m and a width of b = 0.04 m The beam mass corresponded to m_b = 1.4 kg, the elastic module was E = 8800 MPa, and the density ρ = 2100 kg / m^3. The beam was made of a sand-cement mixture, according to the recommendations [19,20], and at point A had a rigid fixation. The free end of beam C rested on a compliant crushed rock support. The span of the beam AB was equal to *a* = 0.2 m (Fig. 3).

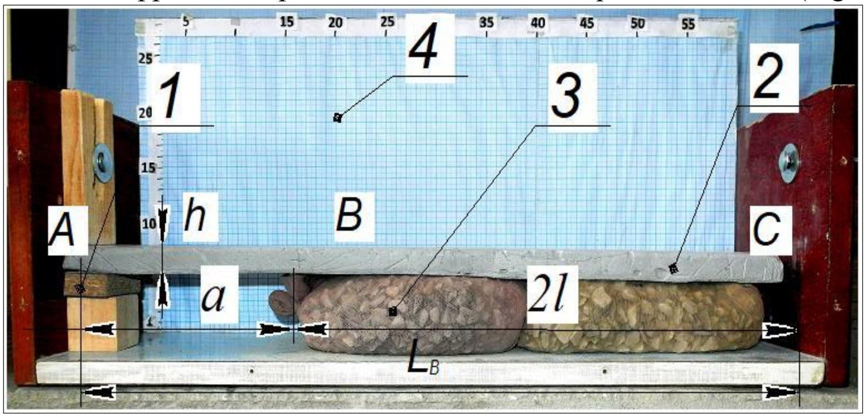

Fig. 3: Photo of a stand for studying the stability of a coal seam roof with a compliant support: 1- rigid fixing of a beam support; 2- beam- direct roofing; 3- malleable crushed rock support; 4-coordinate grid: L_b- beam length, L_b = 0.6 m; h - beam thickness h = 0.02 m; a - beam span a = 0.2 m.

When testing the models, all the force criteria were determined in accordance with [18, 21].

Granulometric composition of filling material from crushed rock for a supple support, its bulk density $\rho_{b.d.}$ (kg / m^3) and voidness M (%) were determined in accordance with [21]. The compaction coefficient k_{comp} of the filling material was calculated as the ratio of the volume that the rock mass occupied before its compaction to the volume that it occupied after the compaction.

In order to determine the compaction coefficient, the starting material was used when the crushed rock consisted of inhomogeneous fractions and the material dispersed into standard fractions. For testing, some special steel cylinders were used according to [20], in which crushed rock was placed, placed between plates on a press, after that it was compressed (Fig. 4a) and then it was removed (Fig. 4b).

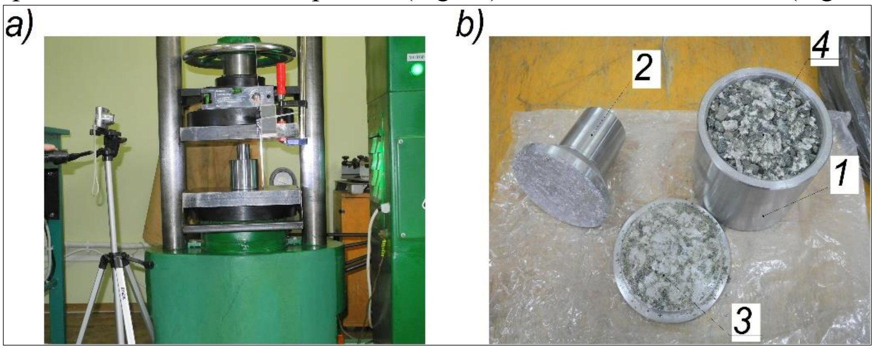

Fig. 4. Photos of the experimental equipment for determining the compaction coefficient for the $k_{упл}$ of the crushed rock: a) a press for compression of a steel cylinder; b) a steel cylinder with a removable bottom and a plunger: 1- steel cylinder; 2- plunger; 3 - removable bottom; 4- compacted crushed rock

Table 1. Granulometric composition of crushed rock

Fraction size, mm	>5	5-4	4-3	3-2	2-1	1-0.1
Percentage, %	1	17	22	25	23	12

Table 2. Laboratory data after dispersion of crushed rock into fractions

No	Fraction size, mm	Bulk density $\rho_{b.d.}$ kg / m^3	Voidness M, %	Compaction factor, k_{comp}
1.	5-4	1680	20	1.49
2.	4-3	1740	18	1.44
3.	3-2	1870	11	1.43
4.	2-1	1880	11	1.31
5.	1-0.1	1990	6	1.13
6.	5-0.1	1840	13	1.53

The laboratory data on the determination of the granulometric composition of crushed rock, its bulk density $\rho_{b.d.}$ (kg / m^3), voidness M, (%) and compaction coefficient k_{comp} are presented in Tables 1 and 2. For each fraction size, there were 10 tests. Using the experimental data of Table 2, the dependencies that reflected the change in the crushing rock compaction coefficient k_{comp} of a certain bulk density $\rho_{b.d.}$, (kg / m^3) (Fig. 5 a, b) were derived.

Figure 5a shows a linear relationship after processing the experimental data, which has a coefficient of $R^2 = 0.59$. Processing the same data using a polynomial dependence gives the coefficient $R^2 = 0.83$ (Fig. 5b). Therefore, this enables us to state that the change in the compaction coefficient k_{comp} of crushed rock of different particle size distribution of a certain bulk density $\rho_{b.d.}$, (kg / m^3) is subject to the parabolic dependence shown in Fig. 5 b.

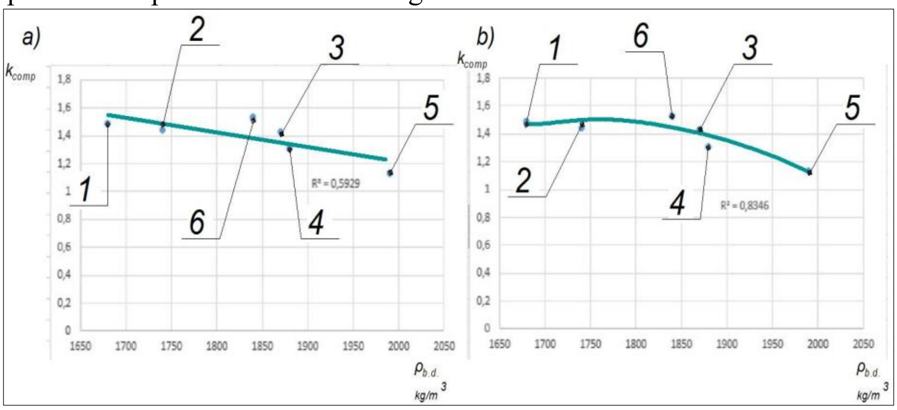

Fig. 5. Graphs of changes in the compaction coefficient k_{comp} of crushed rock of a certain bulk density $\rho_{b.d}$, (kg / m^3) for various approximations of experimental data: a) in the form of a linear relationship ($R^2 = 0.59$); b) in the form of a polynomial dependence ($R^2 = 0.83$); 1- fraction (4-5) mm; 2- (3-4) mm; 3- (2-3) mm; 4- (1-2) mm; 5- (0.1-1) mm; 6- (0.1-5) mm

In order to determine the rigidity of the supple support, the deflection of the beam X, (m), was established when a rock mass of m = 0.1 kg was dropped onto it from a height of h = 0.3 m and an inelastic impact was recorded.

When testing models of displacement, the beams were recorded with a digital camera. With photographic images and the pixel coordinates of the points, using the basic principles of photogrammetry [23], the beam position in space was determined before and after the action of an external force on it. The experimental data are presented in table.3. It should be noted that the displacement of the beam X (m) recorded in the experiments corresponds to the equivalent static force P_{eq} (N), the value of which can be determined by the expression, as in [22]

$$P_{eq} = S \cdot p, \tag{18}$$

where $S = m \cdot V$ is the magnitude of the shock pulse, kgm / s;

$p = \sqrt{\frac{g}{x}}$ - the frequency of natural vibrations of the beam in the studied system, 1 / s.

Under the force of a load falling on a beam, the impact duration t_{dur} (s) can be determined as in [24], by the expression

$$t_{dur} = \frac{\pi}{2} \cdot \sqrt{\frac{x}{g}}, \tag{19}$$

and the maximum compression ΔX, (m) of the compliant support, according to the expression [18]

$$\Delta X = \frac{m_B g}{c} + \sqrt{\frac{m_B^2 g^2}{c^2} + 2gh\frac{m^2}{c(m+m_B)}} \tag{20}$$

Table 3. Data from experimental studies to determine the rigidity of a supple support

No	Fraction size, mm	Beam displacements, x, m	The rigidity of the supple support, C, N / m
1.	5-4	0.0026	5700
2.	4-3	0.0024	6100
3.	3-2	0.0023	6400
4.	2-1	0.0019	7700
5.	1-0.1	0.0015	9800
6.	5-0.1	0.0032	4600

Fig. 6 shows graphs of changes in the displacement of the beam ΔX (m), the duration of the impact t_{dur}(s), after the rock block weighing *m = 0.1* kg from the height of *h = 0.3 m* onto the beam that is equivalent to the static force P_{eq} (N) corresponding to this displacement and compaction coefficient k_{comp} of crushed rock of a certain bulk density $\rho_{b.d.}$, (kg / m^3).

It can be seen from the above dependences that with an increase in the bulk density of crushed rock to $\rho_{b.d} = 1820$ (kg / m^3), the beam displacements increase to a certain value which equals $x = 0.0038$ *m*, and then decrease (Fig. 6, dependence 1). The same regularity is observed in the analysis of the dependence, reflecting the change in the duration of the impact t_{dur} (s) (Fig. 6, dependence 2). The minimum value of the equivalent static force P_{eq} (N), which equals $P_{eq} = 13.5$ N, corresponds to the maximum displacement of the beam with a bulk density that equals $\rho_{b.d}$ = 1820 kg / m^3 and the maximum value of the compaction coefficient k_{comp} = 1.53 (Fig. 6, dependence 3 and 4).

Fig. 7 shows graphs of the change in stiffness C, (N / m) of the supple support and its maximum compression at a certain bulk density $\rho_{b.d}$, (kg / m^3) of crushed rock, taking into account the compaction coefficient k_{comp}.

It can be seen from the obtained dependences that the minimum stiffness of the supple support, when C = 4700 N / m, corresponds to a bulk density which is equal to $\rho_{b.d}$ = 1820 kg / m^3 (Fig. 7, dependence 2).

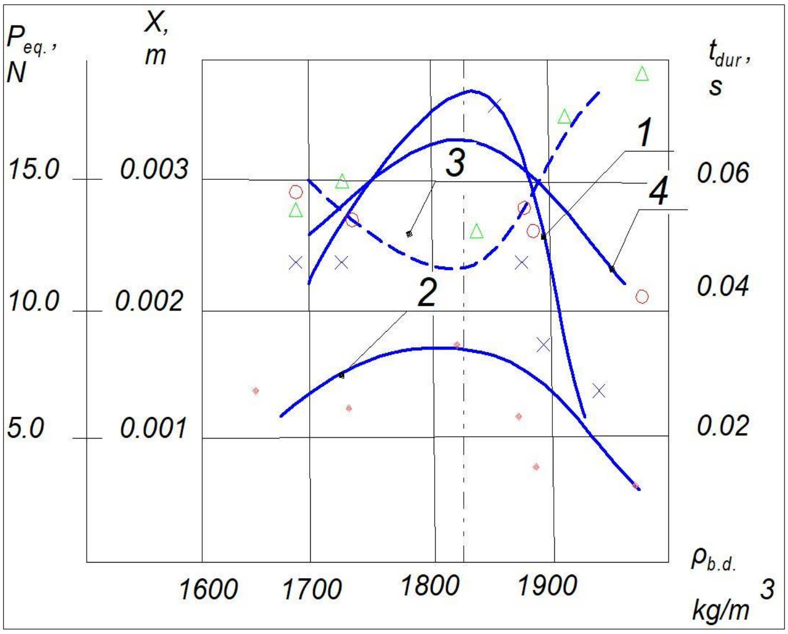

Fig. 6. Graphs of changes in beam displacement X, (m), impact duration t_{dur} (s), equivalent to the static strength $P_{eq.}$ (N) and compaction coefficient k_{comp} of crushed rock of a certain bulk density $\rho_{b.d}$ (kg / m^3): 1-X, (m); 2- t_{dur} (m); 3- P_{eq}, (N); 4 k_{comp}

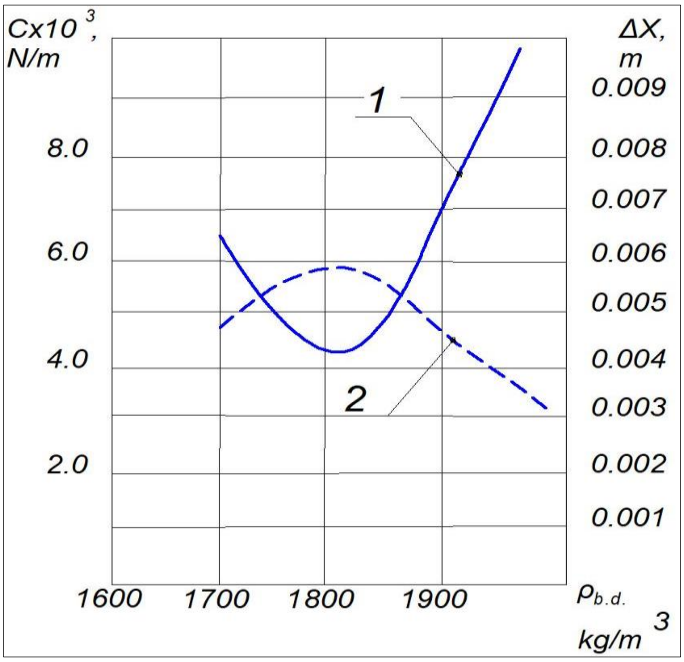

Fig. 7. Graphs of changes in the stiffness of supple support C, (N / m) and its maximum compression ΔX, (m) after a load of mass m, (kg) falls on a beam at a certain bulk density of crushed rock $\rho_{b.d}$ (kg / m^3): 1-C, (N / m); 2- ΔX, (N / m).

The use of the research results in solving the problem of the stability of local preparatory roadways, when a roadway is located between the solid coal and the backfill array, which enables us to justify the safety of the preparatory roadways while mining steep seams. We use the obtained patterns in laboratory studies on optical models to assess the stress-strain state of the mine array, taking into account the applied method of supporting the haul roadway.

The purpose of laboratory research was to determine the patterns of stress distribution around the haul roadways with different methods of protection, when there is a decrease in negative rock pressure in a mine array.

The theoretical foundations of modeling by photoelasticity are described in the works of a number of researchers [25,26].

For the experiments, optical models were made of igdantine (material composition: glycerin 30%, gelatin 25%, and water 45%). Model dimensions: 300x300 mm, thickness 20 mm, geometric scale 1: 100. The models which imitated a roadway stratum with different methods of support: pillars of coal, rolling cogs from sleepers, or stowing a mined-out space. The similarity criteria, elastic and optical model constants were determined in accordance with the methodology developed in the IHD named after A.A. Skochinsky [25]. The study of the models was carried out on the polarization installation PPU-4, a known method of matching colors and distribution bands of shear stresses [26]. A total of 16 models were worked out. The stress distribution was studied in the vicinity of the haul roadway, passed through a coal seam with a thickness of m = 1.0 m. The roadway was an arch one (height *2.5* m, width *2.5* m). The studies were carried out on models with layers that were considered as the direct and main roof and soil of a coal seam, the thickness of which corresponded to 2 m and 4 m, where m is the thickness of the developed seam, m. The rigidity of the rolling cogs and the stowing mass corresponded to $c_m = 35$ N / m, pillars of coal - $c_p = 150$ N / m. During the simulation, the method of controlling the roof by a complete caving was imitated, when the haul roadway was supported by pillars of coal or by keeping it on cogs, when rolling cogs from sleepers were used and complete filling of the mined space, when a wide entry strip from the rock was located over the preparatory roadways.

The model was loaded according to the scheme in which the vertical load corresponded to a depth of H = 1200m, and the horizontal load was created by repelling the side walls of the model. The dip angle was $\alpha = 60^o$. Rolling cogs from sleepers, mounted over the roadway, had a contraction of up to 50% and were modeled with foam rubber impregnated with paraffin. The filling of the mined space from crushed rock, had a contraction of up to 30% and was modeled by foam rubber (taking into account paraffin impregnation).

When analyzing the static field of shear stresses in a mine array, it was assumed that the bulk stress state of the lateral rocks, as they move away from the mine contour, will change from a state close to generalized tension to generalized compression in the depth of the array [27].

Fig. 8 a, b shows diagrams of models of the static field of tangential stresses in a coal array while protecting the haul roadway by coal pillars of different sizes.

An analysis of the static stress field in the models indicates that when supporting the seam haul roadway with coal pillars of height h = 4 m, we have the maximum stress concentration in the lateral rocks and around the haul roadway (Fig. 8a). With an increase in the height of the solid pillar to h = 8 m, i.e. 2 times, the stress concentration is somewhat modified. This occurs as a result of an increase in the area of contact between the pillar and the rocks of the roof and soil of the coal seam (Fig. 8b). As the height of the pillar increases, the stress concentration moves to the depth of the array. The use of the pillars of coal for supporting haul roadways is accompanied by their intensive forceful impact from the roof and the coal seat. The study of model patterns of stress distribution around the haul roadway when supporting with coal pillars of different sizes shows that in all cases the greatest tangential stresses are concentrated near the production contour and at the upper pillar boundary, at the bend of the roof rocks and coal seat of the mined seam.

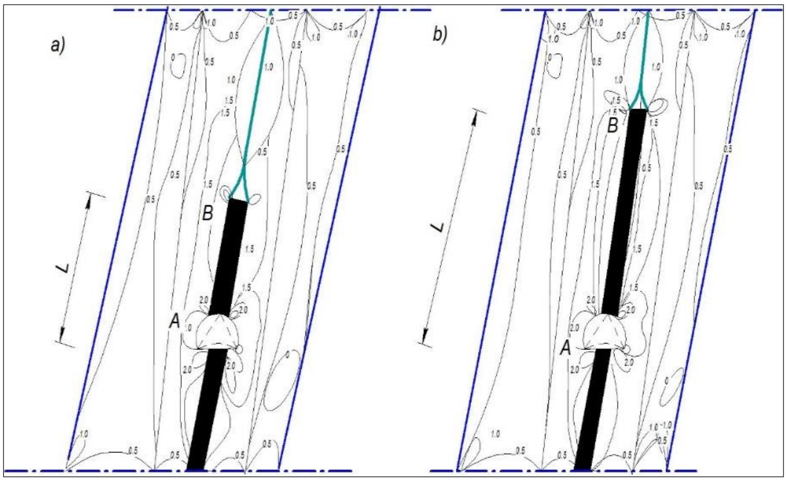

Fig. 8 Model diagram of the static field of tangential stresses around the haul roadway when supporting with coal pillars: a) pillar height h = 4 m; b) the height of the support pillar h = 8 m; L is the length of the roof area supported with a pillar, m

From the analysis of the models it can be concluded that the negative rock pressure around the preparatory roadways are minimized when using compliant supports in the form of rolling cogs from wooden sleepers or filling the mined space as support structures (Fig. 9 a, b).

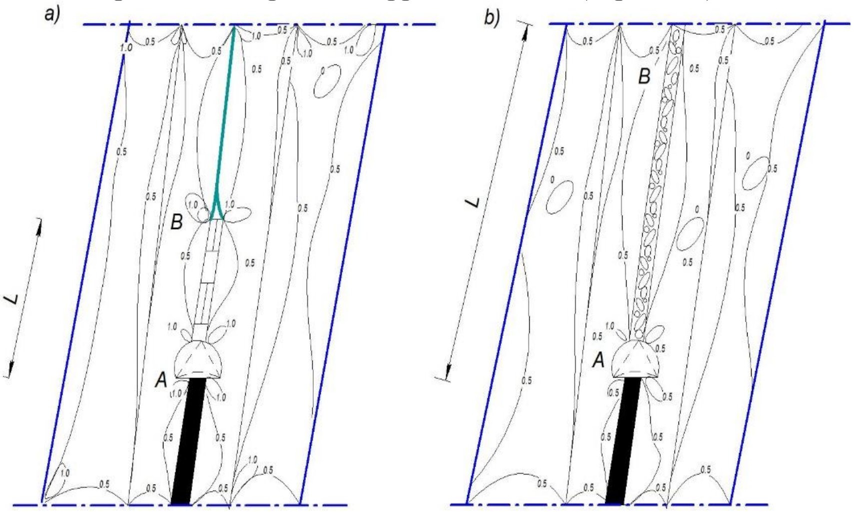

Fig. 9. Scheme of models of the static field of tangential stresses in a coal array with roadways a) while supporting the haul roadway by rolling cogs from wooden sleepers; b) when supporting the roadway by filling the mined space; L - length of the roof section supported by the support structure

When protecting section preparatory workings by rolling cogs from sleepers, which are placed above the haul roadway, the concentration of stresses in the lateral rocks noticeably decreases compared to support with coal pillars (Fig. 9a). This occurs due to the smooth deflection of the roof, but on a limited contact area of the rolling cogs with the roof and the coal seat.

Figure 9b shows a diagram of the model of the static field of tangential stresses around the haul roadway, with the method of controlling the roof in the longwall by filling the mined space.

Experimental data indicate that the use of filling the mined space for supporting the lateral rocks reduces the stress concentration in the coal array around the haul roadway compared to supporting with coal pillars. It is obvious that the placement of the filling mass along the entire length of the mined space of the longwall (behind the longwall face), in the form of a compliant strip above the roadway, provides a smooth deflection of the lateral rocks. Redistribution of stresses at the boundary between the contact of the roof rocks and coal seat with the backfill array occurs due to an increase in the area of the actual contact of the settling rocks with the support structure (Fig. 9 b).

It is assumed [28] that the change in the stress-strain state of the lateral rocks occurs due to the modification of the stress concentration zones and depends on different sizes and the rigidity of the support structures. The degree of influence of the stiffness of the pillars or pliable support structures located above roadway is proposed to be estimated by the coefficient k_m^τ, according to the expression

$$k_M^\tau = \frac{cL^3}{EI}, \tag{21}$$

where c - the rigidity of the security structures, N / m;

EI - the bending stiffness of the rocks of the immediate roof, N · m2;

L - length of the roof section supported by support structures, m.

In relation to the studied models, we estimate the values of the coefficient k_M^τ.

Fig. 10 shows graphs of the change in the coefficient k_m^τ from the length of the roof section supported by the support structure.

It can be seen from the above dependences (Fig. 10) that with an increase in the stiffness C (N / m) of the supports in the model and the length of the section L, (m), the coefficient kmτ increases. It was found that at $C = 35$ N / m and $L = 0.1$ m, the values are $k_m^\tau = 0.0018$, and at $C = 150$ N / m and $L = 0.1$ m, the values are $k_m^\tau = 0.01$, which is 5.5 times higher. This means that a decrease in the rigidity of the supports and an increase in their size can significantly reduce the stress-strain state of the lateral rocks in the mine array and ensure their operational condition.

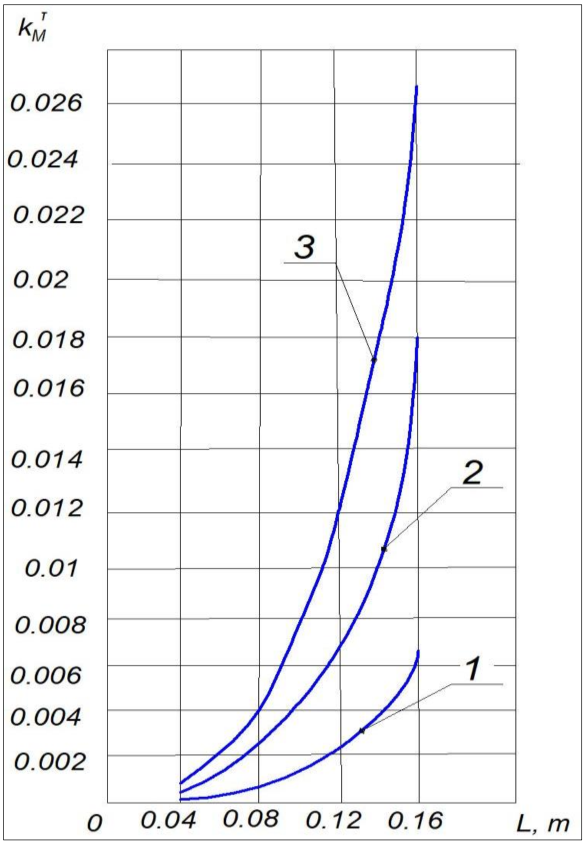

Fig. 10. Graphs of changes in the coefficient k_m^τ in the model depending on the length of the section L, m: 1 - rigidity of the support structure with c= 35 N / m; 2- rigidity of the support structure at c = 100 N / m; 3- rigidity of the security structure at c = 150 N / m

Mine experiment. In order to study the features of the rock pressure on the contour of the haul roadway with different protection methods, some field observations of the displacement of the lateral rocks were made in the conditions of the Tsentralnaya mine at the Toretskugol state mine while mining l_3 at a horizon of 1146 m.

At a specially equipped monitoring station, using the VNIMI tape measure, the displacement of the lateral rocks on the contour of the roadway was determined while finding the convergence of the benchmarks with respect to each other in the directions that are most characteristic for a steep fall. The measurement error did not exceed ± 2 mm. The layout of the benchmarks at the monitoring station is shown in Fig. 11.

While making experimental observations, the displacement of control points located along the contour of the roadway was determined along the roadway L, (m).

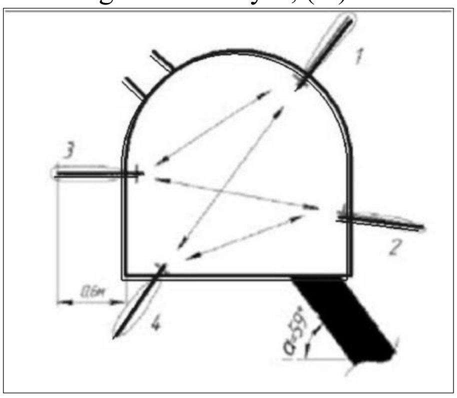

Fig. 11. Diagram of a monitoring station for determining the displacements of the lateral rocks on the contour of the haul roadway of the seam l3 at the horizon of 1146 m of the Central mine: 1,2,3,4- benchmarks; 1-3, 1-4, 2-3, 2-4 – convergence of benchmarks 1.2 in the direction of benchmarks 3.4.

The experiments in the conditions of the Tsentralnaya mine were carried out in the haul roadway of the seam l3at a site with a length of l = 55 m when it was supported by hogs and at a site with a length of l = 78 m when coal pillars were used. The cross-sectional area of the roadway is S = 8.5 m^2, the distance between the frames of the AP-3 roof support with a wooden puff is 0.8 m. The roadway was mined by drilling and blasting operations. The speed of mining v_d = 18m / month. The speed of extraction. = 12m / month. The size of the coal pillars was h_p = 8m, l_p = 5m, where h_p is the height, m; l_p- length, m.

The thickness of the coal seam l3 in the conditions of the mine "Central" m = 1.17 m, the angle of incidence of the seam α = 59º. A clay shale of medium stability with a thickness of up to m = 4.0 m lies in the immediate roof. The main roof was represented by sand shale with a thickness of up to m = 7 m. A clay shale with a thickness of up to m = 15 m lies in the coal seat. The lead of the roadway was L = 100 m.

Processing of experimental data on rock displacement is presented in the form of dependencies shown in Fig. 12a, b. According to the results of measurements of the convergence of the benchmarks in the mine, it was found that the maximum displacements, when supporting the roadway with bunches of hogs, are marked at a distance l = 55m behind the longwall, in the directions 1-4 and 1-3, when U1-4 = 350mm, U1-3 = 290mm (Fig. 12a). When supporting the roadway with coal pillars, the convergence of the benchmarks in these directions were, respectively, U1-4 = 440mm, and U1-3 = 320mm, at a distance l = 75m behind the longwall (Fig. 12b).

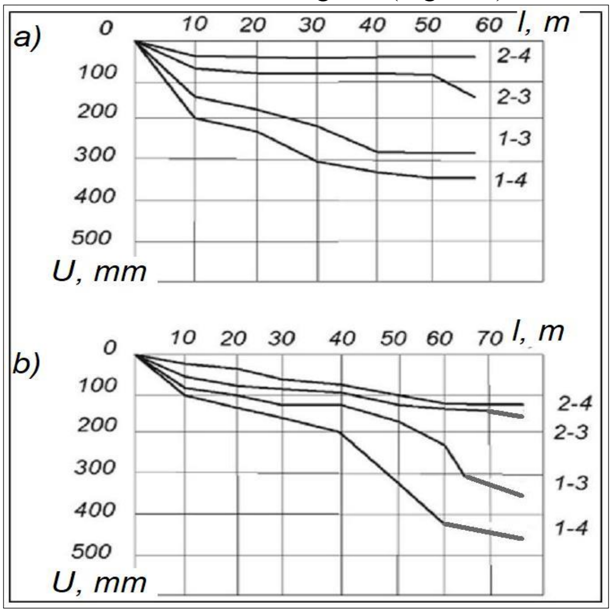

Fig. 12. Graph of measurements of rock displacements U, (mm) on the contour of the haul roadway of the seam l_3 along its length l, (m): a) when supported by bunches of hogs; b) when supporting with coal pillars.

When conducting experimental studies, it was found that the greatest convergence of benchmarks took place from the side of the seam roof, as a result of which the arch support was deformed, and therefore, the cross-sectional area of the roadway was reduced. It is characteristic that the rock displacements from the side of the hanging side, in all cases, were represented by a layered bending of the roof. The clay shale lying in the rocks of the immediate roof got broken by a series of cracks, as a result of which there was a rock fall into the roadway. While supporting the preparatory roadway with wooden structures, behind the mark l = 50m behind the longwall, the displacements of the lateral rocks are almost eliminated and stabilized. In the case of supporting the roadway with coal pillars, such a pattern was not seen.

Thus, as a result of theoretical and experimental studies of the stability of lateral rocks with a supple support, its influence on the operational state of roadways was determined. It was proved that while mining edge coal seams, the stability of the roof rocks within the bottomhole space and behind it depends on the parameters of the support and the magnitude of its compliance. Due to the increase in the flexibility of the supports, the stiffness of the geomechanical system decreases, when, as a result of the limited deflection of the coal seam roof, a stable arch was formed over the haul roadway, and the action of compressive stresses was observed in the bedding plane.

It was recorded that under the impact of falling rock on the roof of a coal seam, its stability depended on the magnitude of the external force, the dynamic stiffness of the compliant support and the compaction coefficient of the backfill array, on which the rocks of the hanging side were based.

It was established that the stability of roofing rocks in the simulated geomechanical system was ensured when the emptiness of the backfill array is M = (7-20)%, consisting of heterogeneous fractions of crushed rock, when the compaction coefficient increases to maximum values (k_{comp} = 1.53) with a simultaneous increase in bulk density. At the maximum value of the compaction coefficient of the backfill array, the maximum compression of the compliant support occurred.

When using the tabs of the worked-out space, when a uniformly distributed load acts along the entire length of the worked-out part of the longwall, a smooth deflection of the lateral rocks and a minimum concentration of stress at the interface between the roof and the coal seat and the backfill array were ensured (when compared with the support of roadways by coal pillars of limited size). The stress-strain state of the lateral rocks in the coal array was determined by the geometric dimensions of the support structure and its rigidity.

The experimental studies in mine conditions indicate that displacements of the lateral rocks on the contour of the haul roadway when supported by compliant supports appeared until the compliant support was completely compressed (below l = 50 m) and stabilized behind the longwall. When supporting an extraction gallery by coal pillars of limited sizes, the convergence of the lateral rocks on the excavation contour behind the longwall (behind the mark l> 30 m) continued due to the destruction of the pillar.

Output. The stability of the lateral rocks and the operational condition of the local preparatory roadways while mining edge coal seams can be ensured by the presence of compliant supports located above the roadway or by stowing the mined-out space during the extraction, when the crushed rock fractions that are not uniform in size provide the maximum values of the compaction coefficient of the backfill array, which holds the roof rocks of the mined coal seam.

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