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Mathematical modeling of tight roof periodical falling

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Abstract. The paper proposes a method to determine of a coal seam roof falling step basing upon the analysis of stress and strain state of the rock mass area with mine workings formed as a result of coal preparatory and extraction operations. A boundary element method has been applied to define stress and strain state (SSS). Fissuring of enclosing rocks was modeled by means of transversal-isotropic medium. Dependence of destructed rocks zone height within the roof of a seam being mined upon the weakening of the rock mass due to its fissuring and mine working geometry has been determined. Effect of fissility on the periodical roof falling step has been studied. Changes in support loads in the process of stope advance have been determined. A scheme of partial backfilling of the worked out area has been proposed to maintain the support in its working order.

1 Introduction

Efficiency of coal extraction in powered stopes depends considerably upon the character of rock roof falling. Major problems arise while developing seams with overlaying heavy roofs. In such cases, roof falling may result in deformation of powered support units, their displacement, and setting support stands in rigid position. In this context, stope operation is often accompanied by coal sloughing and rock outbursts into the working area. There are also certain difficulties with the maintenance of development workings in the neighbourhood of the stope.

In practice, to localize the described rock pressure manifestations, arbitrarily left support facilities in a worked out area are often applied i.e. complete or partial backfilling or coal pillars [1], [2], [3]. Another approach is in preliminary softening of tight roof rocks by means of drill and blast operations. Those measures are labour-consuming, dangerous, and not always efficient. There is an alternative method decreasing the intensity of rock pressure manifestation – stage-by-stage caving of tight roof. It is applied mostly in longwalls (longer than 200 m). The method provides caving of the rock mass within specific areas of its outcrop by turning the stope line [4].

Dangerous rock pressure manifestations result in accidents and injuries. According to

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the data by the Ministry of Energy and Coal Mining, 68 rock fallings took place in Ukrainian mines within the period of 2015 – 2017, accounting for 35% of all the accident types; in 2017, total amount of injuries due to the factors made up 43%. It is also noted that mining and geological working conditions for Ukrainian miners are constantly deteriorating along with the deepening of mining operations where the amount of seams prone to intensive secondary roof caving increases.

There is a series of methods making it possible to evaluate with a certain degree of accuracy the risks of dangerous dynamic phenomena. In particular, there are papers containing recommendations for predicting roof caving and localizing their consequences. For instance, paper [5] proposes a method of roof control involving gradual turn of a longwall with roof caving by sections which length is not more than primary roof falling step; turning angle is connected with the direction of natural fissility of the rock mass. A value of the primary roof falling step was determined empirically.

To predict roof falling while coal seam extracting, it is important to evaluate the moment of the phenomenon beginning. Their periodicity is connected with the value of roof falling step. It is obvious that its determination should be based on the analysis of stress and strain state of rocks within potentially dangerous rock mass area.

Application of numerical methods has promoted considerably the process of rock mass modeling taking into consideration its structural heterogeneity and development of mining operations. Finite element method (FEM) is the one to be used more often. Thus, for instance, in paper [6] finite element method is aimed at studying rock mass-face support interaction; and papers [7], [8] consider stress and strain state of rock mass near a joint while coal seam mining. Paper [9] uses finite element method as the basis to model consequent movement of a longwall stope: going away from an installation chamber and approaching to a break-down chamber. The authors use an elastoplastic model of the medium; however, only plane or two-dimensional longwall cross-section is considered.

Paper [10] applied FEM in two-dimensional arrangement to study a change in stress and strain state of a border rock mass of a permanent anchor-supported inclined mine working in terms of its deepening.

As it is known, the highest intensity of rock pressure manifestation in mine workings is observed near the moving face. That is the result of changes in geometry of a worked out area as well as mutual effect of a stope and development mine workings. To take the facts into consideration, it is essential to consider spatial stress state of rock mass area nearby of conjugated workings. Moreover, to obtain solutions with the required accuracy within the framework of the mentioned FEM, the considered rock mass should be approximated with great number of elements that complicates the task significantly.

To represent mutual effect of a stope and development mine workings as well as such specificity of mining and geological conditions of Donbas mines as the presence of fissure system, it is required to develop more flexible algorithms making it possible to reduce problem order while preserving the advantages of a spatial model. Modification of a boundary element model [11] being quite efficient to model three-dimensional area of the rock mass containing thin coal seam helps do that.

The paper describes a method to determine roof falling step basing upon the analysis of spatial stress and strain state of rock mass area with mine workings formed as a result of flat coal seam preparatory and extraction operations. Numerical modeling is based upon one of the boundary element method modifications.

2 Methods and results of research

The paper has analyzed three-dimensional stress and strain state of the rock mass area containing a stope, development mine workings, coal pillar in front of the face, worked out area partially filled with the caved rocks, and areas of pillars or caved rocks bordering the development mine workings. Thus, computational area included all basic elements forming stress and strain state of roof rocks of the coal seam being mined.

The considered area of the rock mass was under the effect of the distributed load with γH components along the normal to the seam and $\lambda\gamma H$ along the strike and upwards. In this context, H is the seam occurrence depth, γ is the density of the enclosing rocks, and λ is the coefficient of lateral pressure. Computational algorithms are based on a method of boundary elements in the form of discontinuous displacements involving a seam element concept. As a result of the calculations, values of stresses and displacements both at the seam surface and in the analyzed area under the seam, i.e. in the roof, were determined.

A model of anisotropic medium was used to study the effect of system fissility of the enclosing rocks along with the model of elastic isotropic material. The applied method to model regular fissility is in its representation by means of transversal-isotropic medium [12]. In that case, it was assumed that along its fissures the medium has characteristics of undisturbed mass; and along the normal to the fissures plane, the medium characteristics are weakened. Similar representation does not preserve all the properties of the rock mass with regular fissility; however, it may be used to study the effects of anisotropy resulting from fissility.

The calculations involved modeling of averaged conditions of mine working construction being characteristic for Central Donbas mines. Properties of the enclosing rocks are: Young's modulus is $E = 3.5 \cdot 10^4$ MPa, Poison's ratio is $\nu = 0.17$, and volume weight is $\gamma = 2.5$ t/m^3 Coal properties are: $E_c = 3.2 \cdot 10^3$ MPa, $\nu_c = 0.13$. Occurrence depth and coal seam thickness are $H = 450$ m and $h = 1$ m. Characteristics of the caved rocks are: $E_{cr} = 3.0 \cdot 10^2$ MPa, $\nu_{cr} = 0.06$. Fissure characteristics are as follows: stiffness parameters along the fissures are $E_s = 3.5 \cdot 10^4$ MPa, $\nu_s = 0.17$; along the normal to fissures – $E_n = 3.5 \cdot 10^3$ MPa, $\nu_n = 0.02$. Distance between fissures is 0.1 m.

Boundaries of destructed rock zones in the mined seam roof are usually set according to one of the criteria. In terms of rock masses being in complex strain state, well-tested Parchevskiy-Shashenko, Balandin, and Hoek-Brown criteria [13], [14] are applied. The paper uses strength criterion by Balandin:

$$\sigma_{eq} = \frac{1}{2\psi}((1-\psi)(\sigma_1 + \sigma_2 + \sigma_3) +$$

$$+\sqrt{(1-\psi)^2(\sigma_1 + \sigma_2 + \sigma_3)^2 + 2\psi((\sigma_1 - \sigma_2)^2 + (\sigma_2 - \sigma_3)^2 + (\sigma_3 - \sigma_1)^2}) \le \sigma^n ,$$

where σ_1, σ_2, σ_3 are principal stresses; σ_{eq} is equivalent stress; $\psi = \sigma^t / \sigma^c$ is coefficient of rock brittleness; and σ^t and σ^c are tensile strength and compressive strength respectively. Value σ^n is used as ultimate stress; the value represents the stress calculated on the basis of elastic law corresponding to residual strength limit in three-link rock sample stress-strain diagram (Fig. 1).

Section of the diagram, characterizing the material breaking phase, corresponds to the values of elastic stresses exceeding σ^n. That is why the area where equivalent stresses σ_{eq} exceed σ^n is interpreted as the zone of destructed rocks. Stress lines $\sigma_{eq} = \sigma^n$ limit the destructed rocks zone in case when enclosing rocks are monolithic, i.e. there is no system fissility.

As a rule, geomechanical problems solving, rock mass fissility is taken into consideration by introducing a correction coefficient reducing calculated strength of the enclosing rocks – so-called coefficient of structural weakening. The latter is understood as rock mass strength - laboratory sample strength ratio. The coefficient is determined either empirically or on the basis of probabilistic and statistic approach [15]. The paper applies an analogue of the mentioned coefficient – rock weakening parameter $\tilde{k}$. It was determined as a ratio of maximum displacements of unsupported mine working contour within continuous and fissured rock masses $\tilde{k} = u \big/ u^{fis}$. Fissured rocks were modeled by means of transversal isotropic medium. Thus, while studying fissure-weakened rocks, value $\tilde{k}\sigma^n$ was used as the ultimate stress. Values $\tilde{k}$, obtained for fissure angles to a stratification plane 0 to 90º, were 0.068 to 0.46. The results are well consistent with the studies [15] where fissility is characterized by means of geological strength index depending upon the fissures size and condition of their surfaces.

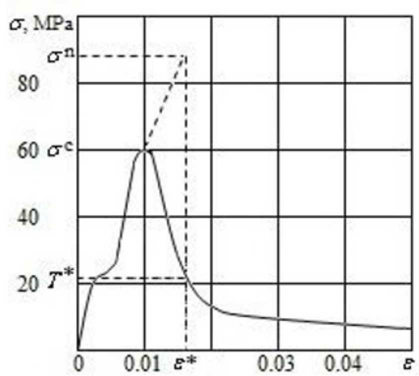

Fig.1. Clay shale compression diagram.

Maximum vertical extension of a rock destruction zone (destruction height) in the roof was determined for cases of continuous and fissured medium. In this context, values of geometrical parameters *d* (longwall length) and *l* (distance from a face entry to a longwall line) as well as weakening parameter $\tilde{k}$ were varied. As a result, the dependence of destruction height upon the indicated parameters for fissured rocks of the main roof was obtained [11]:

$$H_d^{fis} = 0.072 \cdot l^{0.75} \cdot d^{0.25} \big/ \tilde{k} \ .$$

Previous dependence has helped evaluate roof falling step in the process of face advance. It was assumed that roof fall took place when the zone of broken rocks above the seam covered the whole main roof along the height. Value of distance from a face entry to a face line, in terms of which falling height became equal to the main roof thickness, was interpreted as a supposed roof falling step.

The described approach was used to determine primary roof falling step under following conditions. Tight roof thickness H_0 was specified as equal to 30 m. Face advance process was modeled in terms of step-by-step changes in distance *l* from a face entry to a face line, $0 < l < l_0$. Calculations were performed for control marks within the stope being $d_1 = 72$ m, $d_2 = 152$ m, and $d_3 = 216$ m, if $l_0 = 80$ m.

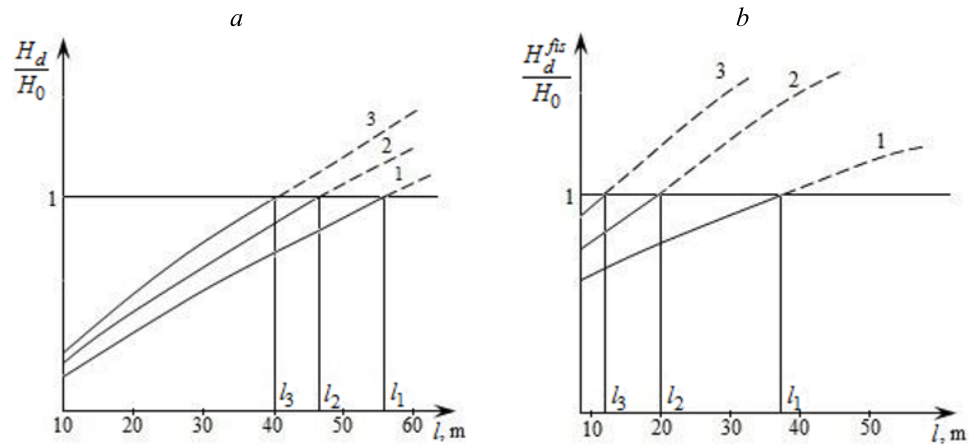

Fig. 2. Changes in destruction height while face advancing for longwall length values: *a* – unbroken medium; *b* – fissured medium; 1 – d_1 = 72 m; 2 – d_2 = 152 m; 3 – d_3 = 216 m.

Fig. 2a demonstrates curves of increase in destruction height without regard to the rock mass fissility. Destruction height H_d increases in the process of face advance until it reaches H_0 value. Horizontal line $H_d / H_0 = 1$ corresponds to the upper boundary of the main roof layer. Moments of primary falling correspond to the abscissas of intersections of curves 1, 2, and 3 with the mentioned line. They made up approximately $l_1 \approx 55$ m, $l_2 \approx 45$ m and $l_3 \approx 40$ m for values longwall length values d_1, d_2, and d_3 respectively. Fig. 2b shows similar results obtained considering the fact of fissility with fissure angle α to the stratification being 20°. In this context, destruction height H_p^{fis} rises faster, and value of primary falling step decreases respectively: $l_1 \approx 35$ m, $l_2 \approx 20$ m and $l_3 \approx 10$ m.

Thus, enclosing rocks fissility results in the reduction of primary roof falling step. The conclusion is consistent with the results of studies [16], where empiric dependences of falling step upon strength and geometry characteristics of the roof layers within 20 m thickness (with and without regard for fissility) are proposed. Data on 55 longwalls of thin gently sloping seams developed by Donbas mines were the basis for the studies. According to [16], ratio of roof falling step in fissured and non-fissured rocks varies from 0.35 up to 0.89. In the paper obtained value of the ratio as that being from 0.25 up to 0.65.

The obtained results were applied to determine support loads while stope advancing. It was assumed that the load acting on a powered support in a longwall is formed by the weight of that share of roof rocks which was within the destructed rocks zone. In case of tight roof hanging in the form of console, support load rises in the process of face advance up to another roof falling since the area of hanging share increases and destruction height grows.

Load, applied for powered support, was calculated as the weight of broken roof rocks related to the total area of canopies of all the support units. Unit sizes correspond to the support of 1KDD complex. Curves 1, 2, and 3 in Fig. 3 represent the load falling on the support unit for longwall length 152 m in three cases: fissured rocks with fissure angles to a stratification plane 20 and 70º and rocks without fissures. Horizontal line $F/F_{max} = 1$ corresponds to nominal value of the load on a support unit according to technical characteristics of 1KDD complex. The curves cross the indicated line at points with abscissas A, B, and C, i.e. value of support load exceeds the admissible one before the moment of the expected roof falling (points A_r, B_r and C_r). Moreover, if fissure angle is 20º, that happens with less than 20 m step.

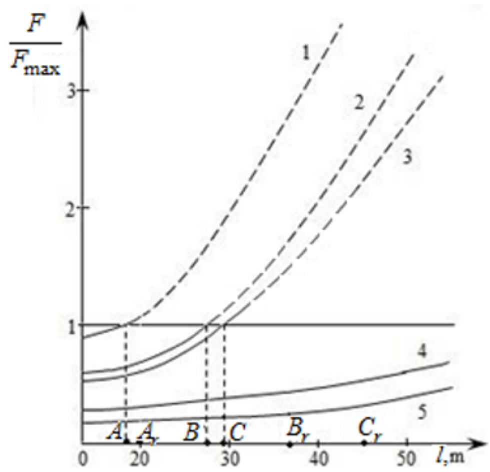

Fig. 3. Change in powered support load while face advancing. Complete caving: 1 – fissured rocks, 20º fissure angle; 2 – fissured rocks, 70º fissure angle; 3 – unbroken rocks. Partial backfilling with area width a = 5 m; 4 – distance between areas is b = 8 m; 5 – distance between areas is b = 4 m.

To avoid the described situation, it is recommended to backfill the worked out area by means of lines being perpendicular to a face line. Curves 4 and 5 in Fig. 3 demonstrate calculated values of support load for that variant of roof control, if backfill area width is $a = 5$ m and distance between areas is $b = 8$ m and $b = 4$ m respectively. In this context supported area is 35 and 50% of the total worked out area respectively.

Generalization of the research results has made it possible to obtain the dependence of support load upon a powered support unit while face advancing:

$$F = 0.66 \frac{\gamma H_d \, l}{l_n},$$

where H_d is destruction height, m; l is distance from a face entry to a stope or to the nearest roof falling point, m; l_n is length of support canopies, m; and γ is volume weight of the rocks, t/m^3.

3 Conclusions

Geometry of the destructed rock zones in a coal seam roof has been obtained on the basis of the solution of spatial problem of the elasticity theory for the rock mass area near development mine workings and stope junction.

Values of step of roof falling, while face advancing, have been determined both in terms of rock masses without fissures and in terms of enclosing rocks with regular fissility (fissure angle to the stratification is 20º). It has been defined that the fissility effect is in the decrease in primary roof falling step by 1.57 – 4 times for a different values of longwall length.

Values of support loads, while face advancing, have been calculated in terms of various geometry of the study area as well as fissure angles to the stratification. Partial backfilling of the worked out area has been proposed to provide accident-free support operation; backfilling parameters for stability maintaining have been recommended.

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