ментов комбинированной крепежной системы при поддержании выемочной выработки в неблагоприятных горно-геологических условиях для повышения ее устойчивости.

Методика. Основным методом исследования стал сравнительно-статистический анализ результатов проведенных вычислительных экспериментов, выполненных методом конечных элементов при нелинейной постановке задачи геомеханики. Моделирование обводнёности, трещиноватости и реологических характеристик породного массива было выполнено в виде поправочных коэффициентов и задания семейства кривых с использованием трех управляющих параметров. Неоднородность породного массива задавалась геометрически.

Результаты. Определены качественные и количественные показатели воздействия канатных анкеров на изменение величины интенсивности напряжений в сталеполимерных анкерах. Выявлена основная зона снижения интенсивности напряжений, примыкающая непосредственно к контуру выработки. Определена степень воздействия на комбинированную анкерную крепь неоднородности вмещающего породного массива и его нелинейных механических характеристик.

Научная новизна. Установлена зависимость изменения напряжено-деформированного состояния сталеполимерных анкеров от параметров установки канатных анкеров с ростом глубины заложения выработки в слабых вмещающих породах. Получена характеристика взаимодействия породных слоев и канатных анкеров над сводом выемочной выработки.

Практическая значимость. Выработаны параметры применения комбинированной анкерной крепи в качестве элемента крепежной системы в горных выработках, расположенных в зонах геологических нарушений, обводненных пород, при наличии тонких прослойков угля, кальцита или углистых аргиллитов в пределах глубины заделки анкеров. Определены пределы угла установки канатных анкеров, обеспечивающие повышение несущей способности крепежной системы сталеполимерных анкеров.

Ключевые слова: *анкер, канатный анкер, породный массив, выемочная выработка*

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CRITERION TO SELECT RATIONAL PARAMETERS OF SUPPORTS TO REDUCE EXPENDITURES CONNECTED WITH CONSTRUCTION AND MAINTENANCE OF DEVELOPMENT WORKING

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КРИТЕРІЙ ВИБОРУ РАЦІОНАЛЬНИХ ПАРАМЕТРІВ КРІПЛЕННЯ ДЛЯ ЗНИЖЕННЯ ВИТРАТ НА СПОРУДЖЕННЯ ТА ПІДТРИМАННЯ ПІДГОТОВЧОЇ ВИРОБКИ

Purpose. Substantiation of the parameter allowing predicting cost of mine working construction and maintenance under certain mining conditions taking into account a type of the mine working support, its parameters, repair size and cost as well as lifetime.

Methodology. Methods for generalization and comparative analysis, processing of underground investigation data, mathematical (numerical) modeling of geomechanical processes, and evaluation of economic efficiency have been applied.

Findings. Methodology to determine total costs for construction and maintenance of an extended mine working has been substantiated. A parameter of a mine working repairability has been proposed. The parameter considers intensity of rock contour displacements and cost of repair operations resulting from critical displacements of roof and floor. The repairability parameter has been used to substantiate the criterion allowing selecting rational support parameters to cut total costs for a mine working operation. A complex of field and numerical studies has been performed for Pivdennodonbaska No. 1 Mine. That made it possible to determine regularities of deformation processes within rock mass around mine working with frame support, and frame and roof bolting.

Originality. Innovative numerical model of geomechanical "mine working support-rock mass" system for Pivdennodonbaska No. 1 mine has been developed. The model allows studying deformation processes connected with floor rock heaving and its dinting in the context of different stages of a mine working operation. Regularities of "mine working support-mass" geomechanical system deformation have been determined making it possible to substantiate rational parameters for maintenance of mine workings to reduce repair cost at Pivdennodonbaska No. 1 mine. A parameter of a mine working repairability has been proposed taking into consideration the intensity of rock contour displacements and repair cost as well as the criterion helping select rational support parameters.

Practical value. Methodology to determine total cost of construction and maintenance of extended mine working can be used for designing rational support systems and selecting ways to improve stability of mine workings.

Keywords: *maintenance of mine working, rock heaving, rock dinting, rock displacement, zone of nonelastic deformation, frame and roof bolting*

Introduction. Mining is inevitably connected with mining deepening. In this context rock pressure grows considerably and operating conditions of underground objects deteriorate resulting in the increase in production cost of the mined raw material and deterioration of the competitiveness. One of the most important constituent parts of production cost of the coal mined in Ukraine is the one connected with maintenance and repair of mine workings. Increase in the stability of mine workings with the purpose of their reuse makes it possible to reduce the total number of mine workings being maintained, to raise concentration of mining operations; thus, it cuts coal mining costs.

Statement of the problem. Problems concerning the design of mine workings with minimum cost for their construction and further maintenance have always been one of the key issues while substantiating technical solutions. Their topicality has increased considerably along with mining deepening, deterioration of mine working operation conditions, and increase in support material cost.

Nowadays in the context of Ukrainian deep mines, the volume of mine workings being retimbered is as high as 50 % relative to the driven one and the volume of repaired mines is 1.7 times as high as the length of driven ones. In this regard more than 40 % of mine workings are repaired before their commissioning; 52 % of the operating mine workings are deformed. Deterioration of mine workings condition due to heaving makes up 45 % of the total volume of deformed ones [1].

It is often necessary to have repeated repairs to stabilize permanent mine workings; as for deep mines of the Donbas in terms of a pillar system of seam mining repetition the factor of repairs within preparatory mine workings is 2...3 and more.

The issue of adequate evaluation of mine working construction cost has been brought up many times by such prominent specialists in the field of mine working support as K. V. Koshelev, Yu. A. Petrenko and many others. In particular, they believe that it is far from being correct to design and construct mine working taking into account only initial costs when further costs for its maintenance are not calculated. It is proper to talk about the cost cut for mine working maintenance to be in operating condition during the whole its lifetime. Objective evaluation of the planned costs within the stage of mine working support design can be obtained by means of corresponding criterion use.

Analysis of recent research and publications. Volumes of repair and, consequently, cost of mine working maintenance are determined by its condition which can be described with the help of various parameters.

$\omega = \bar{L}/L$ stability index being the ratio of total length of $\bar{L}$ sites not requiring any repair works to L total length of a mine working is often used to evaluate the condition of extended mine working [2]. Mine working is either totally stable if $\omega = 1$ or completed destroyed if $\omega = 0$. ω parameter characterizes mine working stability within the certain period of mine working operation. Thus, it is not informative enough in the context of evaluation plan for construction and maintenance costs.

Another peculiarity of ω stability parameter is the fact that it does not consider the character of "support-rock mass" deformation system such as prevailing rock contour displacements and displacement intensity.

Similar concept concerning a mine working condition can be obtained through the parameter of mine operation conditions $\theta = R_c k_c/\gamma H$ where R_c is uniaxial rock compression strength, kc is a factor of structural reduction, γ is specific rock weight, H is mining depth [3]. The parameter helps estimate a degree of operation complexity within a mine working and, thus, make a substantiated decision concerning parameters of support, cost of support, and maintenance of mine working.

Another criterion to estimate conditions of a mine working is its total cost $\sum C$ consisting of capital expenditures connected with construction C_c and maintenance C_ω [3]. By doing so, it is necessary to work for maximum reduce of the cost rate

$$\sum C = C_c + C_\omega \to \min. \qquad (1)$$

Unfortunately, the parameter does not contain specific information on the condition of a mine working. It should be noted that the cost connected with maintenance of a mine working depends on capital cost value.

It is also quite a reasonable idea to estimate condition of a mine working on the value of rock contour U_t

displacements which are an aggregated factor of "mine support-rock mass" system behaviour [4]. The value of displacements helps determine support load, destruction area, and nature of deformations in time using them to specify parameters of support.

The performed analysis shows that none of the factors gives a chance for comprehensive and reliable estimation of a mine working condition in time and correlates it with total expenditures connected with its construction and further maintenance.

The objective of the article. The objective of the research whose results are stated in the paper is substantiation of the parameter making it possible to use a mine working condition to forecast cost of its construction and maintenance taking into consideration type of support and its parameters, repair cost, and lifetime of the mine working. To achieve the research objective it is required to perform a number of underground and numerical investigations concerning estimation of the mine working condition and selection of rational parameters of support in the context of specific mine workings. Final selection of a support should involve a criterion determining both efficient and economically expedient operation of the mine working.

Statement of the research basic material. Working area of western longwall 12 of C_{18} seam at Pivdennodonbaska No. 1 mine was chosen as the research object (Fig. 1) [5].

Extraction pillar is mined in return run. The length of the longwall is 230 m. The length of the working area is 1050 m. The thickness of the seam is 1.09 m. Occurrence mode of the deposit is complicated. Enclosing rocks have tendency to failure, heaving, and strength loss while soaking. Consequently, for conveyer passage support and its reuse while mining western longwall 12 of C_{18} seam, three floor rock liftings and retimbering of the mine working take place. Moreover, the first floor lifting is performed before the first longwall approach. Fig. 2 shows generalized results of measurements performed in the working area.

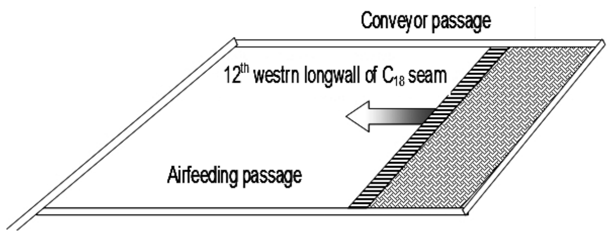


Fig. 1. Excerption of mining plan within western longwall 12 of C_{18} seam

Taking into account substantial volume of repair being performed, such support of conveyer passage comes expensive. Moreover, the technique aimed at mine working preservation is absolutely inappropriate as for the geomechanical factor.

As it is known, heaving in deep mines results from deformation processes involving the whole border area of rock mass. As [6, 7] explains, it is the indicator of great mining depth when displacements of the mine working contour and rock destruction are rather important.

Dinting results in greater heaving as well as decrease in stability of a mine working. Besides, it disbalances surrounding rock mass.

After dinting, rate of heaving experiences sensible increase. Mining practice confirms that after two or three dintings a mine working needs retimbering.

Consequences of the technological process taking place in mine workings are confirmed by the results of heaving analysis mentioned in [6]. Thus, it has been identified that the ratio of rock mass plastic loosening β in the neighbourhood of mine working experiences linear increase depending upon the increase in the number of dintings: $\beta = 1.12$ after dinting one; $\beta = 1.25$ after dinting two; and $\beta = 1.33$ after dinting three.

It has also been demonstrated that after heaving process in mine working, deformation processes affect the floor rock seriously duplicating size of destructed rocks.

Thus, to preserve stability of a mine working and reduce its cost it is required to apply such designs of sup-

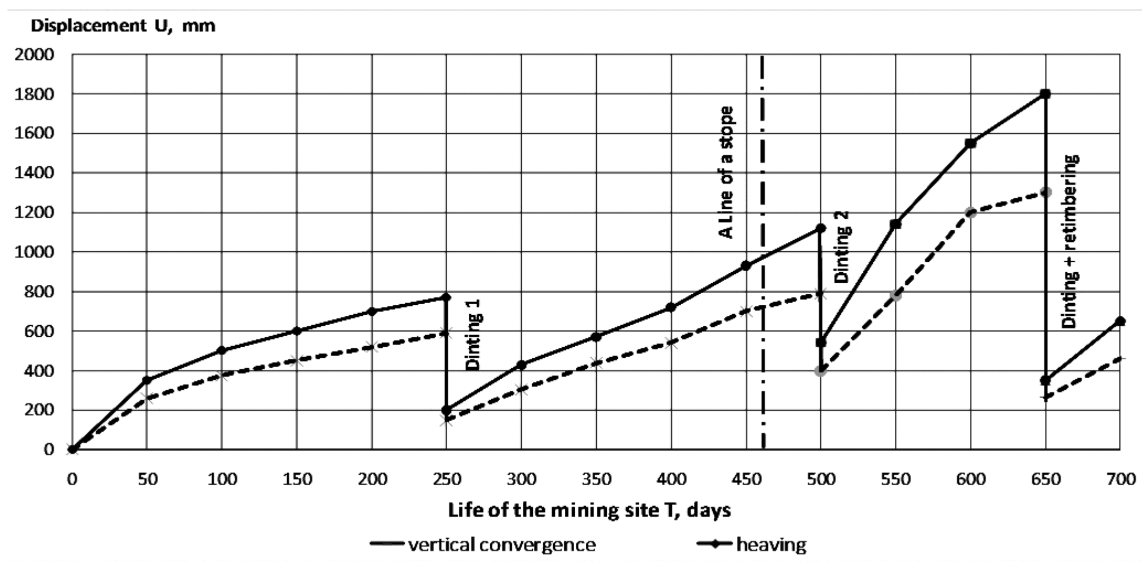


Fig. 2. Comprehensive schedule of rock mass border area deformation in conveyor passage

port which will scale down displacements of rock contour and will make it possible to perform no more than one dinting during the whole lifetime of a mine working.

In mine working of Pivdennodonbaska No. 1 mine in the context of support means being used currently, dependence of roof and floor rock displacements (Fig. 3) is determined by the following expression

$$U_a = 0.076(a \ln (T) - b)\theta^{(0.9-c)}, \qquad (2)$$

where T is the time of mine working site operation from the moment of its performance, months; a and b are coefficients depending on parameters of θ mining conditions; c is variable depending on λ coefficient of horizontal stress. For the conditions of deep coal mines: $a = -0.038\theta + 0.052$; $b = -0.1\theta + 0.105$; $c = -0.51\lambda^2 + 0.8\lambda + 0.47$; $\lambda \approx 1.0$.

Underground investigations have determined that the loss of mine working section is mostly the result of floor heaving; final contour displacements are determined by the following ratios: $U_f = 1.4\,U_a$ is for the floor; $U_r = 0.6U_a$ is for the roof.

To reduce repair volumes, the support structure should prevent contour displacements maximally.

Depending on the level of mine working permanence, the following is applied to achieve the objectives:

- closed support structures for long-life mine workings with complex support conditions;
- combined support structures on the basis of frame metallic support with bolts, backfilling of the area behind the support and other means to improve stability are used for basic preparatory mine workings;
- frame and roof bolting for areal mine workings with 1.5...2 years of operating period.

Cost of mine working maintenance within its whole $\sum C$ performance period taking into account cost of C_c as well as number of dintings and retimberings ($C_\omega = C_d + C_r$) can be determined by the expression (3)

$$\sum C = C_c + C_d + C_r = C_c + C_d^1 n + C_r^1 m, \qquad (3)$$

where n is the number of dintings within the whole period of the mine working operation; m is the number of retimberings within the whole period of the mine working operation; C_d^1 is the cost of one dinting; C_r^1 is the cost of one retimbering.

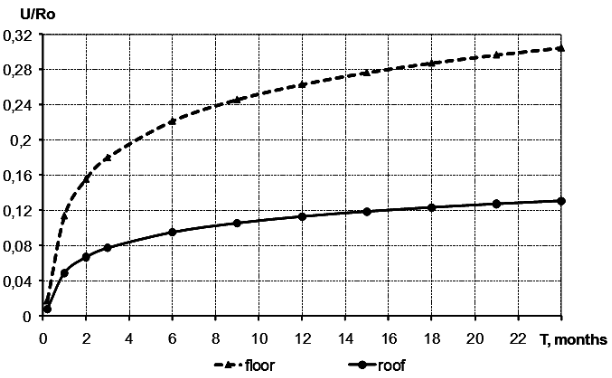


Fig. 3. Dependence of roof and floor rock contour displacements

The number of dintings and retimberings can be determined according to the value of rock contour displacements both within the whole period of its operation and the value of critical displacements resulting in the necessity to carry out repair works

$$n = U_f / U_f^*; \quad m = U_r / U_r^*. \qquad (4)$$

Critical displacements at which U_f^* dinting of heaved rocks and U_r^* retimbering are taken as those being equal to 0.6 m.

It is quite a problem to determine actual costs for repair works as they are included in various expenditure items. Thus, averaged estimated costs were calculated for their estimation at Pivdennodonbaska No. 1 mine involving the application of „Строительные Технологии – СМЕТА“ © “Computer Logic Group” program package.

While calculating, various operation volumes were considered with possible dinting range of 0.5...0.8 m. Reuse of frames of three-member arch yielding support as well as 70 % of ties were taken into account for retimbering operations. Volumes of operations for mine working widening are taken as those being equal to 20 % of its excavated cross section.

The results of the performed calculations have made it possible to determine that in terms of mine, the cost of its retimbering is 2.5 times as high as the cost of its dinting, i. e. $C_d^1 = 2.5C_r^1$. Taking $C_r^1 = C_{sr}^*$ as the cost of a single repair, total cost of mine working can be represented by the expression (5)

$$\begin{aligned} \sum C &= C_c + nC_{sr}^* + 2.5mC_{sr}^* = \\ &= C_c + C_{sr}^*(n + 2.5m) = C_c + P_r C_{sr}^*, \end{aligned} \qquad (5)$$

where $P_r = (n + 2.5m)$ is the parameter of mine working repairability.

Consideration of rock displacement intensity and time of mine working operation are the features of the approach to select mine working support.

Mine working operation will be the most efficient and expedient in case if no more than one dinting ($n \le 1.0$) is performed during its performance period and if roof displacements will not result in mine working retimbering. In this context roof rock displacements should not be more than the value of support yielding being equal for standard three-member structures of 300 mm (i. e. $m = 0.5$). Thus, mine working support and additional measures should ensure such roof and floor displacements under which maximum value of repairability parameter will be $P_r = (1.0 + 2.5 \cdot 0.5) \le 2.25$ that can be considered as the criterion of expediency for repeated mine working use considering both the condition of mine working and volumes of repair works within the whole period of its performance.

Let us consider the possibility to reduce the cost of supporting conveyer passage of western longwall 12 of C_{18} seam in order to preserve it for reuse. As it has been shown earlier, use of frame roof bolting with roof-bolt setting immediately after mine working performance is the efficient way to ensure mine working stability. It will allow preventing foliation of near-contour mass, large deformations, and rock displacements.

The first task in this context will be to determine such a number of roof bolts at which floor rock displacements will not necessitate rock ripping before the first longwall approaching. Operating experience shows that in this case heaving value within the conjunction point with a longwall should not be more than $U_f \leq 0.4$ m.

Let us perform numerical modeling of the behavior of "mine working support-mass" geomechanical system for the considered conditions of coal mining at Pivdennodonbaska No. 1 mine.

Initial calculation data are as follows. Initial stress field generating by the weight of the overlying rock for the specified depth is $\sigma_y = \gamma H = 10$ MPa. In this context $\gamma = 25$ kN/m^3 is specific rock weight, $H = 400$ m is mine working depth. Boundary conditions are specified within displacements – all the boundaries are fixed firmly. Boundary of mine workings is free from stresses. Table 1 shows physical and mechanical properties of enclosing rocks.

Several stages to determine stress and strain state of the mass area including conveyer passage and western longwall 12 of C_{18} seam were modeled.

Stage one of the calculations was aimed at adaptation of deformed rock mass model and calculation algorithm to real rock properties and conditions of mine working operation, i. e. "calibration" of both model and calculation procedure was performed [8]. Fig. 4 shows calculation scheme to solve the problem.

Initially, single mine working (conveyer passage) was modeled within undisturbed rock mass. Variants with floor rock heaving and their dinting were studied to evaluate mine working condition under specified conditions.

The next stage involved introduction of the system of standard steel and polymer 2.4 m long rock bolts into calculation scheme. Steel and polymer rock bolts were simulated by "Phase-2" facilities as steel rods fixed in the mass by means of polymer [9].

The applied research method makes it possible to determine displacements of mine working contour as well as broken rock area generating support load. It is possible to find the area on the basis of the adopted theory of strength.

Hoek-Brown strength criterion is the most tested and widely used in applied program packages. The criterion allows evaluating the level of rock breakage within the considered ambient point in terms of total effect of standard and shearing stresses taking into account both natural and technogenic disturbance of rock mass [10].

"Phase-2" software contains module implementing test of generalized Hoek-Brown criterion in the form of

$$\sigma_1 = \sigma_3 + \sigma_{ci}\left(m_b \frac{\sigma_3}{\sigma_{ci}} + s\right)^a, \qquad (6)$$

where σ_1 and σ_3 are maximum and minimum rock stresses, m_b is Hoek-Brown constant take for rock mass, s and a are constant values taking into account genesis of rocks and their conditions, σ_{ci} is ultimate uniaxial compression stress limit of rock mass in the intact condition.

Criterion ratio (6) is tested within each rock mass point; thus, breaking zone occurring as a result of stress

Table 1

Physical and mechanical properties of coal and enclosing rocks

Characteristic	Coal	Sandstone	Aleurolite	Argillite
Elasticity modulus (Young's modulus), MPa	9200	5700	2900	3000
Poison's ratio	0.26	0.25	0.25	0.25
Compressive strength, MPa	20	50	25	23

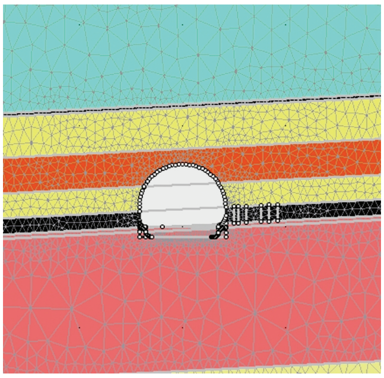


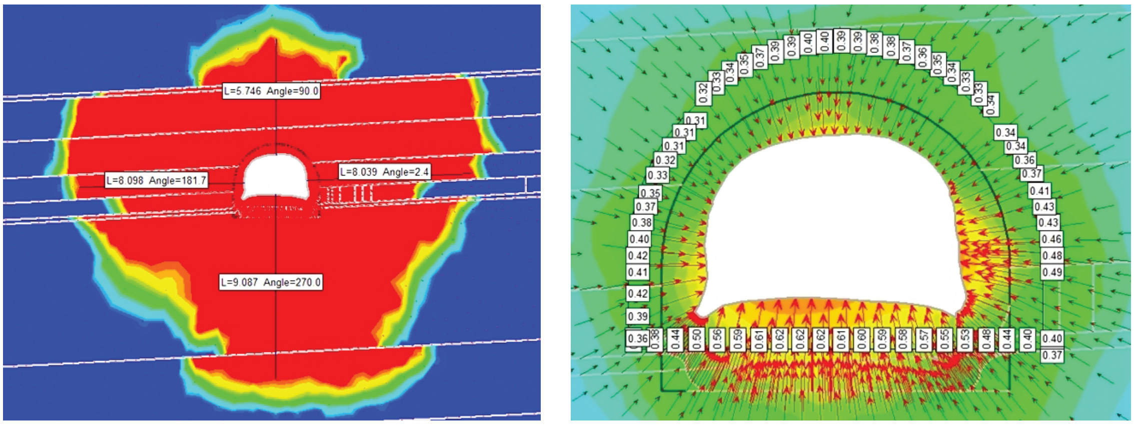
Fig. 4. Calculation scheme for undisturbed rock mass modeling

concentration within mine working areas is detected. Fig. 5, *a* shows broken rock zone according to Hoek-Brown criterion at the stage of mine working operation before the first longwall approaching. Fig. 5, *b* shows mine working contour displacements.

The obtained displacement values (0.69 m within the floor, 0.39 m within the roof, 1.02 m within the walls) correspond to the displacement value under real conditions of operation of conveyer passage of western longwall 12 of C_{18} seam. Coincidence of calculation and observed values is reached by means of correcting s, a, m constants being the constituent parts of the criterion (6) and taking into account both genesis and structure of rock mass on the basis of analysis of geological data and visual mine working inspection.

Floor heaving makes it impossible for the mine working to operate. That is why rock dinting is performed before longwall approaching. This process is modeled by means of "rock" extraction within a mine working along the height of floor raising. Fig. 6 shows rock displacements when floor dipping is performed. Sharp increase in broken rock area around mine working (by 15.5 m^2) proves negative effect of rock dipping.

Comparison of calculated contour displacements and real condition of a mine working shows that ambient deformation model within the frameworks of "Phase-2" program is calibrated; it can be used as the basis to predict the occurrence of mining pressure for other situations and support types. The stage before the first longwall approaching will involve use of roof bolts



a *b*

Fig. 5. Results of numerical modeling of single mine working beyond the area of stoping effect:
a – is broken rock zone (BRZ); b – are displacements within mine working contour

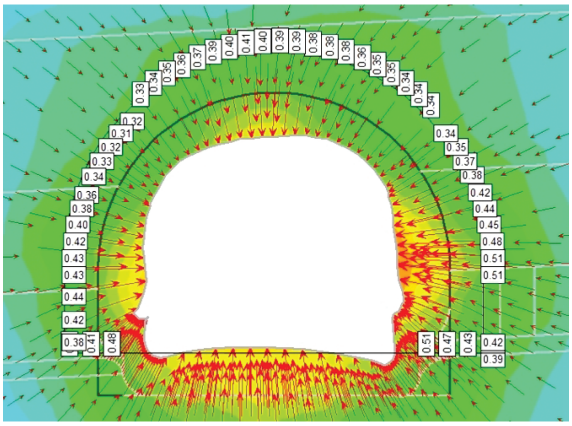


Fig. 6. Displacements within mine working contour after floor rock dinting

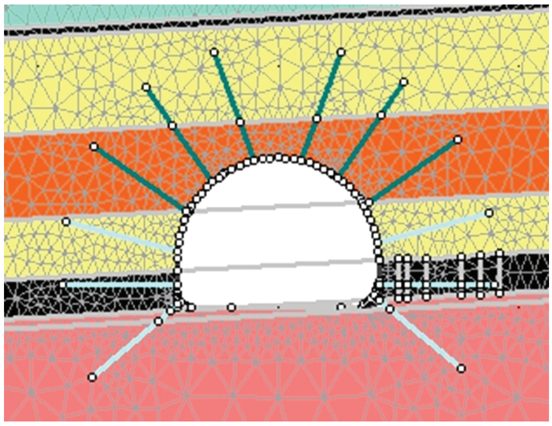
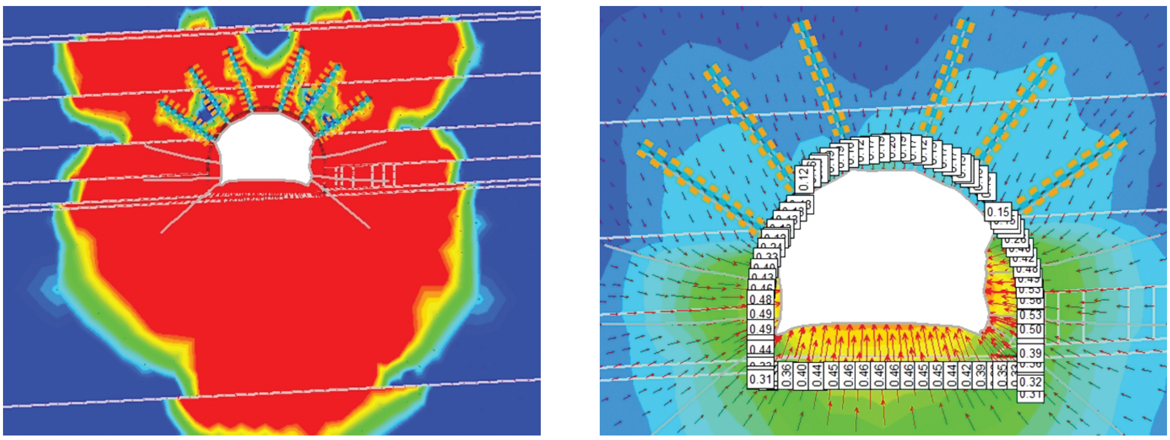


Fig. 7. Calculation scheme to model rock bolting systems

as the reinforcing components. Fig. 7 shows calculation scheme to solve the problem.

One should reach the result of floor rock displacement by the value being no more than 0.4 m, when rock dinting is not required, to preserve standard mine working condition while longwall driving. However, on the contrary to the considered conditions, during this stage it is necessary to take into account longwall effect resulting in intensification of rock displacement, especially within the floor. To consider bearing pressure of longwall face moving at the front, the so-called K_h = 1.3 surcharging coefficient is introduced; conditions along vertical



a *b*

Fig. 8. Results of numerical mine working modeling with rock bolting within stoping effect zones of:
a – zone of broken rocks; b – displacements within mine working contour

boundary of the studied area change. K_h coefficient value is substantiated from the consideration of mine field 3D model [10].

Fig. 8 shows modeling results while mounting 6 rock bolts within the mine working arch. Introduction of the rock bolts reduces development of displacements before longwall approaching: by 20 cm in the roof, by 8...10 cm in the walls, and by 19 cm in the floor (U_f = 50 cm). It is obvious that rock bolts mounting only within hill area allows cutting displacements within mine working roof considerably. To reduce heaving of floor rocks it is required to increase the zone of reinforced near-contour mass at the expense of rock bolts arrangements along the whole mine working contour.

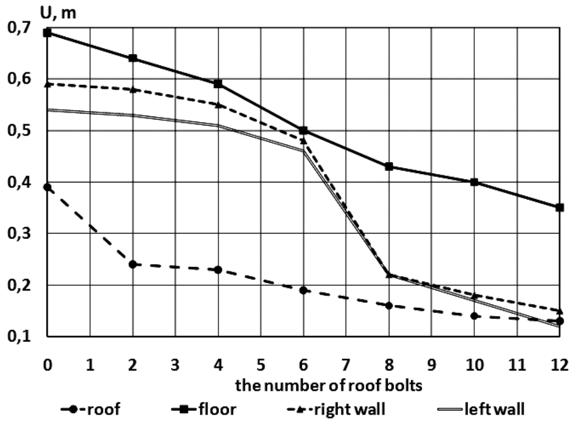
Fig. 9 represents the results of modeling rock bolting system with the different rock bolt number. Limitation of floor rock displacement to the required value is 40 cm with 10 rock bolts and 35 cm with 12 rock bolts. At that, if mine working section while driving is S = 13.4 m^2 and within conjunction with longwall in terms of fixation scheme adopted in mine being S = 7.7 m^2 (57 %), then in terms of frame roof bolting the section is as follows: S = = 9.9 m^2 (74 %) if there are 10 rock bolts, S = 10.2 m^2 (76 %) if there are 12 rock bolts. It helps provide sufficient cross section area of a mine working to operate within "longwall-entry" conjunction without labour-consuming and costly floor rock dinting.

However, to achieve the result it is required to increase capital cost for mounting rock bolting while driving.

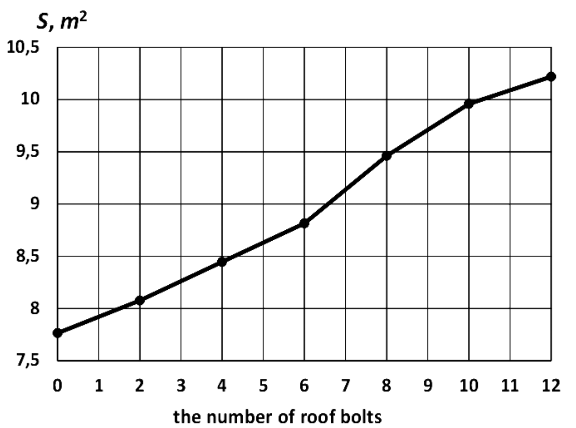
Fig. 10 shows the dependences of C_ω cost increase for repair works for various support structures taking into account C_c capital cost and period of mine working operation.

The calculations show that the capital costs increase while using frame roof bolting (12 rock bolts along the mine working perimeter) is compensated at the expense of considerable cost reduction for repair works; in 6 months of operation total cost of the proposed support will be lower than the applied frame structure.

An option of using costly circular support has been also considered. The support limits total rock contour displacement by U_f = 0.3 m yielding value; however, it is very expensive as well as labor-consuming while mounting (Table 2).



a



b

Fig. 9. Effect of rock bolts number mounted within a mine working:

a – upon the change in contour displacements; b – upon the change in cross section area

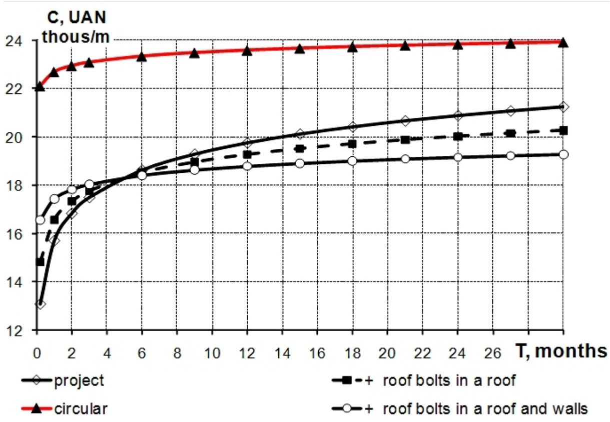


Fig. 10. Increase in repair cost for various support structures

Table 2

Cost of a mine working (thousand UAH/m) taking into consideration expenses connected with construction and maintenance depending on various support types

Support	Capital cost C_c	Maintenance cost C_ω	Integrated cost $\sum C$
According to the specification: KMP-A3/11.2 from SVP-27, 1.25 metallic	13.08	8.16	21.24
KMP-A3/11.2 with 6 roof bolts in the roof only	14.82	5.44	20.26
KMP-A3/11.2 with 12 roof bolts along the whole contour	16.26	2.11	19.28
Circular, metallic, 1.25 metallic	22.31	1.61	23.92

Fig. 11 shows changes in P_r repairability of a mine working depending on the period of its operation and support type.

As it has been demonstrated above, those support structures associated with extra measures are rational from the viewpoint of deformation processes development and considered as economically expedient if their repairability coefficient is no more than $P_r \le 2.25$. As it is seen from the data in Fig. 11, such supports are: circular support (P_r = 0.95; that is contour displacements will not result in the necessity of repair operations), and frame and roof bolting where roof bolts are mounted along the whole perimeter of the mine working (P_r = 1.3. Operation period will involve only one dinting).

For the frame support used currently in the mine, repairability coefficient is $P_r = 3.9$ when its operating period is $T = 2.5$ years, and integrated cost of its construction and maintenance $\sum C$ is almost UAH 21.24 thousand/m. Total cost of frame and roof bolting will be less owing to the decrease in repair volume even in the context of increase in capital cost (P_r = 1.3...2.5).

As for the high cost and labor intensity of closed support structure (C_c = UAH 22.31 thousand/m), it is expedient to construct it only when operation period of a mine working is long.

Conclusions and recommendations for future research. To substantiate rational support parameters to decrease total expenditures connected with construction of a mine working and its maintenance, repairability coefficient has been proposed involving intensity of rock contour displacements and cost of repair operations resulting from critical displacements of roof and floor. A number of field studies and numerical ones have been carried out for Pivdennodonbaska No. 1 mine. The studies helped determine regularities of deformation processes taking place within near-contour rock mass. The obtained results relying upon the specified criterion of a mine working repairability made it possible to substantiate rational support structure helping to reduce expenditures connected with repair operations. The proposed technique is applicable in the process of support system design and for the methods aimed at improvement of stability of mine workings.

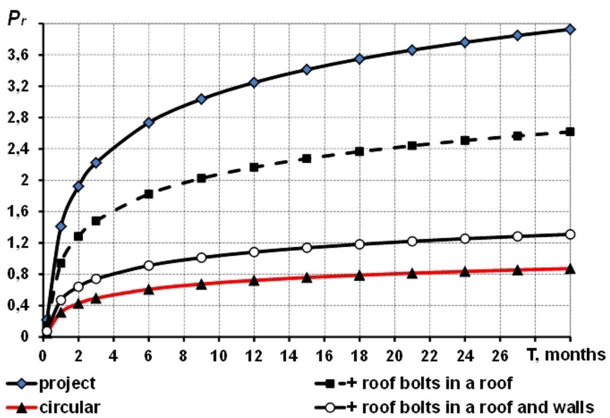


Fig. 11. Dependences of P_r mine working repairability parameter on period of its operation and support type

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Мета. Обґрунтування показника, що дозволяє за станом виробки прогнозувати вартість її спорудження та підтримки в конкретних гірничо-геологічних умовах з урахуванням типу й параметрів кріплення, обсягів і вартості ремонтних робіт, а також терміну експлуатації.

Методика. Застосовані методи узагальнення й порівняльного аналізу, обробки даних шахтних досліджень, математичного (чисельного) моделювання геомеханічних процесів, оцінки економічної ефективності.

Результати. Обґрунтована методика визначення сумарних витрат на спорудження та підтримку протяжної виробки. Запропоновано показник ремонтування виробки, що враховує інтенсивність зміщень породного контуру й вартість ремонтних робіт, викликаних критичними зміщеннями покрівлі та підошви. За показником ремонтування обґрунтовано критерій, що дозволяє вибрати раціональні параметри кріплення з метою зниження загальних витрат на експлуатацію виробки. Для умов шахти „Південнодонбаська № 1“ виконано комплекс натурних і чисельних досліджень, що дозволили встановити закономірності протікання деформаційних процесів у масиві порід навколо виробки з рамним і рамно-анкерним кріпленням.

Наукова новизна. Для умов шахти „Південнодонбаська № 1“ розроблена нова чисельна модель геомеханічної системи „кріплення виробки-масив“, що дозволяє вивчати деформаційні процеси, пов’язані зі здиманням порід підошви та їх підриванням, стосовно до різних етапів експлуатації виробки. Встановлено закономірності деформування даної геомеханічної системи, що дало можливість обґрунтувати раціональні параметри підтримки виробок для зниження витрат на ремонтні роботи в умовах шахти „Південнодонбаська № 1“.

Практична значимість. Методика визначення сумарних витрат на спорудження та підтримку протяжної виробки може бути використана при проектуванні раціональних систем кріплення й способів підвищення стійкості виробок шахт.

Ключові слова: *підтримка виробки, здимання порід, підривання підошви, зміщення порід, зона непружних деформацій, рамно-анкерне кріплення*

Цель. Обоснование показателя, позволяющего по состоянию выработки прогнозировать стоимость ее сооружения и поддержания в конкретных горно-геологических условиях с учетом типа и параметров крепи, объемов и стоимости ремонтных работ, а также срока эксплуатации.

Методика. Применены методы обобщения и сравнительного анализа, обработки данных шахтных исследований, математического (численного) моделирования геомеханических процессов, оценки экономической эффективности.

Результаты. Обоснована методика определения суммарных затрат на сооружение и поддержание протяженной выработки. Предложен показатель ремонтируемости выработки, учитывающий интенсивность смещений породного контура и стоимость ремонтных работ, вызванных критическими смещениями кровли и почвы. По показателю ремонтируемости обоснован критерий, позволяющий выбрать рациональные параметры крепи с целью снижения общих затрат на эксплуатацию выработки. Для условий шахты „Южнодонбасская № 1“ выполнен комплекс натурных и численных исследований, которые позволили установить закономерности протекания деформационных процессов в массиве пород вокруг выработки с рамной и рамно-анкерной крепью.

Научная новизна. Для условий шахты „Южнодонбасская № 1“ разработана новая численная модель геомеханической системы „крепь выработки-массив“, которая позволяет изучать деформационные процессы, связанные с пучением пород почвы и их подрывкой, применительно к различным этапам эксплуатации выработки. Установлены закономерности деформирования данной геомеханической системы, что позволило обосновать рациональные параметры поддержания выработок для снижения затрат на ремонтные работы в условиях шахты „Южнодонбасская № 1“.

Практическая значимость. Методика определения суммарных затрат на сооружение и поддержание протяженной выработки может быть использована при проектировании рациональных систем крепи и способов повышения устойчивости выработок шахт.

Ключевые слова: *поддержание выработки, пучение пород, подрывка почвы, смещения пород, зона неупругих деформаций, рамно-анкерная крепь*

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