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Substantiating the optimization solutions for the mine working fastening system interaction with the enclosing rock mass

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Abstract

Purpose. Determination of the rational interaction modes between the fastening system and the extraction working enclosing mass in the zone of stope operations influence.

Methods. An algorithm for searching for optimal solutions for the interaction modes between the fastening system and the coal-bearing mass has been substantiated. The deformation-strength characteristics of the fastening system elements have been agreed. The design parameters of the support elements have been optimized according to the criterion of their equal strength. According to the optimal parameters, a methodology for calculating the function that describes the rational deformation-strength characteristic of the fastening system, depending on the mining-geological conditions, has been developed and substantiated.

Findings. Computational experiments have been conducted to determine the rock mass deformation-strength characteristic. Based on the normative documents, the sizes of the natural equilibrium arch have been calculated. The adequacy of methodical principles for minimizing the load on the fastening system has been proved. The patterns for the influence of geomechanical factors on the choice of optimal parameters of the fastening system deformation-strength characteristics have been determined. A methodology for calculating the rational parameters of the fastening system and its constituent elements has been obtained.

Originality. Combined studies of minimizing the load on the fastening system have been conducted. The patterns for the influence of geomechanical factors on the choice of load-bearing capacity and the yielding property value of the fastening system have been determined. Regression equations have been obtained for calculating the fastening system optimal parameters with a geomechanical index of working conditions. This enables implementation of a unified strategy for resource-saving improvement in fastening systems.

Practical implications. A methodology has been developed for obtaining the weakening mass deformation-strength characteristic, depending on the depth of mine working location, the texture of the rocks in the coal-overlaying formation and its strength properties. The applicability of the methodology for the implementation of a unified strategy of resource-saving improvement of the mine working fastening systems for the Western Donbas mines has been proved.

Keywords: *rock mass, extraction working, optimization, methodology, fastening system*

1. Algorithm for determining the rational parameters of the fastening systems

Currently, one of the main scientific-technical problems of the energy supply security in Ukraine is the creation and implementation of a modern integrated technology for coal mining in difficult mining-geological conditions of mining thin and very thin coal seams [1]-[6]. The solution to this problem is in the search for optimization parameters in the rock pressure interaction with a set of innovative fastening and security systems [7]-[8]. During the research, modern algorithms of analytical and numerical models, original experimental studies are used to achieve the mine working stability by controlling the geomechanics of the "mass – strengthened rocks – support" system interaction [9]-[11].

In the previous works [12]-[15], the processes of loading the fastening systems have been substantiated and studied, as well as the mutual influence of the deformation-strength characteristics of the enclosing rock mass and combined fastening structures has been schematically represented. The works [16]-[19] study the issues of developing an algorithm for searching for optimal interaction modes between the fastening system and the strengthened rock mass; an algorithm has been developed that includes the following key positions:

– the need to form a minimum load on the combined support, depending on the mining-geological conditions for the mine working maintenance;

– analysis and selection of appropriate deformation-strength characteristics (load, displacement) of all load-bearing elements;

– ensuring the minimization of each fastening element material consumption when the condition of the entire structure equal strength is satisfied.

The solution of this hard complex problem is possible by means of computational experiment methods, since the princi-

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ple of joint deformation of the weakening mass, rocks in the natural equilibrium arch and in the fastening system has already been used in the calculation by the finite element method [20], [21]. At the same time, the influence of the fastening system characteristic $P(u)$ on the mass deformation-strength characteristic, as well as the influence of $P(u)$ on the behavior of rocks in the natural equilibrium arch are assessed.

At the previous stages of the research [22]-[28] a methodology has been developed for obtaining the patterns for loading and displacement depending on the index H/R (where H is the depth of the mine working location, R is the integral strength characteristic; $R = 0.5\ (R^r + R^b)$ where $R^{r,b}$ is the calculated resistance of roof and bottom rocks). This made it possible to specify the conditions for optimizing the fastening system deformation-strength characteristics with minimizing the load.

Thus, when substantiating the optimization calculations for the mine working fastening system interaction with the enclosing rock mass, testing of the proposed methodology for the reliability of its recommendations is a required step.

2. Methodology

When developing any new methodology for calculating the interaction parameters of the mine working fastening system and the rock mass containing it, and even more so when substantiating the optimization solutions for this interaction, testing of the proposed methodology recommendations is a required step. Thus, the methodology recommendations should be tested for adequacy and reliability in terms of consistency with existing ideas, analytical studies and the accumulated experience of mine observations. Therefore, a set of test calculations, conditionally divided into two stages, has been performed:

– the first stage assesses the adequacy of a new technique in the technology of conducting a computational experiment to calculate the deformation-strength characteristic of a rock mass weakening around an underground mine working;

– the second stage is designed to analyze the degree of reliability of the calculation results according to the optimization methodology as a whole in relation to the existing methodical developments that form the basis of the relevant normative documents.

A new technique in the technology of conducting a computational experiment provides for the simulation of various displacement values of border rocks by introducing an easily deformable rock layer with adjustable deformation properties into the direct roof [24]-[27]. The reasons for this decision were substantiated earlier in the work [28]. Their essence is in the possibility of obtaining a relationship between the load q_1 on the fastening system and the weakening rock mass, depending on the value of "permissible" displacements U of the mine working contour in a wide range of their change U = 300-1100 mm. On geomechanical models, such a displacement value of the mine working contour can be achieved without introducing a "virtual" rock layer, for example, by using a physical model of the so-called "complete" deformation diagram of rocks and fastening materials. However, here difficulties arise both in the methodical order and in the technical possibilities of conducting a computational experiment. The methodical difficulties are as follows.

The function $q_1(u)$ is determined for a fixed value of the geomechanical index H/R and the type of the generalized coal-bearing mass texture. The latter can be modeled unchanged, and fixing a stable value of H/R in the course of a multivariate computational experiment is far from always possible. The fact is that the function $q_1(u)$ should be constructed in a wide range of changes in the yielding property U = 300-1100 mm of the fastening structure. The value U itself is determined after performing a specific calculation of the geomechanical system SSS. Here, given the specified mechanical characteristics of all the constituent elements, it is necessary to find such a depth value H (in the entire interval of its increment in the process of step-by-step calculations), which would correspond to a fixed index H/R value. This, in addition to the high complexity of the performed operation, is not always possible: often either the depth H or the displacement value U are outside the studied intervals of their change. Therefore, it is necessary to vary the value R, which means to perform additional computational experiments and all in order to obtain one point of the desired function $q_1(u)$.

There is another methodical difficulty – the uncertainty of the final result of calculating the displacements U for a fixed R. After all, the use of a complete deformation diagram of the geomechanical system materials implies a set of appropriate mechanical characteristics (for example, for a rock – deformation and decay moduli, ultimate compressive strength, residual strength at the stage of loosening). By varying these characteristics at a fixed U, it is possible to obtain a different displacement value U of the mine working contour [28]-[30].

The problems of the technical procedure for conducting a computational experiment using a physical model of a "complete deformation diagram" of rocks and fastening materials are well known [31]-[34]. In addition to a large number of incoming mechanical characteristics, the computational process "failures" often occur due to inadequate transition of increments at the so-called critical points (limiting state, weakening and loosening stage boundaries), as well as due to simple insufficiency of the computing resource. This situation often arises when calculating relatively simple geomechanical models, to which the studied geomechanical system cannot be assigned: stratified mass with its texture disturbances, zones of uncontrolled collapse and hinged-block displacement in the mined-out space, frame support with a combined roof-bolting system, multi-element security structure [35], [36]. Therefore, the combination of the above reasons has led to the abandonment of using the physical model of the "complete deformation diagram" of the materials of the geomechanical system elements, replacing it with an elastic-plastic physical model with a "bilinear" deformation diagram of materials. It uses an additional yielding property simulator in the form of an easily deformable immediate roof rock layer.

Nevertheless, such a technique for conducting a computational experiment needs to be tested for the adequacy and reliability of the results obtained [37]-[40]. The following methodical substantiation is aimed at its implementation. Its essence is in the comparison of mass displacement U_{xy} curves on a plane XY using relatively simple geomechanical models, as well as displacements U of the mine working contour using the above two physical models. The relative geomechanical system simplicity is conditioned by the desire to obtain a reliable calculation result when using a physical model of the complete deformation diagram of materials of the constituent elements. The calculation results are qualitatively and quantitatively compared with the SSS determined for the same geomechanical system, but using a physical model of elastic-plastic deformation of materials in the presence of an additional yielding property simulator.

2. Results and discussion

When implementing this algorithm, a model of a single mine working has been chosen, fastened only with a frame support of the KMP-AZ series and located in a coal-bearing soft rock mass. Such a model (in order to make an objective comparison) was studied in the work [28] and, in terms of its parameters, it is quite suitable for the comparative base function. Figure 1 shows the curves of full displacements U_{xy}, indicating cardinal differences in displacements (in the surrounding mass and in the mine working contour) at different depths of its location. Thus, at $H = 200$ m, the displacements in the area of the mine working arch are only $U_{xy} = 35$-38 mm, and at $H = 600$ m, these displacements increase to 2000 mm or more with a maximum value of $U_{xy} = 2469$ mm. As can be seen, if to consider the complete deformation diagram of rocks and fastening materials, quite real figures can be obtained of the rock pressure manifestations and loss of in-seam working operational state at a large location depth and a small strength of the surrounding rocks. Such geomechanical situations are observed in the practice of mining the coal seams in the Western Donbas.

(a) (b)

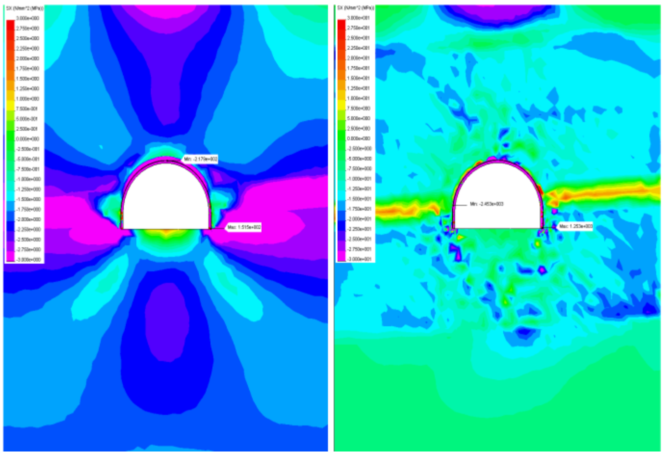

Figure 1. Curves of full displacements U_{xy} in the geomechanical model of a single mine working in a stratified rock mass with low strength at a depth H of its location: (a) H = 200 m; (b) H = 600 m

To confirm the results obtained in the work [28], the author has performed similar SSS calculations for completely identical geomechanical conditions: the initial parameters for performing the computational experiment are detailed in the work.

However, the basis for recalculating the specified geomechanical model SSS is the development of the results obtained in a convenient form for subsequent analysis of the level of differences in the patterns of changes in the mine working contour displacements with an increase in the depth H of its location (with a constant R, variation of H is identical to the variation of H/R) for two physical models: a complete material deformation diagram and an elastic-plastic model with a yielding property simulator. The methodology for this comparison is presented below.

First of all, it should be noted the minimum discrepancies between the distribution parameters of the curve of full displacements U_{xy} according to the data in the work [28] (Fig. 1) and the performed calculations (Fig. 2):

– at $H = 200$ m, the arch contour full displacements have been obtained in almost the same range of $U_{xy} = 35$-38 mm;

– at $H = 600$ m, the curves U_{xy} are qualitatively very similar to each other, and quantitatively, with the same location of the maxima U_{xy}, the obtained value of $(U_{xy})_{\max} = 2446$ mm differs from that in the work [28] by only 0.9%.

(a) (b)

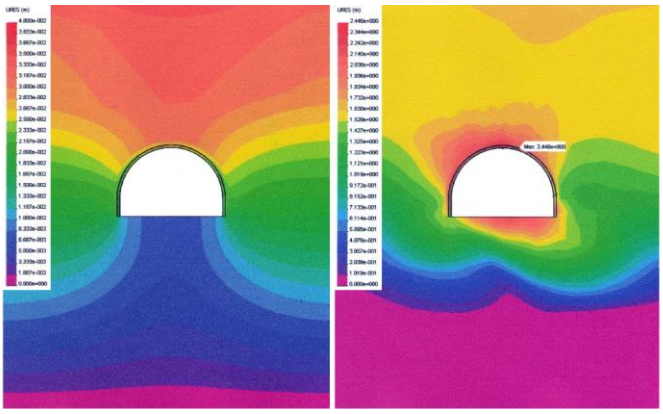

Figure 2. Curves of full displacements U_{xy} in the geomechanical model of a single mine working in a stratified rock mass with low strength according to the calculations performed by the author at a depth of its location: (a) H = 200 m; (b) H = 600 m

The given data prove the correctness of constructing a geomechanical model and performing a computational experiment, which enables further objective and more detailed consideration of the development of the mine working contour displacements with an increase in the depth of its location. In a methodical plan, in order to increase the objectivity of the analysis, eight points have been identified on the mine working contour, corresponding to certain nodes of the finite element mesh:

– node No. 13329 – arch keystone;

– nodes No. 13327 and No. 13362 – the arch springs from the side of the dip and rise, respectively;

– nodes No. 1120 and No. 370 – bearings of the frame prop stays from the side of the coal seam dip and rise, respectively;

– node No. 536 – the center of the mine working width along its bottom;

– nodes No. 541 and No. 531 – points along the mine working bottom at a distance of 1/6 of its width from the bearings of the prop stays from the side of the dip and rise, respectively.

The vertical U_y and horizontal U_x displacements of the indicated points are shown in the graphs of Figure 3 depen-ding on the relative time of step-by-step calculation (from 0 to conventional 1); this horizontal scale corresponds to the mine working depth from 0 to 600 m. Here are the depen-dences of the development of both vertical and horizontal displacements in the contour, not only in the roof, but also in the mine working bottom; their uniformity enables, in our opinion, a more detailed and objective comparative analysis to assess the reliability and adequacy of the proposed methodology for determining the weakening mass deformation-strength characteristic. The whole family of graphs has a common peculiarity regarding the depth of the mine working location: an insignificant increase in displacements in the area of predominantly elastic mass deformations and their intensive development (starting from some depths H) with a high growth gradient in the area of the super-limiting state (the stage of rock loosening). This common trend does not contradict the existing ideas about the rock pressure manifestations with the complication of mining-geological conditions for maintaining the mine working.

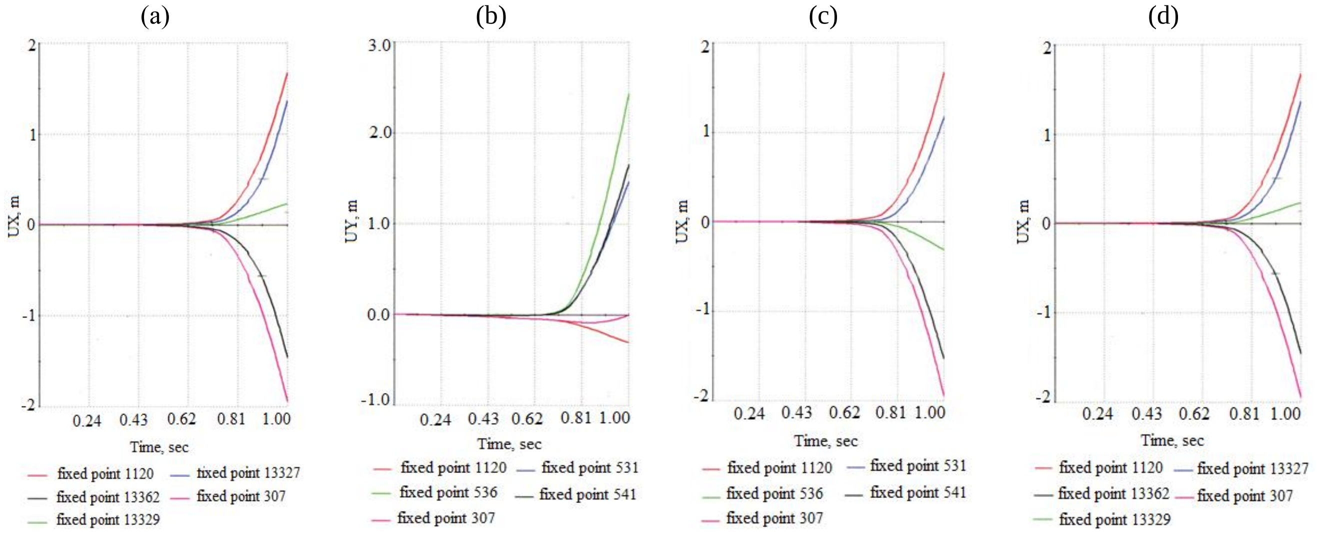

Figure 3. Change in vertical (a), (b) and horizontal (c), (d) displacements at fixed points of the mine working contour in its roof (a), (c) and bottom (b), (d) according to the SSS calculation data using a physical model of the complete rock and material deformation diagram of fastening elements

All the above studies were intended to create a comparative basis for assessing the adequacy and reliability of the proposed methodology for determining the deformation-strength characteristic of a weakening mass $q_1(u)$. This methodology is distinguished by increased reliability of calculations due to a simplified elastic-plastic physical model and the use of an artificial yielding property simulator.

Therefore, of great interest are the dependences of the increase in H with increasing relative time parameter of the step-by-step calculation from 0 to a conventional unit, which corresponds to the interval H = 0-600 m. For the convenience of comparative analysis, the displacement growth U graphs are combined in one figure for the same measuring points in the springs and the arch keystone (Fig. 4).

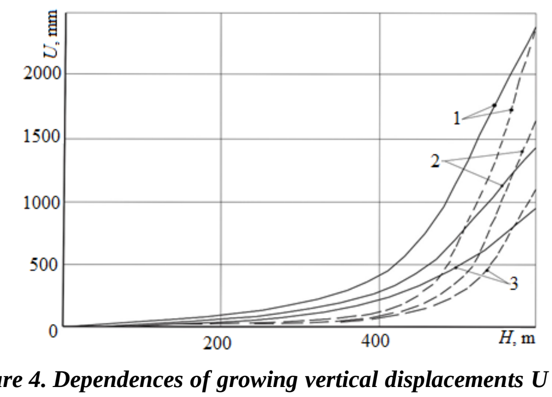

Figure 4. Dependences of growing vertical displacements U of the mine working arch with an increase in the depth H of its location when using a physical model of the complete material deformation diagram (– – –) and according to the proposed methodology (——): 1 – arch keystone lowering; 2, 3 – displacements of the arch springs from the side of the coal seam dip and rise, respectively

Here, the arch keystone displacements according to the proposed methodology are brought into agreement with the value U of the comparative base by selecting the deformation modulus of the yielding property simulator.

The first main parameter of the comparative assessment is the dependence of the development of the mine working contour displacements with an increase in the depth H of its location.

Here, the main attention is paid to vertical displacements $U = U_y$ and the vertical load q_1 determination, and, since U_y increases with increasing H (Fig. 3), and then, at a fixed R, the dependences $U(H/R)$ and, finally, the desired function $q_1(u)$ can be obtained.

Thus, initially the two extreme points (H = 0 m and H = 600 m) of the dependences are the same for the displacements in the arch keystone, and we are interested in three positions of the comparative assessment, which are ranked according to the degree of importance:

– the correspondence of the compared functions $U(H)$ to each other within the interval of change $0 < H < 600$ m, since a priori the initial and terminal points will be the same;

– the degree of differences in functions $U(H)$ for both arch springs in the entire interval of $0 < H < 600$ m, which is necessary to assess the compliance of the general situation with the vertical displacements of the mine working arch;

– the degree of differences in functions $U(H)$ for horizontal displacements in the springs and the arch keystones for a general understanding of the adequacy level of the compared geomechanical models to each other (Fig. 5).

A pair of graphs *1* in Figure 4 show the abundance of trends in the growth of arch keystone displacements with an increase in H, which is quite natural and does not require explanation. Qualitatively, they are similar to each other, and quantitatively, there are differences in the depth range of $0 < H < 500$ m. For example, at H = 200 m, the difference is up to 2.4 times, but in absolute terms – only 53 mm; at H = 400 m the differences reach 3.5 times with an absolute difference of 295 mm; at H = 500 m, the deviations are reduced to 1.7 times, but the absolute difference increases to 490 mm; with a further increase in H, the relative and absolute difference in the dependences decrease. Similar trends are observed for function $U(H)$ deviations in both arch springs, but with smaller values: relative – 1.14-3.07 times, absolute – 145-195 mm.

Thus, with a high qualitative similarity degree of the graphs, their quantitative differences are quite significant. This discrepancy is conditioned by the main reason – the real complete rock deformation diagram is usually represented by three linear dependences: elastic deformation areas, a descending branch of the rock weakening and loosening process with a constant value of "residual" strength [41], [42].

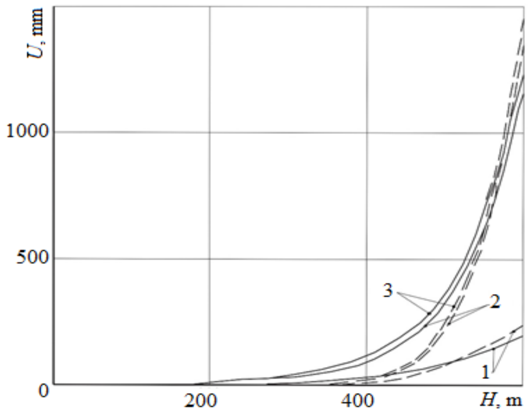

***Figure 5.** Dependences of growing horizontal displacements U of the mine working arch with an increase in the depth H of its location when using a physical model of the complete material deformation diagram (– – –) and according to the proposed methodology (——): 1 – displacement of the arch springs; 2, 3 – displacements of the arch springs from the side of the coal seam dip and rise*

The points of one straight line transition (in the model of the complete deformation diagram) to another contribute to a sharp change in the displacement growth gradient U. In fact, numerous experimental studies [43] of complete deformation diagrams show that the transition of rocks from one state to another is smooth: from practically elastic to elastic-plastic with probable plastic flow (in the long section of the diagram), especially weak argillites and siltstones; from limiting to superlimiting with weakening of the rock and then to the stage of loosening, not with constant values of residual strength, but gradually decreasing with increasing deformations.

It is not yet possible to reflect real rock deformation diagram in detail, especially in complex problems of geomechanics, and the three-line approximation leads to certain distortions in the form of the function $U(H)$. In fact, it is more flattened with a smoother growth gradient change, as it has been obtained on the graphs $U(H)$ calculated by the proposed methodology. In addition, extensive experimental measurements of the mine working roof displacements at different depths of their location, which are summarized in normative documents, for example, in [44]-[47], confirm a smoother change in the function $U(H)$.

Nevertheless, the first main result of the comparative analysis is in proving the sufficient adequacy of the proposed methodology for determining the patterns of development of vertical displacements in the mine working arch contour with an increase in the depth of its location. This conclusion is also confirmed by the analysis of tendencies in the growth of horizontal displacements in the mine working arch (Fig. 5).

The second main parameter of the comparative assessment is the deformation-strength characteristic $q_1(u)$ of the weakening mass itself, and the methodical technique of conducting the computational experiments is used here [40]. When performing computational experiments using a physical model of the complete deformation diagram of rocks and materials of fastening elements, as well as by the proposed methodology, the geomechanical index H/R is the same value of $H/R = 120$ m/MPa. On the basis of this, as well as when using the same type of the coal-bearing mass texture, it is possible to ensure the objectivity of the comparison. Further, in each of the two computational experiments performed, it is recommended to carry out five calculations for a different displacement value U of the mine working arch keystone; the maximum value of $U = 2420$ mm has already been obtained at $H = 600$ m. To calculate the remaining five different values u_j ($j = 1, 2, 3, 4$), the following methodical technique is used. Restriction of displacements $u_j < u_{\max}$ is achieved by tightening the fastening structure operating mode by an artificial increase in the yield limit of the frame support steel by four different times: 3, 10, 30 and 100. In this case, the calculated values of u_j are smaller than $U_{\max}$, and we have five points $(q_1)_j$, u_j for constructing the function $q_1(u)$ of the weakening mass deformation-strength characteristic; they are determined according to the recommended methodology. The obtained piecewise-linear graphs are compared with each other and the degree of their correspondence to each other is determined.

The presented algorithm for testing the adequacy and partially the reliability of the weakening mass deformation-strength characteristic $q_1(u)$ is implemented and illustrated in Figure 6. Qualitatively, the functions $q_1(u)$ are very similar to each other, and quantitatively, there are deviations of different signs in the range from -12.0% to +24.1%, which can be considered a completely satisfactory result.

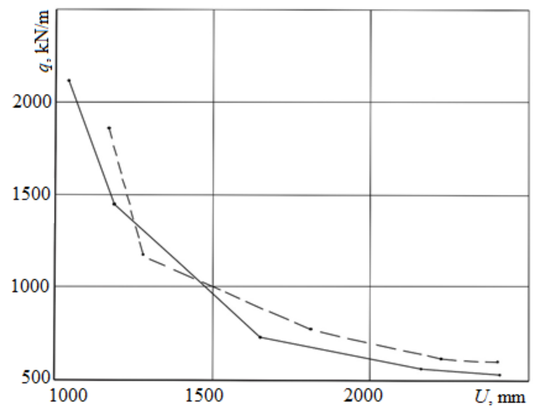

***Figure 6.** To the analysis of the methodology adequacy (– – –) for determining the weakening mass deformation-strength characteristic q₁(u) when compared with a physical model (——) of the complete deformation diagram of the materials of the geomechanical system elements*

Summing up the first part of testing in terms of the adequacy degree when determining the weakening mass $q_1(u)$ deformation-strength characteristic according to the developed methodology, it can be concluded that its use in solving the problem of optimizing the interaction modes between the fastening system and the rock mass is acceptable.

The next stage of research is the reliability analysis of the methodology for optimizing the interaction modes between the fastening system and the surrounding mass in order to determine the patterns of the relationship between the fastening system optimal parameters and geomechanical factors.

Thus, an accessible methodology for calculating the fastening system deformation-strength characteristic has been developed, depending on the mining-geological conditions for maintaining the mine workings.

4. Conclusions

An algorithm for searching for rational interaction modes of the fastening system and the coal-bearing mass surrounding the extraction working has been substantiated; the algorithm involves performing a number of studies that are closely related to each other by the general interaction process parameters:

– the minimum possible load formation in specific mining-geological conditions for the mine working maintenance;

– coordination between the deformation-strength characteristics of the elements included in the fastening system;

– optimization of design parameters of fastening elements according to the criterion of their equal strength.

The methodical principles of minimizing the load on the fastening system of reused extraction workings have been determined. These principles are based on the use of a set of studies using the FEM (determining the deformation-strength characteristic) and recommendations of normative documents for calculating the natural equilibrium arch sizes (deformation-strength characteristic of rocks in the arch).

Based on the determined principles, a methodology has been developed for obtaining the deformation-strength characteristic of a weakening mass, depending on the main influencing geomechanical factors: the mine working depth, the texture of rocks in the coal-overlaying formation and its strength properties.

When determining the deformation-strength characteristic of rocks in the natural equilibrium arch, the methodical provisions of the normative documents have been transformed and supplemented by taking into account such a geomechanical phenomenon as limiting the arch sizes due to the reaction of the fastening system.

Testing of the methodical principles for minimizing the load on the extraction working fastening system in the Western Donbas conditions indicates their sufficient adequacy and reliability:

– on the one hand, with non-optimal interaction modes between the support and the rock mass, differences with normative documents in the results of load calculations give acceptable deviations for mining-engineering calculations;

– on the other hand, the developed optimization scheme indicates a significant effect of reducing the load on the support.

The patterns for the influence of geomechanical factors on the choice of optimal parameters of the fastening system deformation-strength characteristics have been determined: its minimum required reaction (load-bearing capacity) and the yielding property value. The patterns have been obtained in the form of graphs and regression equations for calculating the optimal parameters of the fastening system. A stable power-law relationship between the fastening system optimal parameters and the geomechanical index of mining conditions, regardless of the coal-bearing mass texture, has been revealed; this makes it possible to implement a unified strategy of resource-saving improvement of mine working fastening systems for the entire Western Donbas area.

Based on the determined optimal parameters for the fastening system operating modes, a methodology for calculating a function that describes the rational deformation-strength characteristic, depending on the mining-geological conditions of maintaining the reused extraction workings, has been developed and substantiated. The methodology is simple and effective for calculating the necessary rational parameters of the fastening system as a whole, for which its constituent fastening elements are selected.

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Обґрунтування оптимізаційних рішень взаємодії кріпильної системи виробки з вміщуючим гірським масивом

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Мета. Визначення раціональних режимів взаємодії кріпильної системи та вміщуючого масиву виїмкової виробки в зоні впливу очисних робіт.

Методика. Обґрунтовано алгоритм пошуку оптимальних рішень режимів взаємодії кріпильної системи з вуглевмісним масивом. Узгоджено деформаційно-силові характеристики елементів кріпильної системи. Оптимізовано конструктивні параметри елементів кріплення за критерієм їх рівноміцності. За оптимальними параметрами виконано обґрунтування та розроблено методику розрахунку функції, яка описує раціональну деформаційно-силову характеристику кріпильної системи залежно від гірничо-геологічних умов.

Результати. Проведено обчислювальні експерименти щодо визначення деформаційно-силової характеристики породного масиву. На основі нормативних документів проведено розрахунок розмірів склепіння природної рівноваги. Доведено адекватність методичних принципів мінімізації навантаження на кріпильну систему. Встановлено закономірності впливу геомеханічних факторів на вибір оптимальних параметрів деформаційно-силової характеристики кріпильної системи. Отримано методику розрахунків раціональних параметрів кріпильної системи в цілому та складових її елементів.

Наукова новизна. Проведено комбіновані дослідження мінімізації навантаження на кріпильну систему. Встановлено закономірності впливу геомеханічних факторів на вибір несучої здатності та величини піддатливості кріпильної системи. Отримано рівняння регресії з розрахунку оптимальних параметрів кріпильних систем з геомеханічним показником умов роботи, що дозволяє здійснити єдину стратегію ресурсозберігаючого вдосконалення кріпильних систем.

Практична значимість. Розроблено методику отримання деформаційно-силової характеристики масиву, що знеміцнюється, залежно від глибини розміщення виробки, текстури порід надвугільної товщі та її міцнісних властивостей. Доведено застосування методики щодо здійснення єдиної стратегії ресурсозберігаючого вдосконалення кріпильних систем виробок для шахт Західного Донбасу.

Ключові слова: *гірський масив, виїмкова виробка, оптимізація, методика, кріпильна система*