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State Higher Educational Institution
"National Mining University"

Mining faculty
(faculty)

Department of Underground Mining
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EXPLANATORY NOTE
of graduation project (work)
Master
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on topic: Investigate the mechanism of the roof support loading
in stopes and development workings
at the mine "Samarska" colliery group "Ternivske"

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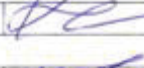


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очисних та підготовчих виробок
на шахті "Самарська" ШУ "Тернівське"

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Дніпропетровськ
2015

Ministry of Education and Science of Ukraine
State Higher Educational Institution
"National Mining University"

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« 22 » 06 2015

ASSIGNMENT
for Master's work

speciality 8.05030101 Mining and extraction

(code and name of speciality)

of mineral resources

student GRg-14m

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Topic of the thesis Investigate the mechanism of the roof support loading
in stopes and development workings
at the mine "Samarska" colliery group "Ternivske"

1 FOUNDATIONS FOR WORK PERFORMANCE

The order of Rector of State University "NMU" from 26.05.15 № 829-н

2 OBJECTIVE AND INPUT DATA FOR THE WORK

Object of research The stress-strain state of rock massif

Subject of research The mechanism of influence of deformation properties
of above-coal rocks strata on the roof support loading

Purpose The improvement of the operational reliability of the roof support in stope
and development workings

Initial data for work Mining and geological characteristics of the rock mass;
technical documentation for carrying out stoping and development work

3 EXPECTED RESEARCH RESULTS

Scientific novelty The establishment of regularities of the roof support loading and
rock pressure manifestation during the stoping and development works in complex mining
and geological conditions of Western Donbass mines

Practical value Fail-safeness of operations; improvement of the sustainability of stopes
and development workings; prevention of the longwall set of equipment landing on "rigid base"

4 REQUIREMENTS FOR RESULTS OF WORK PERFORMANCE

To present results of the performed work on the basis of computer modeling of the behavior of coal-containing rocks thickness in the vicinity of the stoping face

5 STAGES OF WORK PERFORMANCE

Name of work stages	Deadline
To review modeling of geomechanical systems in mining	10.02.15
Describe the technology of computational experiment	24.02.15
Substantiate the displacement of rocks in above-coal thickness and loading mechanism of roof support	10.03.15
Analyze the deformation mechanism of above-coal strata during the stoping	27.03.15
Analyze the loading mechanism of roof support under the displacement of above-coal strata	16.04.15
Analyze the influence of technological parameters on loading of powered roof support	30.04.15
Investigate the influence of geomechanical factors on the load of powered roof support	15.05.15
Make the modeling of displacement of the coal contain massif in the vicinity of the stope	05.06.15

6 IMPLEMENTATION OF RESULTS AND EFFICIENCY

Economic effect from the realization of results of the work is 2,3 ths. UAH per linear meter of the working; total economic effect is 3680 ths. UAH

Social effect from the realization of results of the work is improvement of the level of safety and working conditions of workers

7 ADDITIONAL REQUIREMENTS

Thesis must be designed according to GOST 3008 - 95 "Dokumentatsiya. Zvity u sferi nauky i tehniky. Struktura i pravyla oformlennya" and in accordance with methodical recommendations and requirements for the performance of degree works by students of the education and qualification level of Master from specialty 8.05030101 Deposits Development and mineral resources extract (underground mining)


Assignment issued


(signature)

Prof. Bondarenko V. I.

(surname, initials)

Assignment took to perform


(signature)

Stud. Lysenko R. S.

(surname, initials)

Issue date of assignment: 26.01.2015

Deadline for the thesis submission to examination commission: 15.06.2015

ABSTRACT

Topic: Investigate the mechanism of the roof support loading in stopes and development workings at the mine “Samarska” colliery group “Ternivske”.

Master thesis work: 203 pages, 49 Figures, 1 Tables, 64 references.

The object of research is the stress-strain state of rock massif.

The subject of research is the mechanism of influence of deformation properties of above-coal rocks strata on the roof support loading.

Purpose of the work is the improvement of operational reliability of the roof support in stope and development workings.


Research methods: overview of scientific research; analytical research; underground investigations; computer monitoring of pressure indicators in hydraulic props of roof support sections; computer modeling that is based on the numerical mathematical method of finite elements.

Results of underground investigations and numerical experiments by finite element method with the calculation stress-strained state of multiparametric geomechanical systems are presented, as well as underground investigations that include the stoping and development works during the extraction of thin and very thin coal seams in the complex geological conditions of mines of Western Donbass; recommendations on parameters of extraction works for these conditions on the basis of monitoring of the plough longwall are formed.

COMPUTATIONAL EXPERIMENT, FINITE ELEMENT METHOD, COMPUTER MODELING, GEOMECHANICAL SYSTEM, STRESS-STRAIN STATE, ROCK PRESSURE, DEFORMATION CHARACTERISTICS, ROCK MASS, MINING AND GEOLOGICAL CONDITIONS, MINING AND TECHNICAL CONDITIONS, FACE, DEVELOPMENT WORKINGS.

Міністерство освіти і науки України
Державний вищий навчальний заклад
"Національний гірничий університет"

ЗАТВЕРДЖЕНО:
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(підпис) (прізвище, ініціали)

« 22 » 06 2015 року

ЗАВДАННЯ
на виконання кваліфікаційної роботи магістра

спеціальності 8.05030101 Розробка родовищ та видобування
(код і назва спеціальності)
корисних копалин

студенту ГРг-14м Лисенко Р. С.
(група) (прізвище та ініціали)

Тема дипломної роботи Дослідити механізм навантаження кріплення
очисних та підготовчих виробок
на шахті "Самарська" ШУ "Тернівське"

1 ПІДСТАВИ ДЛЯ ПРОВЕДЕННЯ РОБОТИ

Наказ ректора Державного ВНЗ "НГУ" від 26.05.2015 № 829-л

2 МЕТА ТА ВИХІДНІ ДАНІ ДЛЯ ПРОВЕДЕННЯ РОБІТ

Об'єкт досліджень Напружено-деформований стан гірського масиву

Предмет досліджень Механізм впливу деформаційних властивостей породи
надугольної товщі на навантаження кріплення

Мета підвищення експлуатаційної надійності кріплення очисних та підготовчих виробок

Вихідні дані для проведення роботи Гірничо-геологічна характеристика
масиву гірських порід; технічна документація на провадження очисних та
підготовчих робіт

3 ОЧІКУВАНІ НАУКОВІ РЕЗУЛЬТАТИ

Наукова новизна Встановлення закономірностей навантаження кріплення та
прояву гірничого тиску при веденні очисних та підготовчих робіт у складних
гірничо-геологічних умовах шахт Західного Донбасу

Практична цінність Безаварійність робіт; підвищення стійкості очисних та
підготовчих виробок; запобігання посадки комплексу на "жорстку базу"

4 ВИМОГИ ДО РЕЗУЛЬТАТІВ ВИКОНАННЯ РОБОТИ

Результати виконаної роботи представити на базі комп'ютерного моделювання поведінки вуглевміщуючої товщі в околиці очисного забою

5 ЕТАПИ ВИКОНАННЯ РОБОТИ

Найменування етапів робіт	Термін виконання
Зробити огляд моделювання геомеханічних систем в гірництві	10.02.15
Описати технології обчислювального експерименту	24.02.15
Обґрунтувати зсування надугольної товщі і механізм навантаження кріплення	10.03.15
Проаналізувати механізм деформування надугольної товщі при очисній виїмці	27.03.15
Проаналізувати механізм навантаження кріплення очисного комплексу при зсування надугольної товщі	16.04.15
Проаналізувати вплив технологічних параметрів на навантаження механізованого кріплення	30.04.15
Дослідити вплив геомеханічних факторів на навантаження механізованого кріплення	15.05.15
Зробити моделювання зсуву вуглевміщуючого масиву в околиці очисного забою	05.06.15

6 РЕАЛІЗАЦІЯ РЕЗУЛЬТАТІВ ТА ЕФЕКТИВНІСТЬ

Економічний ефект від реалізації результатів роботи становить 2,3 тис. грн. на погонний метр виробки; сумарний економічний ефект становить 3680 тис. грн.

Соціальний ефект від реалізації результатів роботи полягає в підвищенні рівня безпеки та умов праці робітників

7 ДОДАТКОВІ ВИМОГИ

Дипломна робота має бути оформлена відповідно до ДСТУ 3008 – 95 "Документація. Звіти у сфері науки і техніки. Структура і правила оформлення", а також згідно з методичними рекомендаціями та вимогами до виконання дипломних робіт студентами освітньо-кваліфікаційного рівня магістра з спеціальності 8.05030101 Розробка родовищ та видобування корисних копалин (підземним способом)

Завдання видав



проф. Бондаренко В. І.

(підпис)

(прізвище, ініціали)

Завдання прийняв до виконання



студ. Лисенко Р. С.

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(прізвище, ініціали)

Дата видачі завдання: 26.01.2015

Термін подання дипломної роботи до ЕК: 15.06.2015

РЕФЕРАТ

Тема: Дослідити механізм навантаження кріплення очисних та підготовчих виробок на шахті “Самарська” ШУ “Тернівське”.

Дипломна робота магістра: 203 с., 49 рис., 1 табл., 64 джерел.

Об’єкт досліджень - напружено-деформований стан гірського масиву.

Предмет досліджень - механізм впливу деформаційних властивостей породи надугольної товщі на навантаження кріплення.

Метою роботи є підвищення експлуатаційної надійності кріплення очисних та підготовчих виробок.

Методи дослідження - огляд наукових досліджень; аналітичні дослідження; шахтні дослідження; комп’ютерний моніторинг показників тиску у гідростіках секцій кріплення; комп’ютерне моделювання, що базується на числовому математичному методі скінчених елементів.

Представлені результати шахтних досліджень та обчислювальних експериментів методом скінчених елементів з розрахунку напружено-деформованого стану елементів багатопараметричних геомеханічних систем, а також шахтні дослідження, що включають в себе очисні й підготовчі роботи при вийманні тонких і вельми тонких вугільних пластів у складних гірничо-геологічних умовах шахт Західного Донбасу; сформовані рекомендації по параметрах ведення очисних робіт для даних умов на базі моніторингу роботи стругової установки.

ОБЧИСЛЮВАЛЬНИЙ ЕКСПЕРИМЕНТ, МЕТОД КІНЦЕВИХ ЕЛЕМЕНТІВ, ПРОСТОРОВЕ МОДЕЛЮВАННЯ, ГЕОМЕХАНІЧНІ СИСТЕМИ, НАПРУЖЕНО-ДЕФОРМОВАНИЙ СТАН, ГІРНИЧИЙ ТИСК, ДЕФОРМАЦІЙНІ ХАРАКТЕРИСТИКИ, ГІРСЬКИЙ МАСИВ, ГІРНИЧО-ГЕОЛОГІЧНІ УМОВИ, ГІРНИЧО-ТЕХНІЧНІ УМОВИ, ЗАБІЙ, ПІДГОТОВЧІ ВИРОБКИ.

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INTRODUCTION

In recent decades, in the coal industry of leading coal-producing countries of the world are strengthened processes of production concentration, grows labor productivity, the quality, performance and reliability of the equipment are improved, to ensure the continuity of production processes is increasingly possible. In this context, the successful development of the Ukrainian coal industry, in terms of technical and technological policy, is possible only under the condition of concentration of mining and underground intensification of production on the basis of effective use of modern techniques and technologies. The positive results of the adaptation of the new equipment is especially pronounced in advanced mines, where the modern level equipment of faces has allowed to provide stable production of a single longwall up to 3-5 ths. tones of coal per day.

In the thin and very thin coal seams of Western Donbass are more than 90% of total reserves, at the same time, mining and geological conditions of the region is distinguished by a number of features:

- relatively high strength characteristics of coal seams - Protodyakonov scale of hardness coefficient is $f = 3 \dots 4$;
- coal-containing rocks mostly weaker than coal seams ($f = 1 \dots 4$), its structure is characterized by strongly pronounced stratification and fracturing;
- a sufficient number of thin layers of soft sandstone and coal seams are flooded, which contributes to a significant softening of already weak basic lithological differences of mudstone and siltstone.

These features significantly impair the conditions of mining operations because of intense manifestations of the rock pressure, and the thin and very thin coal seams are not contribute to high performance of the stoping work. Therefore, to achieve high performance of a face on Western Donbass mines has been applied the plough GH 5.7 № 9-38; 800-1585N «BUCYRUS». At the mine “Samarska” colliery group “Ternivske” in 161st longwall of the seam C_6 installed and successfully works the plough GH 9.38ve / 5.7 in complete with roof support DBT. Extraction site is located at the depth of $H = 300 \dots 490$ m and developed up-dip of the seam C_6 (the average

angle of slope 4°) with the extraction height 0,9 m. In the 300 m long longwall reached the average load on a face of 3000 tons/day, and on some days production reached 5000 tons/day of the raw coal. At the same time with relatively rhythmic work of the powered mining complex, the average advance rates of the stope varied over a wide range due to emergency stoppages, during of which there was an increase of rock pressure manifestations in the form of roof lowering, increasing the load on the support and hydraulic props safety valve reaction of individual sections. Also, the height of the workspace of the longwall was decreasing, which was dangerous in terms of the reliable operation of complex elements during the renewal of stoping work after the debugging. Also were observed oscillations of the intensity of rock pressure manifestations at different average daily advance rates of the stope.

Above information about the structure and properties of the above-coal strata is valuable in terms of the data analysis of computer monitoring of the roof support loading process and its projection on geomechanics of the displacement of roof rocks layer as the main factor for the load formation on the powered support.

The object of research is the stress-strain state of rock massif.

The subject of research is the mechanism of influence of deformation properties of above-coal rocks strata on the roof support loading.

The idea of work is the usage of the finite element method computational experiment for calculation of the stress-strain state of the rock mass and establishment of regularities of the roof support loading.

The purpose of work is the improvement of operational reliability of the roof support in stope and development workings.

The assignment of research is to analyze and model the mechanism of the roof support loading in stopes and development workings at the mine “Samarska” colliery group “Ternivske”.

Scientific result: established a series of regularities in the monitoring process of the plough operation under the analysis of the pressure reading in hydraulic props of support sections of the longwall set of equipment using various combinations of the geomechanical and technological factors.

Scientific novelty is the establishment of regularities of the roof support loading and rock pressure manifestation during the stoping and development works in complex mining and geological conditions of Western Donbass mines.

The reliability of scientific conclusions, regularities and recommendations is confirmed by the use of the main provisions of existing concepts of the rock displacement of above-coal strata in the complex mining and geological conditions of Western Donbass; sufficient volume of the performed theoretical and experimental researches coincides (the discrepancy of results is 7-23%).

Practical value: fail-safeness of operations; improvement of the sustainability of stopes and development workings; prevention of the longwall set of equipment landing on “rigid base”.

Scientific provisions: under the variation of the structure of above-coal strata (along the length of the longwall), the degree of water content, the intensity of fracturing and other geomechanical factors affecting the stability of the roof rocks occurs pressure fluctuation upward or downward from the average value in different areas of the longwall.

Research methods: overview of scientific research; analytical research; underground investigations; computer monitoring of pressure indicators in hydraulic props of roof support sections; computer modeling that is based on the numerical mathematical method of finite elements.

1. OVERVIEW. MATHEMATICAL MODELING OF GEOMECHANICAL SYSTEMS IN MINING

1.1. History of mathematical modeling development

Models and simulations are used by mankind since ancient times. Language, writing and drawing were developed by using the models. Rock paintings of our ancestors, and later paintings and books there are models, information forms of transmission of knowledge about the world to the next generation. Model used in the study of complex phenomena, processes, designs of new facilities. Well-built model, tend to be more available for research than the real object. In addition, some of the objects in general cannot be studied by direct research.

Originally the elements of mathematics appeared due to the need to solve practical problems: the measurement of location, facilities, navigation and others. Mathematicians are always interested in the numerical solution, the greatest of which are combined in their research study about the nature phenomena, obtaining the mathematical description and analysis. First of all, the analysis of complicated models put the demand for the creation of special numerical methods for solving problems. The names of some of these methods are the method of Newton, Euler, Lobachevsky, Gauss, Chebyshev and Hermite. Their development were engaged the greatest scientists of our time.

The first period of mathematical modeling started three or four thousand years ago. It was connected with the calculation of basic tasks of arithmetic, algebra and geometry. Own fingers were computational tools and then it was abacus. Initial data contained a few numbers and most of the calculations were carried out without rounding.

The second period began with Isaac Newton, when the problems of astronomy and geodesy were solving and mechanical structures were calculating, which were reducing to ordinary differential equations or algebraic systems with a large number of unknowns. Usually calculations were carried out with rounding, but sometimes the result required high precision, so it had to leave up to eight significant figure. Computational tools become more diverse: the table of elementary functions, the

adding machine, the slide rule. At the end of this period appeared relatively complex mechanical type calculators.

The third period began in the mid of 40-ies of XX century. The problem that emerged in the course of military operations of Second World War served as an incentive. Large speed of combat vehicles attack (aircraft, tanks, ships) required the development of effective means of active protection. Outstanding scientists of that time undertook to solve this problem. Among them was the Norbert Wiener, the famous American mathematician. In this sense, for example one of the applied problems that was needed Wiener's solutions is considered. For effective firing of air defense is necessary to provide the future position of the aircraft, in other words extrapolate its trajectory for some time ahead in order to the projectile and the plane at the same time hit the "meeting point". However, accurately predict the future trajectory of the aircraft is not possible, because the pilot, who performs evasive maneuver, can act quite willfully. Therefore, the trajectory should be considered as a random and a weapon that is firing should provide a random trajectory. It is the task that was set by Norbert Wiener as a mathematician. At first glance, he took up completely hopeless task, because to predict how will pass a random trajectory is simply impossible. Exact trajectory foresight is impossible, but when firing anti-aircraft projectile special accuracy is unnecessary. And in addition, there are not so many randomness in the flight path of the aircraft, because its maneuver is limited by physical laws of motion of the aircraft and its design. Residual uncertainty can be attributed to randomness which can provide only approximate by statistical methods. This problem was solved Wiener. Method, which he proposed, required large amount of computation that must be done in those moments before the plane was approaching the target. Therefore, the new calculations principles that exclude mechanics became necessary. They gave impetus to the creation of electronic computers (PC).

Starting from the 50-ies of XX century, science has come close to the study of phenomena where cause and effect differ by several orders of magnitude. Due to these phenomena have arisen electron tubes, transistors, computers, lasers, etc. In most cases, these phenomena are very poorly exposed to traditional methods of

analysis. Later, in the course of development and improvement, this line of theoretical analysis was transformed into a new, modern technology and methodology of theoretical research, which was called computational experiment [1]. The basis of the computational experiment is mathematical modeling. The theoretical basis is applied mathematics and technical base is a powerful electronic computers.

Mathematical models are one of the main tools for understanding phenomena of the surrounding world and they are understood as the basic laws and communication that characterize phenomenon under study. This may be formulas or equations, sets of rules or requirements, which are expressed in mathematical form. Many centuries for the description of variety of phenomena in mathematics, mechanics, physics and other exact sciences used mathematical models.

The foundation of any mathematical model is specific algorithm, which indicates the sequence of computational and logical operations that are performed to obtain numerical solutions. History of mathematics is connected with all algorithms. The word “algorithm” is derived from the name of a medieval Uzbek scientist Al-Khwarizmi. The ancient Greek scientist was known algorithm for finding the number of “pi” with high accuracy. Newton proposed an efficient numerical method for solving algebraic equations, and Euler proposed method for numerical solution of ordinary differential equations. It is well known that the modified methods of Newton and Euler still occupy an honorable place in the arsenal of computational mathematics.

Now we can talk about three generations of mathematical modeling. First it is usually a question of mathematical notation of individual phenomenological observations on real objects. They are characterized by the simplicity of description, typical linear of equations and small dimension (often reproduced only one or two variables). Methods of analysis are mainly associated with the analytical solutions. In the second generation there are models that recreate the object “in all its fullness”. Models reflect the structure and functioning laws of the object and they become substantially nonlinear. Purely mathematical apparatus is complemented by logical-semantic unit. Dimension increases up to several tens of variables. Such models are

called “complex” or “large” and the working tool becomes a computational experiment. In our time, the transition begins to the third generation of mathematical models of the virtual world that can be described as reproducing of three-dimensional world by computer resources. This significantly increases the amount of processed and displayed information. For example the number of displayable details reaches several thousand.

Beginning of the development of computational experiment technology falls on the 50-ies year of XX century. Date of the first major results of computational experiments in the Soviet Union was officially recorded. 1968 year, when the USSR State Committee for discoveries and inventions has certified discovery of the phenomenon, which in fact nobody has observed. It was the discovery of so called T-layer effect (the temperature of the current sheet in the plasma). In this case, computational experiment was preceded by full-scale experiment that has already “been ordered” by the results of mathematical modeling. In recent years, numbers of Nobel prizes in chemistry, medicine, economics, and physics was awarded to works that had methodological basis of the mathematical modeling.

Computational experiment, in contrast to full-scale experimental units, allows to accumulate results that obtained in the study of any range of tasks, and then quickly and flexibly apply them for solving problems in a completely different field, which indicates universality of mathematical models.

Many fundamental problems of applied modeling were identified in first time by I. A. Poletaev. He was the first who gave the original classification of models by the purposes of their use: “search” model for hypothesis testing, “portrait” model for replacement of object in the experiment and “research” model that in the modern sense means orientation on the complex computational experiment. For example, in the early 70-ies of XX century new phenomena of solitary wave (“sameton”) were discovered (or rather they were ignored). Arising as a result of the earthquake, this wave has a strange resistance, and extends into the ocean over long distances almost without power losses. It was modeled in the computational experiment and was observed in practice. Mathematical theory of this phenomenon is unknown, but the

numerical study allowed to determine the conditions of occurrence, distribution and properties of the phenomenon.

In the period from 1850 to 1875 years, due to the efforts of Maxwell, Castigliano and Mohr and before them, Navier and Saint-Venant were developed the basic concepts of analysis theory of bar structures, which became the source of matrix methods of structural mechanics. Eighty years later, these studies formed the basis of the finite element method.

Development of the theory and auxiliary disciplines that relate to the finite element method was extremely slow during the period from 1875 to 1920s. It was mainly due to the real difficulties in solving algebraic equations, as soon as the number of unknowns becomes large. For structures at that time almost always was applied approach that was based on setting of stress distribution with the parameters of load as unknowns. Approximately since 1920, with Meinie (US) and Ostenfeld (Netherlands) efforts have formulated the basic ideas of numerical research of frame and truss structures, which were based on setting of displacement as unknowns. These developments preceded by the modern matrix methods research of structures. The most important limiting factor in the analysis was the dimension of the tasks that defined by amount of unknowns parameters of displacements or loads. It was until 1932 when Hardy Cross has proposed the method of moment distribution. The method of moment distributions allowed to investigate numerically the behavior of structures in more complicated tasks than it was before. This method became the basis of the numerical study of behavior of structures for the next 25 years.

Computers appeared in the early fifties, but their true value, in both theoretical and applied aspects was not so obvious at that time. Nevertheless some scientists, who foresaw the influence of computers, made an attempt to formulate in convenient for computers, well developed by that time, matrix form of calculation algorithms of truss structures. Until 1950 in the works of Courant, McHenry and Khrennikov have been already displayed almost all key positions that characterize the solution of problems with using finite element method. Since the mid-fifties, the finite element method in its development has passed through a series of continuous modifications.

In the development of the finite element method took part such scientists as Zenkevich, Oden, Freese, Pobedrya and others. Researchers have written finite element equations for the solid deformed body, elastic plates and other engineering forms. Once ratios were obtained for the study of the static behavior of linearly elastic material, the attention of specialists was attracted by such aspects as dynamic behavior, different buckling modes designs, geometric and physical nonlinearity. Subsequent to those studies, began a period of rapid development of computer programs that implement the finite element method and allow to large number of specialists to use the obtained results in their work.

1.2. General characteristics of mathematical modeling

Ongoing growth of knowledge and depth of understanding of the physical phenomena made it necessary to create more complex models that more appropriately describe not only objects but also processes: universal approach in this case was the mathematical modeling. Mathematical model is model of a real object or process, which is based on systems of mathematical equations that describe the specific process or object. Computer model is mathematical model, which is implemented by means of computers. If the state of the system changes over time, the model is called dynamic, otherwise - static.

Computational experiment is the method of studying of objects or physical processes by mathematical modeling. It provides that after construction of a mathematical model, its numerical study is carried out to determine the behavior of a system in different conditions or different modifications [2]. Numerical study of the model makes it possible to determine variety characteristics of processes, to optimize a design or operation modes of designed devices. It happens that during a computational experiment, the researcher unexpectedly opens up new processes and properties, which previously was unknown.

Processes in the system can occur in different ways depending on conditions in which the system is located. Monitor the behavior of the real system under various conditions can be difficult, sometimes impossible. In such cases, the constructed

model can be reused for return to the initial state and monitor its behavior. This research technique is called simulation modeling.

After the advent of electronic computers, new methods, which focus directly on the computer, began to develop rapidly, for example the Monte Carlo method [3]. For solving a problem, the first stage of the work - formulation of a mathematical model (problem). Model for the physical process usually consists of the equations that describe the process. These equations include coefficients in form of characteristics of objects or of substances that involved in the process. Any studied phenomenon infinitely complex, it is connected with other natural phenomena. Therefore, the mathematical model should cover the most important features of a phenomenon for this problem, because if the model is not selected carefully enough, no matter what methods we used for the calculation, all conclusions won't be reliable enough and in some cases may be completely wrong. After the problem was set, an algorithm for its solution is developed, i.e. numerical method itself.

The scope and possibilities of computational experiment increase with development of computer technology. Complexity and diversity of solved problems are increasing: a large amount of information, which obtained during an experiment, requires adequate methods of its presentation. Instead of arrays of numerical data and simple graphs are increasingly being used visual images [3, 4] that facilitate complete and timely interpretation of the results.

Software systems that supply a computational experiment are massive and complex. The appearance of the original version of the program only in the most general terms outlines the direction of research. But the main work concerning programming is still ahead - it is associated with multiple modifications of the program that display the evolution of mathematical models and methods of calculation. Number of cycles of computational experiment, each of which is associated with a modification of the calculation program, often reaches tens of thousands. Therefore, the rational organization of such modifications is the key to effective programming of this class of problems.

The complexity of mathematical models, as well as the need to store many of variant modules, leads to the fact that the characteristic sizes of the program fund of computational experiment are quite impressive [5]. The number of modules that are involved in the calculations, commonly up to several thousand, and the total length of the code are hundreds of thousands of lines. All of these difficulties can be overcome by a computational experiment. Many important practical problems have been solved successfully. These tasks can be divided into a number of elementary problems, such as calculation of the integral, solution of the differential equation and the like. Many of elementary problems are simple, they are well studied, for them have already developed methods for the numerical solution and exist standard software [5, 6]. During the computational experiment researcher using the user interface can “play” on the model variety of options which are interesting to him [7]. In such a way, researcher receives a powerful tool for analysis and prediction of the behavior of complex nonlinear multivariable objects and phenomena, the study of which by the traditional methods is difficult or impossible.

Important positive qualities of computational experiment are its universality which makes it easy to transfer this technology to the study of other objects. This circumstance is typical for general mathematical modeling and is generated by the fact that many phenomena and processes have the same mathematical model. Multipurpose direction and methodological flexibility of computational experiment allows solving new challenges quickly and effectively. They are based on the experiences of mathematical modeling, the bank of computational algorithms and software.

The second feature of the computational experiment, as technology of scientific research, is its interdisciplinary nature. We constantly emphasize this fact, saying that applied mathematics united theorists and experimentalists for quicker achievement of a common goal. Computational experiment can be considered as a convenient form of cooperation of mental labor, which increases its productivity [8]: in the single cycle of the computational experiment works theorist, experimentalist, applied mathematician and programmer.

The following features and advantages of the computational experiment compared to full-scale experiments should be noted. Firstly, the computational experiment is carried out even when the full-scale experiment is impossible. Second, the application of the computational experiment sharply reduces development costs and saves time. It is provided by multivariant calculations of simple modification of mathematical models for simulation of various real conditions. The creation of new products and technologies associated with the need of hard, expensive and continuous debugging. Computational tools allow save a lot of time and money at this stage of development.

Computational experiment should be considered as a new technology of scientific research for the future, as a trend, as logic of the development of scientific research organization. Stronger connection of theoretical and experimental research in a single technology of scientific research is an unstoppable trend of our time and the main link of this methodology are the mathematical modeling and the computational experiment.

Rapidly developed direction of computational mathematics - numerical optimization methods [9], which study extreme (highest or lowest) values of functionals on sets of the structure. The first thing to recall is problems of mathematical programming (linear and dynamic). Also optimization problems include *min/max* problems, which appeared during problems solving of operations research and games theory. Solution of complex problems (especially large size problems) brought to life one of the main directions in the theory of numerical methods - study of the stability of methods and algorithms to various kinds of errors [10]. Unsustainable problems required special definition of approximate solutions and the development of appropriate methods to find them. Unstable problems include a wide class of problems which are connected with problems of the processing automation of experimental results. Application of computers is continuously expanding range of users and therefore there is the trend of automation, which makes less significant user familiarity with numerical methods. This makes new demands on algorithms and standard programs for solving typical problems.

The advent of next-generation computers with large operation speed caused the appearance of a narrow link in the human-machine system: the speed of programming that gave rise to the new stage of programming - the creation of algorithmic languages with translation from the algorithmic language into the internal language of the machine. Due to the greater proximity to universal programming languages, their implementation has simplified programming and significantly expanded the range of users. Perspective direction for the implementation of this technology is the use of universal supercomputers with mass parallelism and programmed architecture. To create computing systems are developed instrument programming systems of real-time on the basis of unified hardware supercomputers. Problem, which algorithmically formulated for solving, comes to the formalization. The formalization is a complex of software and mathematical tools for the analysis of the algorithm in accordance with requirements for parameters of a supercomputer. Transformation of the algorithm into the form, which effective for solve a problem, is carried out by converting the information graph of a problem in the form of frames and by separation of the algorithm on structural and procedural components. The structural component is represented as hardware implemented computing fragments (frames), and the procedural component - in form of sequence of their activation. Formation of the procedural component includes the following stages:

- 1) separation of information graph into subgraphs by the criterion of minimizing the time of problem solving;
- 2) formation of restrictions on data placement in distributed memory channels according with the requirements on the permissible number of channels;
- 3) formation of non-conflict structures and the synthesis of procedures of addressing and changing of commutation.

Highlighting of structural and procedural components of the algorithm allows to determine the minimum base diagram and rules of its buildup, and also allows to submit applied problem in the form of inductive program. This program can be performed in any combination of supercomputer basic modules. Parameters of the inductance program is amount of basic modules and their combination, and

translation of a problem in machine code (configuration command of macro processors and commutators, operators of controllers of extended memory) is performed on the final stage of development of the complex.

In recent years, software packages, which focused on the computational experiment, became widespread [8, 11, 12]. These software packages represent workbench of mathematical models creation. On the basis of these models computational experiment is implemented. Consider the most popular from similar software products. Mathematica package is implemented for different computers that are compatible with the IBM, PC, Macintosh and workstations Next and Sun, as well as supercomputers Gray. Mathematica package relates to computer systems of symbolic mathematics. This feature allows to obtain a solution for the specific data and in general terms. The package is focused on scientists, mathematicians and analysts. It contains a large set of numerical methods and algorithms and has a modern interface. The software package relates to interpretational systems that implement the analysis and interpretation of data. Computing environment allows to the user add new functionality that provides adaptation of the system for different specific tasks.

Another powerful mathematical tool is the package of Matlab (Matrix Laboratory). The package is designed for mathematical modeling and provides research in many areas of scientific and technological applications. Packet structure can effectively combine different approaches to the creation of mathematical models, including analytical approach and simulation. The basis of the simulation is statistical experiment. The package contains the object-oriented programming language. Approximately 30 instrumental applications of the package allow to provide the solution of differential and algebraic equations, integral calculus, symbolic computation and so on. In addition to the standard set of mathematical functions, the package also contains non-traditional algorithms - means of digital image processing, searches of solutions based on fuzzy logic, the unit of design and analysis of neural networks. Matlab can work with Windows, UNIX and Mac OS.

The present market leader of mathematical packages is MathCad. This software is focused on solving various problems of analysis and interpretation of data: the solution of individual algebraic equations and their systems, the solution of ordinary differential equations and their systems, the solution of differential equations in partial derivatives, statistical data processing (interpolation, extrapolation, approximation, etc.), work with vectors and matrices, the search of the extremum of functional dependencies. Numerical and analytical solutions of various problems provide by features of symbolic mathematics, which are integrated to the system.

At the present stage of history for solving engineering, scientific and practical problems based on solid modeling are used engineering design systems (AutoCAD, SolidWorks, Cosmos 3D). Models created by these software tools, during the computational experiments are used in systems of engineering calculations, such as NASTRAN, DesingSTAR or Cosmos M. Recently, in this segment of applied research, integrated systems of engineering design and analysis became popular. This allows to perform calculations and process results without leaving the familiar environment of the development of original mathematical model.

At the moment, the most common software package for problems solving by the finite element method is the package from ANSYS Inc. This commercial product, developed since 1970, has become one of the pioneers of the finite element analysis. The development of this package also contributed to the establishment of industry of computer-aided design.

Currently, the ANSYS program runs on most operating systems of computers - from PC to workstations and supercomputers. A special feature of the program is file compatibility of different versions of ANSYS family for all applicable platforms. This means that the model which was created on a PC can be processed on a supercomputer. Multipurpose orientation of the program is in implementation of tools for describing the system response to the impact of different physical nature. The program allows to use the same model for solving related problems such as durability under thermal load, magnetic fields effects on the structural strength, heat and mass

transfer in the electromagnetic field. That provides flexibility during the use of the program for wide range of engineering problems.

For researchers, which implement mining engineering tasks, this program offers an ever-growing list of calculated tools, which can take into account variety of structural nonlinearity. The program makes possible to consider the most general case of the contact problem for surfaces. It admits the presence of large (finite) deformations and rotation angles. This program allows to perform interactive optimization and analysis of the impact of electromagnetic fields, to obtain the solution of hydroaerodynamics problems etc. All these are possible together with parametric modeling, adaptive meshing, using p -elements and extensive opportunities to create macro commands by using the ANSYS parametric design language [13].

The ANSYS program uses three iterative algorithms: highly efficient PowerSolver algorithm on the basis of the method of conditioned conjugate gradient, algorithm on the basis of the method of Jacobi conjugate gradient and implementation of the method of Cholesky partial conjugate gradients. With access to these tools, the researcher can choose one of the most suitable for the successful solution of his problem. In general case for large and complex tasks is preferable to use an iterative solver. It provides an opportunity to obtain more efficient solution of spatial problems of different physical nature and other labor-intensive analysis which are mathematically described by sparse, symmetric, positive definite matrices.

Module ANSYS Design Data Access (DDA) provides transfer into the models program created by the computer aids design (CAD) of third party developers, which allows to extend the range of three-dimensional modeling tools used at the initial stage of the formulation of the problem. DDA Connection software can work in conjunction with developments of many of the leading suppliers of CAD-programs including companies Parametric Technology Corporation, EDS/Unigraphics and Computervision Corporation. The latest version is presented by the DDA Interactive software module. This module allows to use for the finite element analysis directly CAD-models, which is achieved by the modern interface between CAD-data and initial data for analysis.

ANSYS/Multiphysics is the most powerful and multipurpose software package in the family of ANSYS program. The package is the software tool for the analysis of wide range of engineering disciplines, which allows to carry out computational experiments in selected areas of knowledge such as strength, heat propagation, fluid mechanics and electromagnetics, and solve related problems. This software package delivery option provides the ability to optimize project designs with a wide range of descriptions of physical environments that allows to carry modeling in the most complete formulation.

Currently, the market of software systems in the field of physical and mathematical applications continues to grow. New software packages are developed on the basis of modern computer technology with using the latest achievements of research methods. This creates software tools that are able to solve complex scientific and engineering problems.

In such a way appearance and development of the mathematical modeling objectively related with the history of mankind. At all times, greatest scientists have been working on getting more or less adequate mathematical description of real objects, phenomena or processes that caused the appearance of the trend - numerical methods for problems solving. Over time, new methodology and technology of problems solving was created. It was called a computational experiment. The theoretical basis of computational experiment has become applied mathematics and the technical - mainframes. Range of application and possibilities of computational experiment are expanding with the development of computer technology. Complexity and variety of tasks are increasing too. Presently computational experiment should be considered as the new technology of scientific research and in the future - as their new organization for effective solutions of problems that humanity will face.

1.3. Features of geomechanical systems modeling in mining

In problems of geomechanics the application of mathematical experiment avoids unnecessary and labor-intensive pilot researches, statement of which requires considerable financial costs and leads to significant loss of production time.

Solving features of geomechanical problems can be distributed into three main groups [14, 15]:

- 1) consideration of the structure of the rock mass, which is characterized by the presence of rock layers that differ significantly at mechanical properties, interstices, several fracture systems (or their absence in some layers), contact areas of layers with different adhesion and friction force, and the like;
- 2) in the process of the problem solving is necessary to consider not only the elastic-plastic deformation, but also the so-called stage of weakening and loosening in certain areas of the rock mass, in other words, for adequate modeling is necessary to take into account the physical nonlinearity of the rock;
- 3) geomechanical processes in the rock mass depend on the rheological properties of rock layers, sizes and relative position of mine workings, changes of their parameters in time and space during mining operations, i.e. geomechanical processes require the construction of a dynamic model.

For the given reasons for solving of geomechanical problems necessary to get the following information:

- structural and mechanical properties of the investigated layered rock mass;
- types and amounts of mechanical influences, which are attached to certain geometric areas of rock mass and artificial structures;
- type of problems, which are subject to numerical investigation: the distribution of stresses, strains, displacements; the destruction of some sections of the rock mass, artificial structures and the like;
- geometrical, mechanical and force parameters of underground artificial structures and their elements.

On the basis of given data is formed the design scheme, type of which determines selection of method for solving a specific geomechanical problems. In such a way compiles the system of mathematical equations expressing the relationship of the given and unknown values. This system must be solved to obtain the final value. Unfortunately, in most cases there is no possibility to obtain

numerical results using only analytical solution. In some problems necessary to involve numerical methods that provide a solution within a certain error [16-18]. Formation of design schemes for problems of geomechanics associated with the need to consider a large number of irregular parameters and complex boundary conditions. Therefore, researchers are forced to simplify the formulation of problems in order to reduce the dimension of the equations that describing the system, and functions that consider features of the initial load and geometry. Because in such way sufficient error already has been entered in the calculation scheme, the wide application of numerical methods in problems of rock mechanics becomes quite reasonable.

At the early stages of numerical methods development was considered it possible to obtain for problems of geomechanics only qualitative results [15]. With the development of numerical methods and perceptions of the stress-strain state of the rock mass becomes possible to obtain not only adequate qualitative, but also sufficiently accurate quantitative results [14]. The variety of calculation schemes can be represented as a combination of three main classes:

- 1) Geometric indications: the notion of planar and spatial solutions; the presence or absence of symmetry; the use of simply connected, doubly connected and multi connected domains, and the like.

- 2) External influences and mechanical properties of the rock mass and engineering structures: static or dynamic loads; consideration of filtration of liquid and gas; consideration of fluctuations temperature; isotropic, orthotropic or anisotropic medium; elastic, elastic-plastic or visco-plastic deformation of the rock mass, with or without taking into account its weakening and loosening, and the like.

- 3) Design and technological features of fastening, protection and operation of mine workings: design and technological scheme of the mine workings construction; maintenance sequence of preparatory works and stoping; characteristic of mine workings and methods of their protection; design and operation modes of roof supports.

Each class of calculation schemes generates specific approach in the formation of a mathematical model, and the combination of considered factors enables to

complicate or to simplify solution. Therefore, historically the picture of phased complexity of mathematical models of geomechanical processes caused by the growth of computing power, on the one hand, and the complexity of operating conditions of underground structures, on the other hand, can be observed. During the solution of geomechanical problems are used sometimes so-called combined numerical methods. One of these methods is the finite element method - initial parameters method. It allows [14] considering technological and design features of the support by presenting it as a defined configuration rod system. Such elements are jointed with the finite element method mesh of the rock mass in the nodes, which are located on the portion of mine working. This approach, in certain extent, allows simplify the solution of the contact problem for the contact area of the rock mass and roof supports of mine working.

At one time analytical methods were widely applied. Using these methods was obtained the definition mathematical solution, which gave an opportunity of the preliminary assessment of the rock mass behavior under simple loading schemes without detailed consideration of design and technological features of the functioning of the geomechanical system and in conditions of homogeneous computational domain. Despite widespread development of such methods, they have a common drawback - at the stage of computational scheme creating is necessary to take certain simplifications, which adversely affect the accuracy of the final result. In summary, it can be argued that variety of analytical solutions and numerical methods in problems of geomechanics continues to grow and evolve. Development occurs by complication of design and technological schemes of underground structures, advancement of the quantity of mechanical properties of rocks, which are accounted in a model, and operation parameters of roof supports and shielding elements of mine workings.

1.4. Trends in application of the finite element method in the study of geomechanical systems

During the solving of geomechanical problems constantly have to face the challenge of calculation of systems that have complex geometric configuration and irregular physical structure. The rock massif has a large number of the structure

characteristics and properties, consideration of which at the mathematical modeling is possible only with using the finite-difference schemes of calculation. Currently the de facto standard in the solution of geomechanical problems is the finite element method.

Since the late 60-ies and to the mid 80-ies of XX century by Soviet scientists-mechanics were made extensive researches. Primarily is necessary to note the work of O. K. Zenkevich, B. Z. Amusin, J. S. Yerzhanov, V. Y. Isaacson, Y. M. Lieberman, A. F. Fadeev, E. G. Morozov et al. They laid the foundation for the application of the finite element method methodology for solving problems of geomechanics in the elastic and elastic-plastic formulation.

Methodological principles of finite element method solutions of large variety of tasks are outlined in the paper “Finite element method in engineering” by O. Zenkevich [19]. In order to solve physically nonlinear problems was proposed three options increment method: variable stiffness, initial stress and initial deformation. The use, of one or another method of calculation specific problem, caused by the feature of loading scheme, the behavior of materials and the combination of boundary conditions.

One of the fundamental work in the field of the finite element method application in problems of geomechanics is the publication of Yerzhanov and Karimbayev, “The finite element method in problems of rock mechanics” [20]. Authors inspected a large range of features, which are inherent in the rock massif, in the context of mathematical description for using with existing mathematical apparatus of the finite element method. In particular, the calculation features of transversely isotropic body with the banding of the massif are considered. In them are demonstrated how on the basis of the generalized Hooke's law is constructed stiffness matrix of the single element, and then is provided a solution for the entire system of flat triangular elements. Were also analyzed methods for determining the stress and strain fields in inhomogeneous media with considering interaction of roof supports with the surrounding massif. Then authors showed the technique of solving the elastic-plastic problem on the basis of experimental bilinear dependence of stress

intensity from the intensity of the deformation. The solution was obtained in the generalized three-dimensional formulation without structural heterogeneity of the rock mass. Relatively to origination of numerical values was used the method of variable parameters of elasticity. It is shown how on the basis of the calculation of stress intensity in the elastic formulation are searched elastic modules that allow to perform following iterative calculation. The sequence of these calculations with variables modules of elasticity in combination with increasing of the load, acting on the system, makes it possible to solve elastic-plastic problems with the descending branch of the full diagram of rock deformation. However, with this approach remains unclear, how close to obtained results are located real phenomena.

In the Fadeev paper “The Finite Element Method in Geomechanics” [21] significant attention is paid to inhomogeneity of physical and mechanical properties of rock layers that form the massif. The classification of mathematical models of rocks according to their strength characteristics was performed. Rocks are divided into three main groups: soft rock, medium-hard rock and hard rock. Also, the technique of integration of mathematical models of rocks in the calculation scheme of the finite element method was represented. Especially is necessary to allocate solution of the problem of determination the stress-strain state of the rock mass with the full diagram “stress-strain”. Method that was proposed by author, for a solution searching by the method of fictitious forces is well conformed with the energetical representation of the equilibrium of a continuous medium in the transcendental state. The solution in this form does not provide complete definiteness of results, but significantly reduces the dependence of the solution accuracy from factors that reflect the original model. Until now, this approach is considered as the most suitable for solving problems, which take into account the behavior of the material beyond the strength boundary.

On the basis of these studies the team of authors has created and successfully uses the computer program complex “Geomechanics” [22], which with a high degree of reliability can perform the following application tasks of geomechanics in the plane and axisymmetric settling:

- determination of the elastic-plastic stress-strain state;
- consolidation of watersaturated soil;
- stationary and non-stationary temperature problem of pressure and gravity filtration;
- evaluation of the seismic stability of constructions.

However, the complex does not allow to solve contact problems in a complex configuration of space statement. Also, the complex has a small set of mathematical models of rocks which are used in the calculations. In a certain extent, this narrows the range of problems, the solution of which is potentially possible with help of the software.

In the work [23] of A. V. Potapov the aspects of finite element method calculations for geomechanical phenomena with low internal friction were considered and the solution of the problem of gob rock deformation was shown. The computational model of the problem can only be spatial, since it is necessary to take into account the specific behavior of the destroyed material, the pieces size of which affect the angle of internal friction of the rock.

Mathematical modeling by the finite element method of origination process of rock failure zone near the mine working and rules of its formation is made in the work [24] of G. V. Babiyuk, A. I. Melezhik and S. A. Kurman. Modeling of the rock mass behavior in the zone of influence of the working face was provided by replacement of the spatial problem into a series of consistently solving plane problems, where initial and boundary conditions for each step are given on the basis of results of the solution of the previous step.

Methodology that avoids the iterative process during the solution of nonlinear problem of geomechanics was given in the work [25] of N. A. Samodelkina. In this paper, was developed a procedure for combining methods of variable parameters of elasticity and initial deformations. The procedure based on the principle, which allows to define additional technogenic movement by changing the area of the initial displacement field. Solution is performed by using the piecewise linear diagram “stress-strain”, which is characterized by elastic modulus, compressive strength,

modulus of recession and residual strength of the rock. The advantage of this approach lies in the combination of simplicity of calculations and accountability of complex rheological behavior of the rock mass.

Combined method of calculation of stress-strain state of the rock mass is developed on the basis of the joint use of a group of finite-difference methods [26]. The essence of the proposed method of calculation is the following: in the first stage the area of the impact of mine working on the surrounding rock massif is determined by one of the numerical methods (finite element method or boundary element method) in the elastic formulation. Then in this region is determined the boundary of the zone the rock mass weakening in accordance with the selected criterion of strength. Within this zone, the behavior of rocks is described by definite relations of weakening rock. The behavior of the destroyed material is analyzed by discrete element method. The proposed method extends the capabilities of calculation of the rock mass in the nonlinear formulation. However, the authors do not give a mathematical formulation of the problem, which provides the transition from one finite-difference model to another. In particular, one of the key problems is still unsolved - the definition of the boundary conditions at the contact area of individual finite-difference schemes.

For the rock mass, which was weakened by several systems of cracks, was created method of numerical modeling of fractured rock masses [27]. In this technique the mathematical model of the material strength takes into account impact of the inclination angle of the fracture system. The proposed model allows visually and reliably estimate the stress-strain state of the fractured rock mass.

In the works of Kovalevskaya, Ovcharenko [28] and Fomichev [28, 29] were considered modeling aspects of the tridimensional breaking state of the rock massif during underground mining of series of coal strata, where are taken into account the angle of internal friction of rocks, the seam inclination and physical and mechanical properties of individual rock layers. As a criterion of strength loss is used generalized criterion of Coulomb-Mises, which can greatly simplify the procedure for determining boundaries of the ruptured zone. Then for elements, which were in the

area of weakening, is calculated coefficient of the residual strength of the rock that is used in the future to determine growth parameters of weakening zones and broken rocks. Unfortunately, the authors did not take into account mutual slippage of rock layers. Model of the rock mass is considered weightless and does not determine the movement of rocks by using the finite element method, and features of the interaction of mine working support with the rock mass in the calculations are not taken into account.

In such a way the brief review of existing researches convincingly proves that the use of the finite element method for solving problems of geomechanics finds broader spectrum of applications. The analysis technology is constantly improved and become more involved. Solution of the tridimensional problem of geomechanics in the elastic-plastic formulation has become the norm. In many cases, researchers take into account large number of factors and their combinations. In general, the development of mathematical models is aimed on searching of accessible accuracy of the calculations, which are carried out by classical methods, but are considered with the maximum possible number of factors.

According to the analysis of works, which were devoted to finite difference modeling of geomechanical processes around excavations, are clearly seen the following trends:

- 1) The finite element method, which has versatile possibilities, is most widely used. However, in solving complex geomechanical problems that are simultaneously analyzed multifactorial rock mechanical properties, the heterogeneity of the surrounding structure of the rock mass and geometrical parameters of the object can be a combination of finite element method, boundary element method and discrete element method to obtain more authentic solution. The main difficulty of such combination consists in an objective display of the physical nature of processes at boundaries of the connection of elements of the model, which is investigated by a group of finite-difference methods.

2) Most problems require spatial statement for more authentic display of geomechanical processes, which are observed at the present stage of studying the state of rocks mass.

3) More and more works is dedicated to accounting of the heterogeneity of mechanical characteristics of the rock mass and its structure.

4) Is constantly growing the number of works that take into account not only the plastic and rheological properties of rocks, but also the complete diagram of their deformation (pre-limit and transcendent state), which brings the geomechanical model to the real object.

In this sense, there are three main groups of factors that influence the development of application technique of the finite element method in problems of geomechanics:

- 1) description of the physical and mechanical properties of rocks;
- 2) analysis of technological factors of construction and operation of underground structures;
- 3) correction of theories of strength and plasticity, which are used in computer modeling.

As a result, it can be argued that the diversity of obtained solutions shows a sufficient degree of flexibility of the finite element method for modeling of geomechanical processes and development prospects of given method of calculation.

1.5. Conclusions

Geomechanical systems that are studied by mankind over a hundred years, are characterized by a complex and multi-factor structure, which, from the dialectical point of view, cannot be perfectly modeled once and forever. There are always new knowledge about properties and behavior of the rock mass, and the development of industry, including mining, expands the variety of practical problems.

Any geomechanical system differs by irregular structure, geological faults, complex geometry, heterogeneity mechanical and power parameters. Therefore, famous geomechanical models developed and improved gradually towards to more authentic display of processes in the rock mass. Therefore, famous geomechanical

models developed and improved gradually along the path of more authentic display of processes in the rock mass. Analytical methods for calculation of rock pressure occurrences in its dialectical development close to the level when the solution of a complex system of equations is no longer possible to obtain analytically and the need of computers application is predetermined. But speech does not go about the bankruptcy of analytical methods: their relative simplicity determines the feasibility of using in specific operating conditions of underground structures, and methods of modes optimization of interaction of support with the rock mass should be entered in the computer simulation technology. However, there is no doubt in the prospects of development of technologies of computer modeling of geomechanical processes, where the most common method is the finite element method. A great future of this direction is determined by continuous growth of the power of computer hardware and improvement of software. Therefore, modern mining engineer should possess a number of knowledge and techniques relatively to computer modeling of geomechanical processes around underground structures of various purposes.

2. DESCRIPTION OF THE TECHNOLOGY OF COMPUTATIONAL EXPERIMENT

Usually, following main factors during computer simulations in problems of geomechanics with using the finite element method, which certainly need to be clarified to ensure the adequacy of computational models, are highlighted:

- size and shape of the region, which is investigated in the model;
- character of initial conditions;
- character and value of boundary conditions;
- method of the description of roof support elements;
- definition of fluctuation range of physical and mechanical properties of rocks, which are used in the modeling of the rock mass.

Size and shape of the area of solid-state modeling of the rock mass around the mine working effect on the dimension of tasks and on the shape of the objective function, which specifies the distribution of displacements across the computational domain. For workings, which are passed with a small angle of inclination, on the basis of test calculations as the basic form of the computational domain was selected parallelepiped with the following parameters:

- top and bottom faces are oriented parallel to the footwall and spaced from the axis on the length not less than ten radii of the circle circumscribed around the contour of the working;
- side faces oriented along the axis of the working, spaced from its axis on length not less than seven radii of the circle circumscribed around the contour of the working;
- side faces, intersecting the axis of the working, are located from each other at a distance, which is determined by characteristics of the roof support of mine working that can be modeled in the computational experiment.

Parallelepiped as a basic form of the area of calculations allows the most simple to attach initial and boundary conditions, which are in good agreement with the actual loading conditions of the rock mass.

Selection of sizes of the parallelepiped caused by the optimization of the dimension of the basic finite element method matrices, which is important for the problems of geomechanics in the elastic-plastic formulation. The increase in linear sizes could nullify the effect on the result of the calculation of boundary conditions, but at the same time is catastrophically reduced the value of available computational resources. This eventually leads to failure of the computational algorithm.

The second important factor, which entitled to additional analysis, is the determination of parameters of the initial conditions. One of the most significant forms of initial conditions is the function of the load application over time. This function can only be determined on the basis of field observations and laboratory tests for specific geological conditions. Test calculations for different forms of the function have shown its significant influence on the formation of weakening areas in the rock mass. This influence affects both the change in the form of weakening zone and the variations of stress peaks in the rock mass.

Third factor that sensitive affect on the picture of the distribution of stresses in the elastic-plastic formulation of the problem is the choice of the theory of plasticity. It is used to describe the limiting and the transcendent behavior of the material. Analysis of test calculations showed that in the choice of Mises and Tresca criteria for relatively complex stress-strain state the transition threshold of rocks into the transcendental state at 10...12% higher than in the selection of Drucker-Prager criterion. In such a case the growth of contour displacements of the working have flatter view and higher value in certain areas of the contour. However, the computational stability of the computational model, which uses the criterion of von Mises, is much higher than for the criteria of Tresca and Drucker-Prager. Therefore, this indicator should pass optimization phase at each change of the computational model because possible computational errors can lead to failure of the iterative process due to the impossibility of defining the target function on the whole model space.

Boundary conditions during the selection are divided into two main groups - passive and active. In problems of geomechanics as passive boundary conditions are

used symmetry, soft base, joints and rigid base. As active boundary conditions are used pressure (distributed load), the force of gravity and rarely concentrated effort. Passive boundary conditions have the quality of constancy and do not change in the course of calculations. Active, vice versa, can change both the magnitude and character. In particular, during the solution of the problem in the elastic-plastic formulation loading of the computational domain occurs by a linear law from zero to a given unit.

In the case, when mine working traversed at a small angle to the horizontal plane the best combination of boundary conditions are the following:

- on the vertical faces of the parallelepiped is imposed symmetry;
- foundation is rigidly fixed;
- to the top face is applied uniformly distributed load (if the working is located outside the influence of the stoping works) or the load, the distribution of which is set in a complex way (if necessary to take into account the influence of the stoping works).

The use of the boundary condition “soft ground” is considered to be ineffective, especially during the calculations in the elastic-plastic formulation. The complexity of calculations and their duration increases, which reduces the reliability of the final results. On the other hand, the damper effect can be achieved by a simple combination of the rigid base and two or three layers of finite elements, which adjoin to the base and have a higher deformability. This slightly increases the problem dimension, but does not complicate the calculation process.

Widespread use of the boundary condition “symmetry” caused by both the type of geomechanical problems and the high level of flexibility of this boundary condition. An important feature of “symmetry” is a small effect on the shape of the objective function and high speed of leveling of perturbations.

For obtaining the most adequate result in the middle section of the computational domain is necessary to model the working, which contains over the length at least three frames of roof support. During the test calculations have shown that for the spatial model with a relatively small computational grid is critical to

provide high detailed elements of mine working support. The optimal should be considered the approach of accurate modeling of the support elements with minimal geometric size greater than 15 of minimum sizes of finite elements that used in the calculation.

An important factor in modeling of the roof support is determination of conditions of its contact with the surface of the rock mass. Since in most design schemes roof support material has better mechanical properties than the rock mass at the sites of contact of the support and rock appears high gradient of displacement. To increase the adequacy of the calculation scheme in this case it is necessary to use the methodology of change calculation of the contact surface. This not only increases the dimension of the problem (the number of nodes on the contact surface is doubled), but also leads to significant computational cost. Therefore, on the one hand it is necessary to use this approach very careful, and on the other - to apply it where expected large displacements of nodes along the contour of the working.

Determination of the set of physical and mechanical characteristics of the rock mass used in the simulation depends from the type of a problem and the availability of relevant indicators, which were obtained during field and laboratory studies. Analysis of the calculations that carried out for the account of strength characteristics of rocks indicates the following:

- orthotropic significantly affects the distribution of horizontal stress;
- account of full “stress-strain” diagram allows most appropriately display the deformation process of the border rocks;
- account of temperature characteristics have little effect on the field of stresses and strains;
- account of rheological properties of rocks allows to obtain full displacements on the contour of the working in any period of its operation.

In such a way, obtaining of reliable, quantitatively and qualitatively adequate results of computational experiment is only possible with complex account of a wide range of characteristics of the mathematical model. A number of these characteristics

must pass optimization for each series of calculations and ensure the convergence of calculations with a small error.

2.1. Algorithm of computational experiment staging

The process of actualization of the calculation scheme of computational experiment is presented in the form of block diagram (Fig. 2.1) and consists of three main stages, which immediately precede the implementation of a computational experiment:

- 1) The creation of geometrical objects;
- 2) Binding of physical and mechanical properties of materials to model elements;
- 3) Optimization of computational model.

Consider in detail the creation procedure of geometric objects. At first, models of roof support elements and rock layers that will be used in the model of the rock mass are created parallel. Type of roof support elements and their qualitative component in the calculation scheme depend from the choice of the method of solving of the problem, assessment of the longitudinal component of the stress-strain state and degree of required accuracy of results in the border zone of rocks.

The creation process of models of rock layers is based on the choice of global contact conditions between surfaces of various geometric elements of calculation scheme. If the model of the rock mass consists of several model layers it is necessary to make the assembly of the rock mass. If models of rock mass are considered as homogeneous assembly step not needed and the researcher goes directly to the formation of mine working contour.

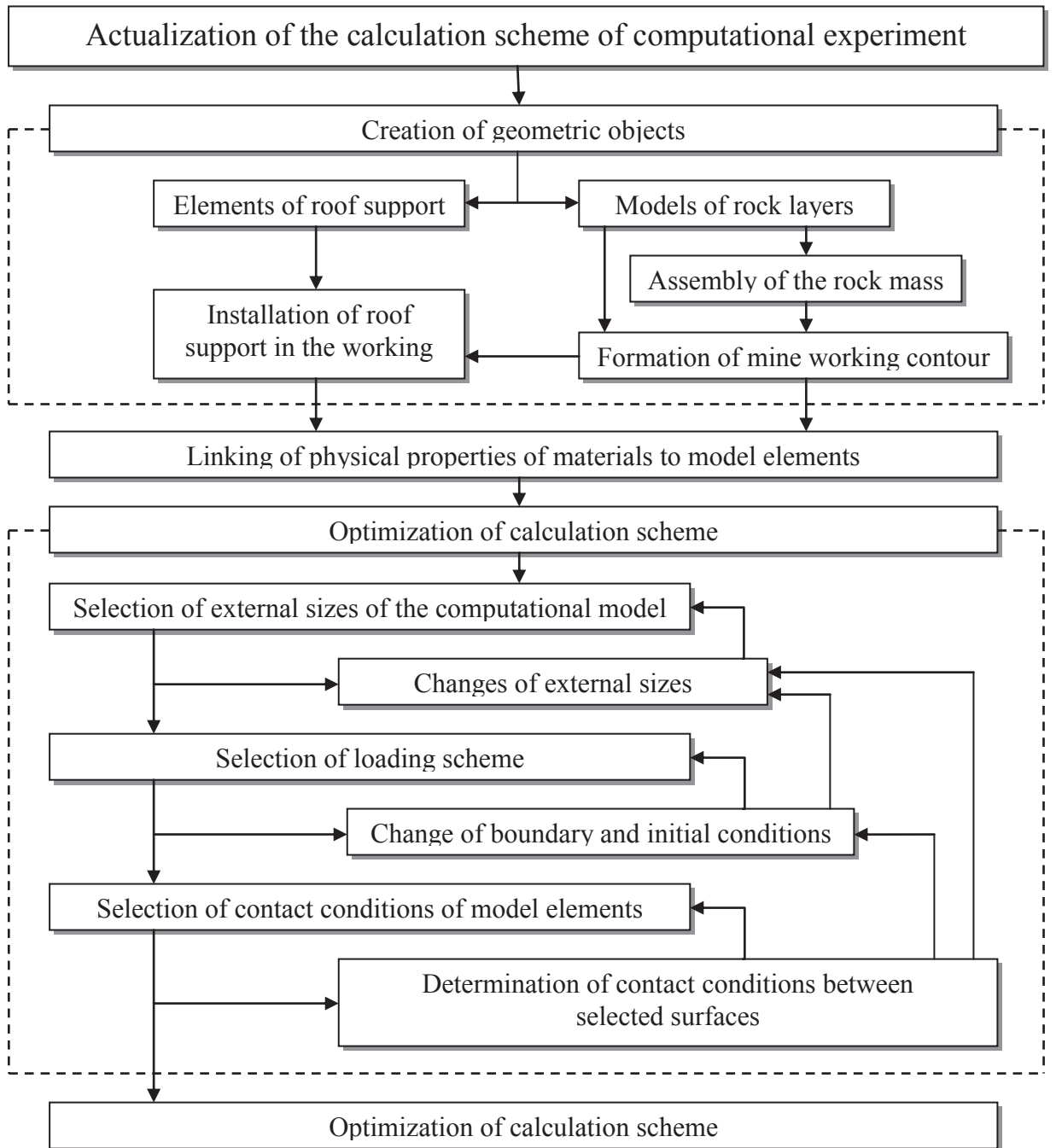


Fig. 2.1. Block diagram of the computational experiment

Formation of mine working contour is agreed with the terms of the installation of the support. At the same time workings and their conjugations are binded to the structure of the rock mass. Necessary to notice that structural and logical errors, which may occur during the formation of mine working contour, lead to the impossibility of proper conjugation of surfaces of the support elements and the rock mass. At the end of the first stage the support elements and the working contour

should be connected. If in calculations are not planned to use elements of the support (as part of the calculation scheme), the blocks “The support elements” and “Installation of the support in mine working” are not performed.

The second stage of actualization of the calculation scheme does not have strongly pronounced structuring. This stage consists of two operations that are constantly repeated. This is the formation of library modules of the description of materials properties and the binding of material properties to specific elements of the calculation scheme. On this stage is determined type of task (linear, nonlinear and the like), on the basis of which will be carried out a computational experiment. If during the optimization of the calculation scheme arises the need to change the type of task, it is necessary to re-develop the library of materials and to bind their properties to elements of the calculation scheme.

The stage of the calculation scheme optimization is the most time-consuming and requires a high level of professional training. It consists of three interconnected steps, each of which implements the constant algorithm of action. The essence of the algorithm consists in the following:

- specific values of parameters are set;
- the test calculation is performed - if the calculation result is satisfactory, the optimization of parameters of the next group is carried out;
- if the result is not satisfactory the additional analysis of the data is performed and on this basis new values of indicators is selected;
- the return to the beginning of the cycle.

The optimization of external sizes of the model is consisted in the choice of heights, widths and depths, at which the influence of boundary conditions on the result of computational experiment are minimized. Loading scheme and conditions of contact of elements of the model are optimized by the criterion of reducing local perturbations, which appear in conjugation regions of the model elements and the surface that receives the external loads.

As a result, created calculation scheme, which is fully or by primary factors satisfies conditions of the problem formulation, is used as the basis of computational

experiment. Any changes in the computational model are within the work on its actualization and have a special character. In the case where the computational experiment is impossible to implement in full volume within the existing calculation scheme, it is regarded as inappropriate and requires re-actualization.

2.2. Sequence of research carrying of the stress-strain state of layered massif around the mine working

At the core of any study there is the following logical structure:

- development and optimization of the computational model of computing experiment;
- the initial testing of computational model for different physical and mechanical performance of properties of its elements;
- the execution of sequence of basic computational experiments;
- the data initial preparation, which consist of the determination of stress and strain tridimensional diagrams of basic elements of the computational model;
- statistical processing and identification of critically important results indicators;
- formation of technological criteria, by which are determined rational indicators of the roof support and roadway maintenance.

Notoriously that under finite element method calculations in problems of geomechanics are used two types of computational models: planar and spatial. The choice is made in the spatial model favor, as it has several advantages:

- allows to consider heterogeneity of the rock mass in two perpendicular and vertical planes;
- most accurately displays contact conditions between the elements of the roof support and the rock mass;
- allows fully realize in calculations all possible mechanical properties of real objects;
- gives the opportunity of adequate modeling of the spatial construction of the roof support.

In such way, the use of spatial calculation model minimizes the opportunity for researchers to do forced errors in the results.

Creation of computational model begins with the study of the rock mass structure in the region of the underground working. Necessary to allocate elements of the rock mass (rock layers), which have typical geometric and mechanical characteristics. For each layer is created three-dimensional model, which has linear dimensions up to hundreds of meters. Then models of layers are assembled in the rock mass model according to its actual structure.

The next step is the modeling of mine working in the rock mass. This operation is quite simple and consists of two steps: the construction of the cross-section of mine working and the arrangement of the axis in the right place of the rock mass model. Then (or parallel) the model of the mine working roof support is created.

The concept of finite element method solutions are usually included approach to the solution of applied problems and the set of numerical methods that allow to get the result in several formulations of the original problem. The choice of the problem formulation must be related to the real object and the computational model.

Comparative analysis of results allows to define the validity of the choice of one or the other type of physical problem. Even more, at this stage of the research in the course of calculations is performed analysis of the significance of individual indicators of the model on results of determination of the stress-strain state system “layered massif - mine working roof support”. This concept includes:

- determination of the best indicators of convergence of numerical methods;
- contact conditions between the elements of the computational model;
- initial conditions of the calculation;
- boundary conditions of the model;
- external size of the computational domain;
- level of detail and type of computational grid.

The finite element method is the so-called grid method, which is characterized by the distribution of the main computational domain into smaller computational

subdomain. They called finite elements. The physical state of the finite element is described by a number of continuous mathematical functions, the type of which for each element can be set arbitrarily. That is why, size and shape of the finite element determines the accuracy and the dimension of calculations. Of course, the construction of computational grid and the determination of its optimal parameters are identified as separate specific subtask of computational experiment. For the problem the choice of the form of the final element as a tetrahedron caused by its sufficient flexibility for the description of variety geometric shapes. In addition, consideration of the finite element as a geometric figure of the second order allows to increase the size of the element without losing calculation accuracy. Computational grid has the variable step, which gives the qualitative description for all elements of the computational model, regardless of their linear sizes. Beyond that, for further analysis of the problem during the construction of the finite element mesh, checkpoints (sensors) were used. At such point during the construction of computational grid is necessarily located the finite element node, which allows to get the “exact” value of stress and strains in the defined point of the computational domain. In these studies, these points are located along the contour of the working and on the external surface of its roof support.

After the construction of the computational grid and the definition of main indicators are proceeded to the conduction of computational experiment. At this stage, the researcher has a mostly passive role. His participation is reduced to monitoring the process of calculation. Also, if necessary, in the calculation configuration of the computer are made changes. For example, during the pursuance of these researches, results are required in additional testing of the model. Necessity of testing is caused by the following factors: intermediate calculations showed the appearance of singular points in areas where they should not be by theoretical concepts; incorrectly selected conditions contact elements of the roof support; analysis of convergence of used numerical methods showed their poor quality.

At the present stage of the development of the computational experiment methodology is accepted to visualize results for the clearness of further analysis.

During the use of the three-dimensional model in the research are necessary to choose typical sections. For example, in the problem, there are three typical sections:

- 1) the vertical section along the excavation, which passes through the axis of its symmetry;
- 2) the vertical cross-section (relative to the mine working) that passes through the axis of symmetry of the frame support section;
- 3) the vertical equidistant cross-section (relative to the mine working) between two adjacent frames.

The most informative section was chosen the last by results of initial testing. For this section are built following four diagrams:

- 1) stresses intensity σ ;
- 2) normal vertical stresses σ_y ;
- 3) normal horizontal stresses σ_x ;
- 4) total displacements U .

Among other possible diagrams (principal stresses, shear stresses, linear displacements and others), these are the most informative and illustrative in terms of applied data analysis. In some cases the diagrams are constructed only for individual elements of the model. This approach is caused by necessity of qualitative analysis of diagrams. Because it is become impossible within single diagram to pick such range of values that would have made available the visualization of stress gradient across the computational domain.

The next step in the preparation of primary data is the construction of ordinary graphs that show the condition or the change of certain mechanical characteristics in one or in several checkpoints. For this purpose sampling of information from the data array is made by a number of the calculated node. Extracted data are collected in summary tables, which are then used to build required charts.

Later the statistical processing of results is carried out and regression equations for calculation of those geomechanical parameters, which interest the researcher, are determined. This information will be needed in the future for the comprehensive assessment of the state of mine working by criteria of the safe operation.

2.3. Features of computational experiment carrying by the finite element method for calculation of the stress-strain state system “massif - support and roof bolting - protective construction”

Determination of stress-strain state of any large system is located under a complex load. The magnitude of this load and its distribution may be changed in space and time. This requires the clear understanding of modeling conditions of the interaction of individual elements of the computational domain. Performance of work formation of such a model is usually done in three steps:

- 1) creation and processing of models of individual and “simple”, in the sense of geometry and load application, elements;
- 2) combination of simple models and conditions of their connections within the common computational domain;
- 3) linkage between external loads, boundary conditions, and the individual elements.

Considering the length and the complexity of processing of such computational models, researchers have resorted to their radical simplification. In this sense, the approach, which allows separate calculations of stress-strain state of mine workings roof support and the rock mass, is implemented. The effectiveness of this approach is sharply decreased with increasing complexity of the interaction between elements. Similar complications are nonlinear behavior of roof support constructions, nonstationary contact forces and consideration of transcendental state of the rock mass. Under such conditions, physically justified solution of the problem is possible only when the most complete description of multiparameter system of stress distribution between elements of roof supports and rock layers is set.

Let's consider complexity and methodology of such a solution for the three-dimensional computational domain, which includes the part of in-seam working outside and inside the zone of its conjunction with the working face and the combination of anchor support and frame-bolt support [28]. The simplest elements of such a settlement system will be anchor, frame of roof support, anchor and frame

connection nodes, bolt-up, backing and model of rock layer in the form of the parallelepiped.

During the research was considered several options of anchors. However, the most actual were the simplest resin-grouted [28] and yielding roof bolts. Because at the present point of Western Donbass mines operation resin-grouted bolt is the most widely used, modeling of this object caused the most interest. This type of anchor has the simplest geometry and the mechanism of interaction with surrounding rocks.

Researches of resin-grouted roof bolts behavior have shown that the active phase of the anchor resistance to the rock pressure occurs after partial weakening of the surrounding rocks. As a result, forces over the entire contour of the mine working are redistributed. Modeling for different geological conditions showed the highest adequacy of the following calculation schemes:

- anchor is represented as tightly mounted steel rod in a hole with the same diameter;
- to the previous scheme is added large diameter supporting washer, which rigidly or through the bolt connection in contact with the anchor;
- similar to the second scheme, but includes an increase in the diameter of the hole up to natural size and filling with solidified polymer composition.

With sufficient simplicity of geometric modeling is difficult to predict the reaction of resin-grouted roof bolt in time and during the choice of elastic-plastic material behavior scheme. This forces during the computational experiment to apply non-static conditions of contact between elements of anchor model and the rock mass model.

Let's consider physics of the interaction between the anchor and the surface of the hole. At the moment of installation of resin-grouted roof bolt between steel, polymer composition and rocks that generate the surface of the hole connections are established on the basis of chemical and molecular effects. Typically, the contact between the steel and the polymer has the more uniform and predictable character. It is determined by high homogeneity of these materials. The contact between the polymer and the rock mass is chaotic in the sense of geometry (fractures and

microcracks on the surface of the hole) and in the sense of materials mechanics (rocks are usually formed from a set of materials with different strength and chemical characteristics). As a result, for the implementation of adequate modeling of operation conditions of the resin-grouted bolt it is necessary to use a wide range of initial conditions and types of contacts [29].

In the simplest geometric modeling scheme of resin-grouted bolt exists one contact surface. For this surface under conditions of high lateral load or significant stiffness of the surrounding rocks, contact is set with continuity observance of the model of “anchor – rock” system. In the case when there is possible to loss contact between rocks and the anchor body, the combined scheme is used. In this scheme part of the surface of the hole model is located in rigid connection with the anchor and another part forms the contact with the anchor in conditions of mutual slippage. If to consider pre-tension of the anchor, in the calculation scheme this condition is realized by applying necessary efforts to normal line at the open anchor end. However, this approach is not fully appropriate relative to the distribution of forces and displacements at the surface of mine working contour. And in the case of significant efforts of anchor pre-tension, it is necessary to proceed to the second geometric modeling scheme.

In the second calculation scheme of resin-grouted roof bolt modeling of pre-tension occurs due to the overlapping of contact conditions for the anchor base plate. There are two basic approaches:

- 1) Anchor is installed into the hole so that the inner face of the base plate is deepened into the rock contour of the working (so called “hot” landing). Depth value determines the magnitude of pre-tension.

- 2) The base plate and of the anchor rod are contacted through the “bolt joint”, force which determines the tension capacity of the anchor. At the same time the contact between the base plate and the rock mass can be nonyielding.

In variants of interaction calculation of anchors and rock mass are faced problems when the shear forces acting on the anchor model become predominant relatively to the longitudinal load. In such cases, the supporting strength of the anchor

is significantly affects strength and deformation characteristics of the polymer composition. As a result, to ensure the adequacy of the calculation is necessary to resort to the third anchor modeling scheme. Due to the use of this scheme displacement the complex modeling of the anchor contour with relatively large deformations in any of the selected directions becomes possible. As a rule, between the anchor and the polymer is selected nonyielding contact, but the contact between the polymer and rocks uses rigid connection or “hot” landing. In conjunction with the experimental selection of mechanical characteristics of the model of the polymer composition such system contact allows to consider condition features of the real rock mass (rock jointing, transversality, scale effect and water intrusion) during the simulation.

From the abovementioned, it becomes clear that the formation of finite element mesh for the general case of resin-grouted roof bolt modeling is non-trivial task. The first problem is common to any element of roof supports. This is the small size of the finite element in the roof support relative to the size of finite elements used to describe the rock mass. In the general case it is considered that the decrease in the linear dimensions of the final element always lead to better quality of the result. This is completely on a par with the reality of relatively simple calculations.

With small size of finite elements describing the geometry of the anchor, they take the form of tetrahedrons. Moreover finite element mesh has strongly pronounced irregular character. This grid can lead to the formation of zones of stress fluctuations. Experience has shown that for the majority of tasks the optimal is the choice of twenty-node finite elements. All these elements have the same geometry. The cross section of the anchor is divided by such finite elements into four equal sectors and the anchor axis of symmetry coincides with the common edge of all four finite elements. Therefore, the entire anchor becomes the set of cylinders consisting of four identical finite elements. Due to the variation of the cylinder height could be changed the density of the finite element mesh, selecting an optimal ratio of the finite elements number and the accuracy of geometry description of the anchor in initial and boundary conditions setting.

Now let's consider the yielding rock bolt. Reasonable description of the behavior of such anchor is only possible with explicit modeling of almost all of its nodes. So the yielding rock bolt model usually consists of following objects:

- bush - simulates the set of rubber bushes that are installed under the pressure between the metal rod of the anchor and the rock mass;
- rod thickened at one end - it is actually the steel anchor, and the thickening is the model of the washer, on which rubber bushes are based;
- base plate - is installed at the mouth of the hole and through the "bolt joint" is contacted with the model of the anchor;
- washer (sometimes is replaced by uniform compression) - closes the free end of the bush and is in contact with anchor through the "bolt joint".

Conditions of contact between the rod and the end of the bush and between the end of the bush and the washer are defined as the stiff connection. Contact between the rod and the inner face of the bush is defined as nonpenetrative and without friction. Contact conditions for the washer coincide with conditions for the resin-grouted roof bolt. Conditions of contact of the external face of the bush and the surface of the hole are selected as a mutual slippage with the specified friction coefficient. In some cases, at a considerable quantity of finite elements, it is advisable to replace the washer to the pressure on the free end of the bush.

The yielding rock bolt constructively provides continuous contact with the surface of the hole. Therefore, to predict and to adequately simulate the initial and boundary conditions for this element of the roof support is not significant problem. This anchor is poorly exposed to shear forces. That is why the anchor location in the computational domain does not affect its model type.

Prediction of the interaction of rubber bush with the hole surface causes the major problems in terms of modeling. A lot of depends on the correct type of relationship of the rubber bush material and rocks that form the contour of the hole. If the selected contact forces will be significantly different from the real, large errors in the performance of calculations may arise. This is connected with the fact that due to the considered conditions of the contact necessary to simulate the behavior of the

anchor in the static and pseudodynamic settlement systems [30]. In simple words, parameters of contact zones determine the moment when takes place the movement of the anchor bush relative to the surface of the hole and occurs the transition of the system to a new equilibrium state.

The only feature of the construction of the finite element mesh for the yielding bolt compared to the resin-grouted bolt is discretization of the rubber bush. Firstly, the bush grid does not fit with the mesh nodes of the anchor rod, because with the high plasticity of the bush material the maximum linear dimension of the finite element must be extremely small to provide a relatively small increment of deformation within a single element. This ensures sufficient stability of calculations in the plastic zone of materials behavior of the computation model. Second, the size of finite elements, describing the surface of the hole, should be commensurate with the finite elements size that describes the rubber bush. In such a way, the determination of nodes relative displacement at the surface of the bush and hole is simplified.

As far as is known, anchors are rarely used individually on most of the coal mines of Ukraine [31]. Basically anchors are operated in conjunction with classical yieldable frame support. Frame support modeling is separate complex task, which can be divided into several stages.

In general cases, researchers are trying to neglect or to simplify the geometry of the yielding frame support. For this purpose are used two main approaches:

- 1) when all structural elements of the support are replaced by a simple geometric figure with a fixed thickness in cross section and an averaged mechanical characteristics (combination of these indicators is based on the forecasted repulse of real support);
- 2) when the roof support is replaced by a uniform pressure over the surface of the mine working contour.

In both cases, the resulting picture of the stress-strain state of the computational domain can be only qualitatively correspond to the elastic state of the real rock mass.

Substitution of the real cross section of the frame by the simple model of a rectangle from one side reduces the dimension of the computational domain and from another site increases the resistance of calculations. But during the calculation with the limit and transcendent state of materials, such simplification significantly affects the growth of the stress on the contour of the working. In such a way, the use of the real cross-sectional geometry of the frame nevertheless increases the computational complexity, but allows more fully describe the factors that influence the stress-strain state of the “massif - roof support” system.

Now consider the influence of simulation quality of the node of yielding frame support on the distribution of stresses and strains in the frame and adjacent rock mass. Application of the yielding node in frame support structure allows substantially raise its operation factors. However, this node, as a factor that influences the stress-strain state system, has complicated physics of process that takes place in real conditions. Design features of this element during the high-precision modeling (Fig. 2.2, a) greatly increases the computational cost and reduces the stability of obtained solutions. Under certain conditions the upper part of the frame can be scrolled relatively to props, which adversely affects the static equilibrium of the entire computational domain. Therefore, the simplified model of the yielding node is applied (Fig. 2.2, b). This model provides continuity of the roof support model with preservation of the remaining geometrical characteristics of the frame support model. However, this modeling technique does not guarantee the possibility of large movements of the frame elements relative to each other in the whole range of considered problems of the geomechanics. When significant movements of the mine working contour are predicted, then the different model of the yielding node (Fig. 2.2, c) is used. In this case, the geometrical authenticity of the external contour of the frame is slightly disrupted. For different models of the roof support, linear deviation does not exceed 50 mm, but now significant linear (up to 400 mm) and radial (up to 20°) movements are allowed by the model [29, 30].

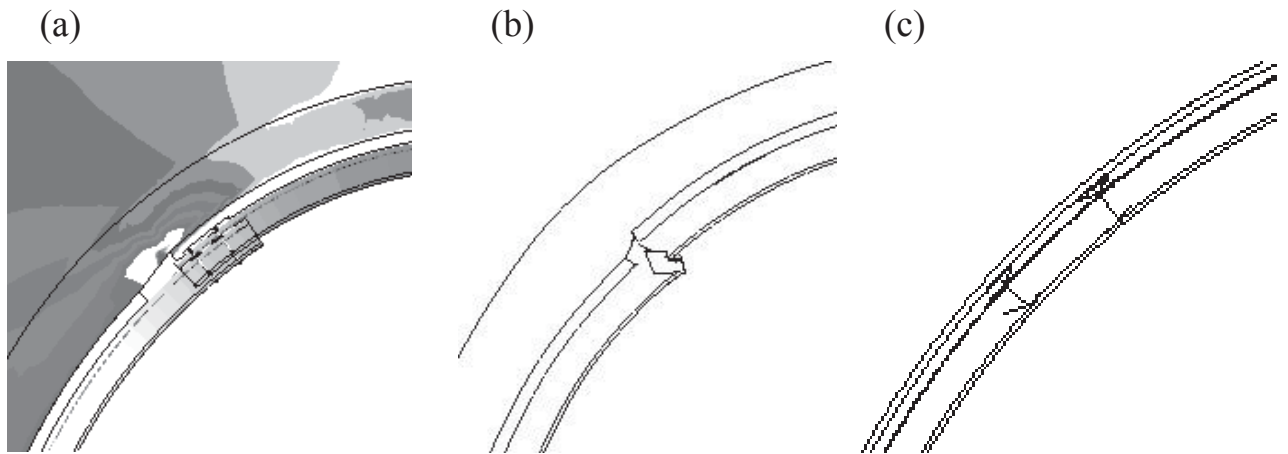


Fig. 2.2. Modeling of the frame support yielding node: a) geometrically accurate model; b) node model that provides its mobility without dynamics with preserving the original contour of the frame; c) node model that provides a large movements without dynamics consideration

Therefore, modeling of frame support yielding is associated with large displacements accountability. In Figure 2.2, b clearly visible movements of the upper part of the frame support, which fell downwards and simultaneously pressed the model of the yielding node in the side prop. As a result, displacement of points of the arch, by using the model of yielding node that shown in Figure 2.2, c, can be up to 330 mm [29]. In turn, this leads to the increase in the zone of the limit state of rocks forming the arch of the mine working.

Frame support elements should be modeled with the low level of detail. Since these objects are entered into the design to ensure the transfer of efforts and perceive external load by the simple scheme [28]. But the neglect of such objects can significantly change the picture of the stress-strain state of the frame and adjacent to working contour rocks. Such elements include reinforced concrete lagging, rock backing, metal coarse-meshed lattice and the like.

Reinforced concrete lagging and backfill can be represented as a single object averaged over mechanical characteristics, the geometry of which corresponds to the geometric characteristics of the described real objects. In most cases, this approach ensures good sufficiency [31]. For options, when it is necessary to divide in the

calculation the lagging and rock backing, the geometry of the concrete blocks have to be described in accordance with the external curvature of the frame. To emulate the behavior of flowing medium, backfill is described by mechanical parameters that correspond to the behavior of the continuous medium with high deformation rates and with using the linear-piecewise law of the strain and stress relation.

Wooden props (relief posts), used to increase the bearing ratio of the frame support in the zone of influence of extraction works, are modeled in the form of cylinders, the radius of the cross section of which corresponds to the actual radius of the prop. In the construction of the finite element mesh is necessary to ensure the presence of 6-10 elements in one cross-sectional of the prop model. The maximum size of these elements should be oriented along the height of the cylinder. Since the object is kept under the influence of axial compressive forces, this approach reduces the probability of the model transition to large transverse displacements. If this effect is persisted, transverse displacements of the side surface of the cylinder along its entire height are limited. In the physical sense, the problem arises with the simulation of wood. In this case the orthotropic material is used, characteristics of which in cross section differ from similar figures along the axis of symmetry of the model.

Hydraulic prop, as the mechanical object, has pronounced dynamic characteristics. In order to escape from the problems that associated with the correct description of these characteristics in the computational model, this prop is replaced by the cylinder with equivalent geometrical and strength characteristics or to the combination of efforts in areas of real prop contact. In the simulation of the object on the basis of large displacements is used the hollow cylinder, the wall thickness of which corresponds to the thickness of the real object and the inner surface is loaded by hydrostatic pressure. Moreover, if is necessary in the calculation to consider pressure discharge in the prop, the internal stress can be defined in the cyclical manner with the real time and without it.

Features of interaction of the working prop with the frame girder and footwall rocks require the special approach in the construction of finite element mesh for these objects. The problem is the occurrence of stress concentration in the models area of

props adjacent to contact surfaces. As a result, it is necessary to decrease the maximum size of the finite element. However, excessive minimization of this indicator is also not acceptable, because normally the contact between props and other objects of the computational domain is rigid. Small size of finite elements can cause fluctuations in the picture of stresses near contact surfaces.

The simplest object in terms of geometrical and physical description is the protective cast strip. Object has the unambiguous modeling scheme, which in most calculations satisfies errors. If we talk about the interaction of the cast strip and other elements of the computational domain, here the main scheme can be considered the rigid connection in all edges of the protective strip model. In cases of possible large displacements and contact with frame support, the contact with the sliding without friction is established.

Description of the models of rock layers as a set of individual geometric elements is quite a trivial task. Geometrically it is a set of parallelepipeds, and mechanically it is averaged characteristics that obtained experimentally or substantiated on the basis of laboratory and field studies. On the other hand, if to consider the rock mass as an aggregate of rock layers, the description of their interaction, within the specific task, becomes applied problem. Depending on the formulation of the problem, the same properties of the rock mass heterogeneity may affect the accuracy of results in varying degrees.

Layering of the rock mass is largely changes the stress distribution around the working contour and rock layers that adjacent to the boundaries. The degree of influence on the stress field for various components of the system may vary from 10 to 270%.

For example, let's consider the stress distribution, which is shown in Figure 2.3. The main reason of the large influence of layering is the difference in the strength of rock layers. Rock layers that forming the bottom and sides of the working have increased stiffness relative to rock layers that form the upper and lower areas of the computational model. This leads to the stress concentration in the range of the geometrical area of the particular layer.

The main feature of the diagram is the lack of symmetry relative to the vertical axis of the in-seam working, in other words the distribution of stress in the sides of the working has different qualitative and quantitative performances. Moreover, the magnitude of this imbalance is dependent on the slope angle and physical properties of rock strata. For different computational models such imbalance can be up to 40%

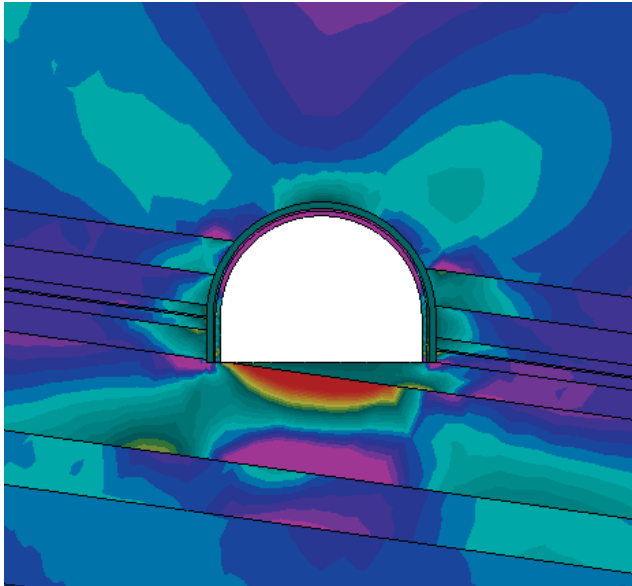


Fig. 2.3. The stress distribution in the thin-layer massif near the in-seam working

in quantitative terms and to 180% in the qualitative. Such indicators for the real in-seam working can be achieved only with consideration of the seam inclination in the computational model, if in natural conditions this angle is more than 3° [31].

Moreover, that rock layers may have different geometries and physical characteristics, to increase the adequacy of obtained results during the modeling, conditions of contacts at boundaries of these layers have to be changed. In the general case there are three types of such contacts. It is rigid contact, contact with

the slippage and contact with the friction force. Use of a particular type of contact may lead to qualitative and quantitative changes in the pattern of stress distribution, which are presented in Figure 2.4.

In the case of rigid contact (Fig. 2.4, a) under horizontal bedding and insignificant range of physical characteristics of rock layers, the pattern of stresses distribution largely coincides with the distribution of stresses in the model, wherein the layering is not considered. But in the case of Figure 2.4, b, the picture of stresses is significantly different from the distribution of stresses in the model without the layering. Such change in qualitative and quantitative indicators of stresses pattern caused by the inclusion in the calculation model of mutual slippage of rock layers.

Now the stress level in the coal seam is higher than in surrounding rocks and its contribution to improving the sustainability of the working increases significantly.

Complexities of the elements interaction modeling of the system “rock mass - supporting roof bolt” are concentrated in interaction areas of anchors and roof

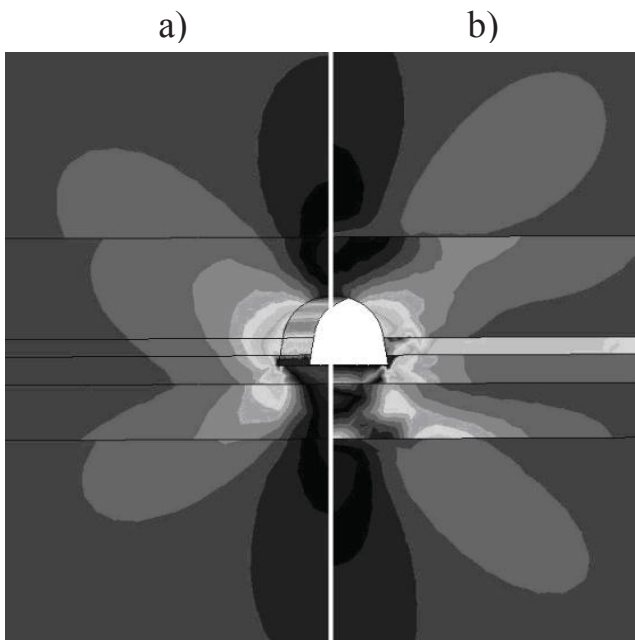


Fig. 2.4. The stresses intensity distribution in the rock mass (a) without slippage on the edge of rock layers; (b) with slippage on the edge of rock layers

support elements. These elements include the coarse-meshed lattice and supporting metal stripes that have several holes for fasteners. These stripes are allowed to combine several anchors that installed between frames of the support, into the single rigid load-carrying structure. The simplest way to unite anchors and supporting metal stripes is rigid contact. Note that the height in cross section of the strip model should be increased relative to the real value.

Contacts between the surface of the supporting strip and the surface of rock mass should be defined as free without friction. This is connected with two factors:

- 1) In the calculation, displacements of individual anchors relative to each other might have different directions and magnitudes, which will cause additional stresses in the strip itself. At the same time, if the strip will be rigidly connected to the rock mass, this will lead to incorrect adjustment of the stress-strain state in the contour of the working.

- 2) When between the strip and the surface of the rock mass will be located coarse-meshed lattice, under real conditions, it won't limit the movement of the supporting strip along the surface of the rock mass.

In the design of the roof support, the coarse-meshed lattice acts as an integration factor that ensures the distribution of efforts between anchors and frames. However, this lattice restricts the free movement of marginal rocks in the working. The magnitude of resistance to such movement in the structure cross-sectional is relatively small. This allows to consider the given structure element as the rigid membrane, which is fixed at points of the contact with anchors and roof support frames. As a result, the coarse-meshed lattice is modeled as the repeating working contour object with the thickness, which is not exceeds the diameter of the lattice rod, and with yielding mechanical properties.

The situation is different in modeling of the frame-bolt support [28]. In this case, the additional challenge is to ensure the competent modeling of the force transfer between the frame prop and the side anchor, which are combined by freely passed through flexible tie bar (wire rope). The rope can freely slide along its entire length relative to side anchors and props of the frame. Then the load, which is perceived by the rope at separate section, can be transferred along it over long distances. This feature of frame-bolt support is indisputable advantages from the standpoint of engineering applications, but in terms of computational modeling is a set of problems.

Let's list these problems: the absence of rope rigid binding - static uncertainty of the system; the complexity of the appropriate description of the rope geometry - the choice of longitudinal axis form of the rope model; the transfer of forces in the anchor contact zones - the solution of the contact problem; consideration of the rope deformation in conditions of occurrence zones of plastic flow in elements of the computational model.

The most appropriate modeling of the rope can be considered following ones:

- 1) the rope is combined with the anchor, the rope body is connected with the free end of the anchor;
- 2) the shape of the longitudinal axis of the rope is selected as the sinusoidal wave with straight lines at function maximum;

3) pre-tension of the rope is modeled by the partial penetration of the rope body into the body of the frame.

But in this version of the modeling is hiding insignificant problem. Since the limitation of longitudinal displacements and small elasticity of the rope model do not allow for different values of displacements of the anchor and frame keep the continuity of the computational model, the adequacy of the model is broken at early stages of the calculation. With relatively small deformations the free end of the anchor is moved into the cavity of the working harder than the frame. As a result, the rope emerges from direct contact with the frame. With the continuation of the elastic-plastic calculation and growth of plastic deformation the rope is pressed into the frame again. However, due to the previously lost contact in the “rope – frame” system, the calculation of forces transfer through the newly formed contact area is become impossible. As a result, this problem is solved in two calculation steps:

- 1) calculation of stress-strain state till the moment of rope tension by the anchor;
- 2) correction of contact parameters between the rope and the frame, and the continuation of the calculation.

All of the above are referred to structural elements of the roof support. However, the functioning of the roof support depends from changes in stress-strain state of the rock mass. And these changes, in its turn, can appear as the result of mining or extinction of the mine working, occurrences of rheological properties of rocks and the like. Fundamental factor for the influence of these changes is become the factors of time.

The result of the time tracking during the calculation of stress and strain in the rock mass is become redistribution of effort that perceived by roof support elements. Also, the geometry changing of workings in time makes adjustments in orientation of the principal stresses areas in the border rock mass. In total, these factors consideration can lead to the radical redistribution of forces on the contact surfaces of the roof support and the rock mass, and can change the conditions of formation of these areas and as a result transform the deformation pattern of the roof support.

Appearance of influence of workings adjacent is also often have the localized in time character. Depending on relative sizes of workings and the duration of the existent, such influence on stress-strain state of the system may have insignificant (for example, conjugation of the haulage gate and the passageway) and highly tangible impact.

Based on the foregoing, the calculation of the stress-strain state in the zone of influence of stoping works for workings with frame-bolt support in mines (such as the Donbass mines) can be performed only on the basis of such factors as:

- constructive frame support pliability;
- large displacement of the rock mass contour;
- stress relaxation in the result of consideration of the rheological properties of rocks;
- mutual influence of mine workings.

In this case, compensation of efforts and geometrical parameters of the computational domain will provide the average and the most probable picture of stress-strain state in the zone of the real object.

2.4. Conclusions

On the basis of considered features of finite element method solutions of geomechanical problems is described general-logic structure of the research methodology of stress-strain state of any system.

Major factors that ensure the adequacy of the geomechanical model were analyzed: size and shape of the computational domain, character and magnitude of initial and boundary conditions, model of the behavior of each main elements of geomechanical system. These factors should undergo optimization for each series of calculations and their proper display should provide the convergence of calculations with a small error.

Was developed the block diagram for the visual representation of the three-stage process of actualization of the calculation model, by which computational experiment is carried out: creation of geometric objects, fixation to them physical and mechanical properties and optimization of the model.

The above factors and stages of the research performance with using the finite element method acquired the certain concretization, where the order of the computational experiment consistently was justified.

3. FEATURES OF ROCKS DISPLACEMENT IN ABOVE-COAL THICKNESS AND LOADING MECHANISM OF THE ROOF SUPPORT IN EXTRACTION DRIFTS

3.1. Mine technical and geotechnical aspects of the reuse of extraction drifts

Research of reserves of profitability improvement of coal mining is inextricably linked with the widespread use of resource-saving technologies of maintaining mine workings. Extraction galleries are distinguished by particularly complex operating conditions. This group of development workings holds key positions in ensuring uninterrupted and highly productive technological process of coal mining. The volume of their construction uniquely determines the main direction of resource saving - reuse with minimization of maintenance costs. The most complex mining and geological conditions of the coal seams extraction are typical for a layered massif of Western Donbass soft rocks, where the maintenance problem of reusable extraction galleries is extremely urgent. Worldwide practice highlights one of priority directions of resource saving at maintaining mine workings - widespread and multi-functional application of the roof bolting as effective tool of involving the rock mass in the work of the resistance of rock pressure manifestation. On coal mines of Ukraine and in particular Western Donbass are widely used technologies of the roof bolt strengthening, especially such type as anchor support [32]. In the zone of influence of coal-face operations intense asymmetric nature of manifestations of rock pressure does not allow to use only roof bolting. Therefore, preserving of the operational status of excavation galleries provide the combination of frame and roof bolt support [33-35], and after mining extraction wall passing in conjunction with the combined security systems [33, 36, 37].

The positive effect of the anchor support on the strengthening of roof rocks is not in doubt, but it will be significantly higher under the complete exclusion of stamping in side rocks and bedrocks of extraction galleries. This process in Western Donbass is called "stamp effect" [30, 31]. The essence of its occurrence is illustrated in Figure 3.1. Outside the zone of influence of coal-face works "stamp effect" is conditioned by two factors:

- 1) the formation in the roof anchor support system (reinforced rock plates of high stiffness) that receives and transmits increased load on underlying rocks in sides of the working;
- 2) placement in sides of the working stronger and harder coal seam in comparison with its immediate roof rocks and bedrocks.

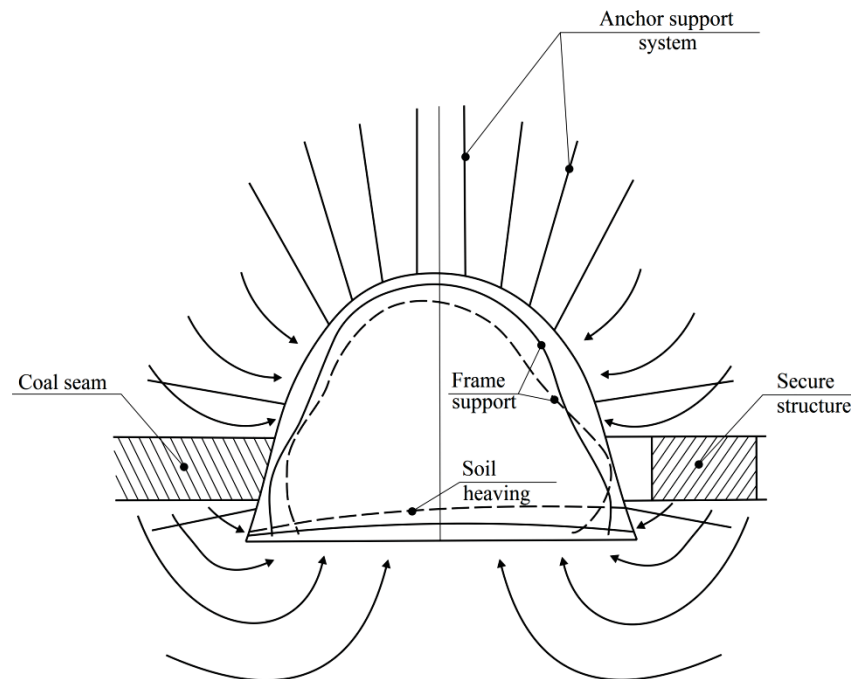


Fig. 3.1. The scheme of “stamp effect” development and the deformation of frame support in front (—) and behind (---) the working face

In the area of bearing pressure ahead of mining extraction wall “stamp effect” is enhanced with the appearance of asymmetry of displacement of the working contour in the direction of the extraction site. After passing mining extraction wall the action of hard coal seam on immediate bedrocks of coal seam is replaced by the load from the secure structure, which promotes displacement of berm rocks in the cavity of extraction gallery and asymmetric occurrence of its soil heaving.

Limitation of “stamp effect” actions and stability increasing of the extraction gallery is accomplished by setting frame-bolt support. The frame and side anchors are connected by yielding mechanical connections that allow to redistribute the load proportionally to the bearing ratio of each element of the frame-bolt system. The essence of the process is shown in Figure 3.2, where is reflected reduction scheme of

bending moment maxima M_i in the frame to the level m_i , when it is exposed the reaction of anchors № 1, 2 through yielding connections. Reduction of bending moment peaks is identically equal to increase of the bearing ratio of the frame by applying reactions of anchors № 1, 2 at certain areas along the height of frame props.

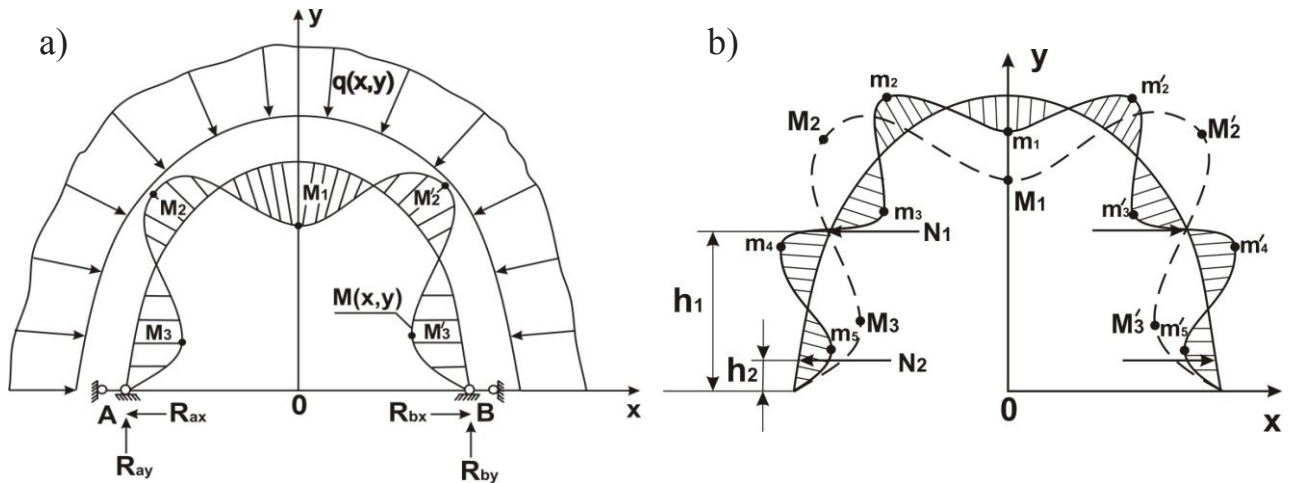


Fig. 3.2. Substantiation of the approach of increasing the bearing ratio of frame support due to anchors: a) the general scheme of the frame loading; b) the reduction scheme of bending moment peaks: (— — —) before tie bar installation; (—) after the installation of flexible tie bars and creation of frame-bolt support as a single yielding load-carrying structure

Mine studies and mathematical modeling [38] indicate the most appropriate areas of the anchors installation in sides of extraction galleries: in the bottom of the coal seam - at the height of 0,2 ... 0,5 m from the footwall; in the roof of the coal seam - at the height of 1,8 ... 2,2 m from the footwall. Presence of yielding connections in the frame-bolt support provides an increase in bearing ratio of the frame by 1,6 ... 2,0 times in the vertical direction and by 2,5 ... 3,5 times in the lateral direction, in addition to the reinforcing effect of anchors.

Significant displacements of the marginal rocks of extraction galleries require coordination of yielding mode of the operation of frames and anchors. Therefore the most efficient is application of constant resistance anchors that provide automatic execution of this condition.

The technology of the phased construction of the fastening system of the growing resistance is provided in accordance with geomechanical regularities the development of rock pressure manifestations in the process of stoping face movement:

- During the drifting of extraction gallery along its length beyond the influence of stoping (section №1) is raised the combined fastening system. The system includes the anchor support in the roof and the frame-bolt support with yielding connections of four side anchors with framed three-unit yielding support of the minimum SVP profile and the maximum step of frames installation. This implements a strategy of resource saving in the counteraction and pressure control mainly due to regulation of parameters of side anchors.

- In the zone of influence of coal-face works (bearing reaction) in front of longwall face (section 2) are installed yielding props of the reinforcement support (hydraulic props, yielding support of special SVP profile and others). If it is necessary, the number of anchors that installed in the immediate roof and bed of the coal seam from the goaf are increased.

- Behind the longwall face (section 3) is raised security structure of variable stiffness to limit the stamping of berm and bedrocks in reusable extraction gallery.

- There are three options for construction technologies of combined security designs:

- 1) support part of the cast strip is raised by the “dry” gunite covering on the basis of pneumatic batch mixer PBM - 2E;

- 2) hydromechanical method in individual plastic containers;

- 3) support part is raised from reinforced concrete blocks in conjunction with organ wooden timbering; fore drift yielding part of the combined security design is raised from wood chocks, support props under beam, etc.

- The cast strip material: anhydrite, phosphogypsum, cement mixtures and others specially designed for these purpose quick-hardening mixtures.

The base mine is selected and the extraction gallery is determined for mine tests of basic schemes of supporting and protection.

Originally is developed, substantiated and tested the base design and technological combined scheme of supporting and protection of the reusable extraction gallery in the conditions of board-and-wall mining system.

Outlined concept of technological regulations of rock pressure control under the maintenance of reusable extraction galleries by resource-saving technologies will provide high-performance work of longwall equipment sets under board-and-wall mining system of coal seams in difficult conditions of coal mines.

3.2. Mine research of displacement development of rock contour of extraction drifts during the stope movement

Researches of manifestations of rock pressure in excavation drifts of Western Donbass mines have a history comparable to the beginning of the development of the region. The increasing complexity of geological conditions of coal seams mining during this period has not changed fundamentally the qualitative picture of the rock pressure rise under the longwall approaching to any fixed section of the excavation drift and distancing from it. These regularities are such that at a certain distance of the section away from the longwall its influence is unessential. During the approaching of the stoping face the growth rate of displacement of rock contour is increased and in the area of bearing pressure is reached maximum (around “longwall-drift” conjugation). After the longwall passing displacement speed is monotonically damped and rock pressure manifestation is stabilized at a certain distance behind the longwall. Another reason for the development of rock pressure perturbation in the considered section of the reusable working is develop of the contiguous extraction district. Moreover, changes in the formation of the load on the support of reusable extraction gallery beyond the influence of the second longwall are related to long-term rheological processes of gob rocks solidifying, reduction of sizes of rock blocks in the roof, creep deformation of roof rocks, side rocks and bedrocks and the like. During the approaching of the longwall of contiguous extraction district, the considering section falls into the zone of its bearing pressure and the process of

displacement of the rock contour of the reusable extraction gallery is significantly activated.

Modern technologies of excavation galleries maintenance in mines of Western Donbass are characterized by wide application of anchor support [34-36]. The support takes the main part load, significantly limiting the lowering of roof rocks and protecting the frame support from excessive overburden pressure. This combination of the anchor support with the installation of frame support was analyzed by measurements of rock contour displacement in excavation galleries. Instrumental observations were carried out at a number of mines in Western Donbass (Mine “Samarskaya”, Mine “Ternovskaya”, Mine “Yubileynaya”), which are different by geological conditions, that provide sufficient objectivity of analysis of the state of excavation galleries in Western Donbass in total.

Regularities in the development of displacements of the roof and bed in the process of the stoping face advance through the seam C_6 of Mine “Samarskaya” for two excavation galleries that are shown in Figure 3.3 and Figure 3.4. Here are clearly seen above-mentioned stages of deformation in the nearby rock mass and loads of roof support during mining of the first excavation section. At 80 ... 100 m distance from the longwall lowering of the roof U_R and raising of the bedrock U_B does not exceed 70 ... 100 mm. They are caused by changes in the stress-strain state of the marginal rocks that are related to extraction gallery construction. On the section (beyond the influence of coal-face works) raising (heaving) of the bedrock is only on 15 ... 25% less than the lowering of the roof with close values of rock-hardness ratios of the immediate roof and bed. Moreover, bedrocks in some areas of the working have the compression strength up to one and a half times higher than immediate roof rocks. Reasons for this phenomenon require detailed understanding in part of the disclosure of the mechanism of contour rocks deformation outside the influence of stoping works. Here is confirmed described in the studies [38] “stamp effect” when harder and therefore holistic coal seam “stamping” (due to the vertical rock pressure) into the cavity of the working weaker (with strongly pronounced plastic and rheological properties) immediate bedrocks.

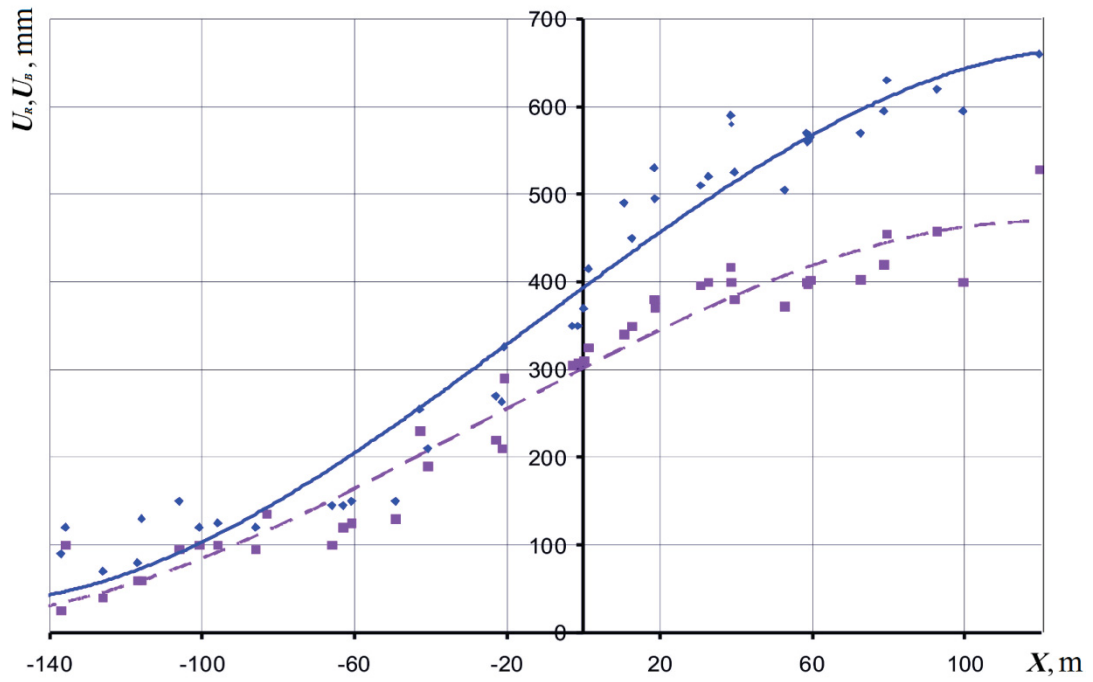


Fig. 3.3. Dependence of displacements of the roof (—) and bed (---) from the distance X to the stoping face of 159 drift of the seam C_6 of Mine “Samarskaya”

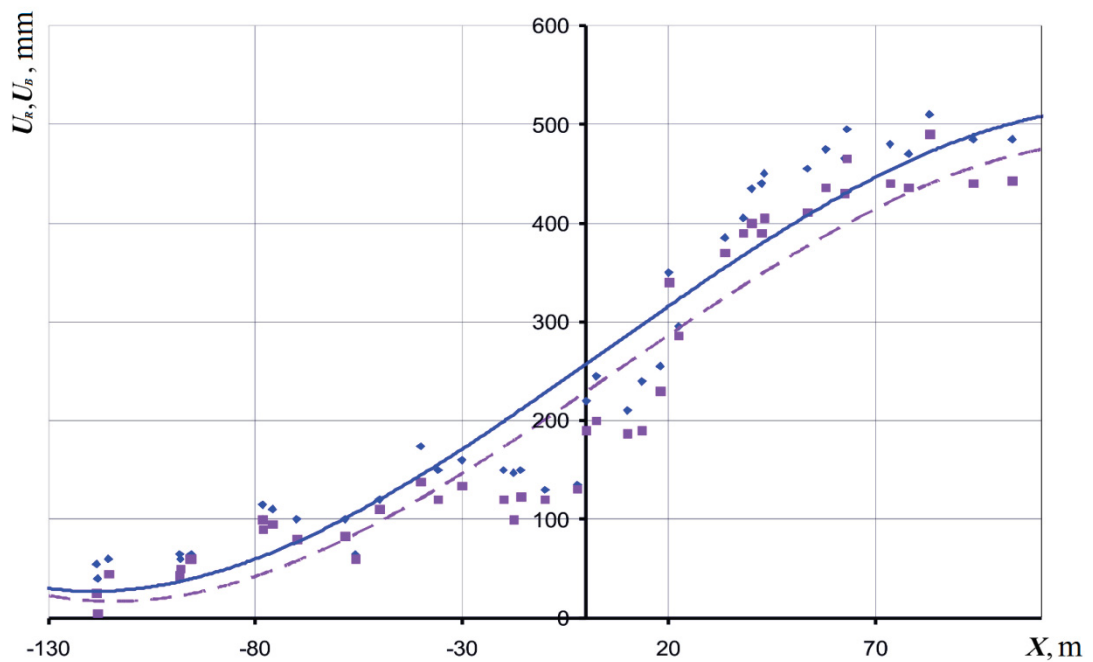


Fig. 3.4. Dependence of displacements of the roof (—) and bed (---) from the distance X to the stoping face of 163 drift of the seam C_6 of Mine “Samarskaya”

Part of instrumental observations of contour displacement development of the extraction gallery on Mine “Ternovskaya” is shown in Figure 3.5 and Figure 3.6.

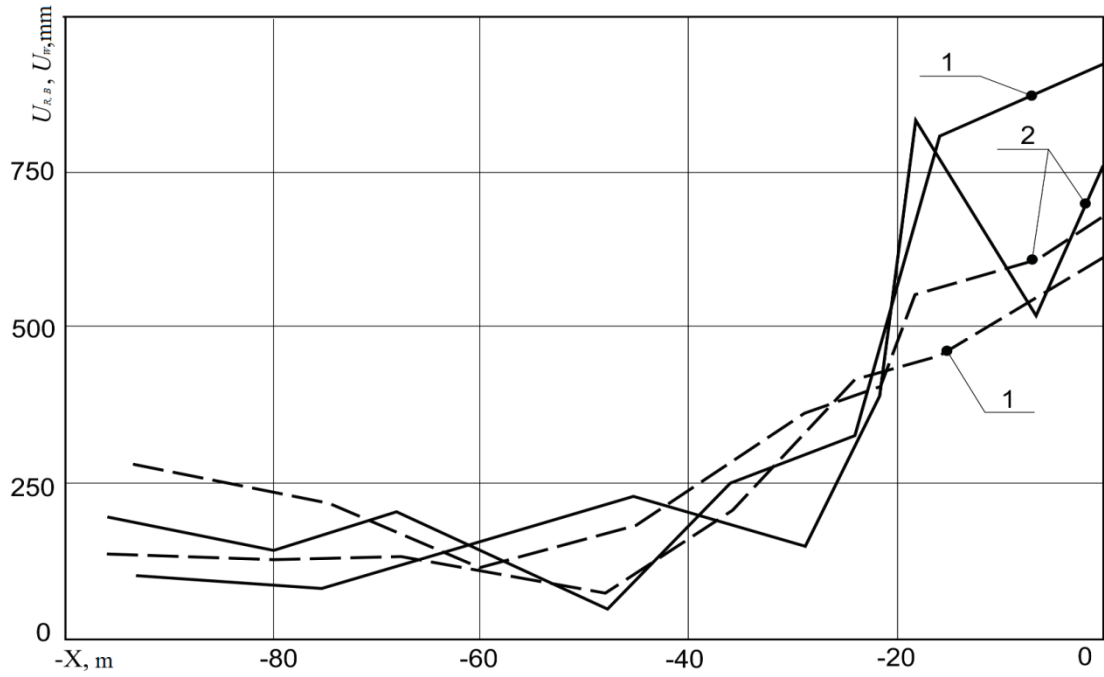


Fig. 3.5. Dependence of convergence of the roof, bed (—) and walls (---) of excavation drifts during the longwall nearing through the seam C_5^V of Mine “Ternovskaya” (543 main drift):

1 - benchmark station №1; 2 - benchmark station №2

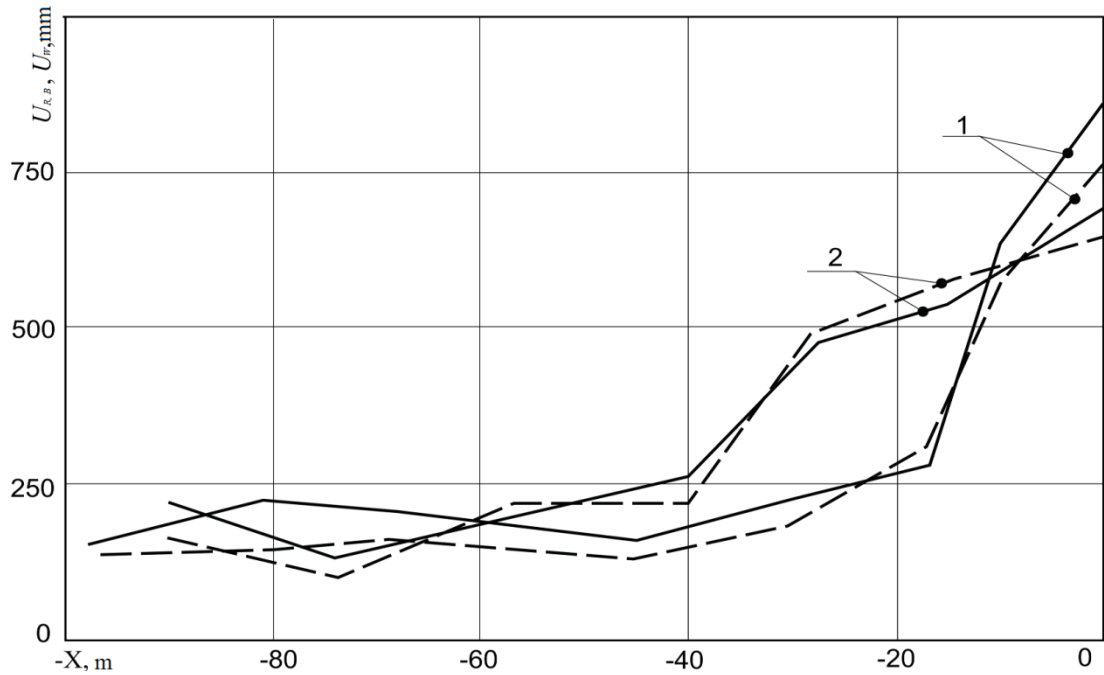


Fig. 3.6. Dependence of convergence of the roof, bed (—) and walls (---) of excavation drifts during the longwall nearing through the seam C_5^V of Mine “Ternovskaya” (543 main drift):

1 - benchmark station №3; 2 - benchmark station №4

The change in mechanical properties of the surrounding roof rock and bedrock towards decreasing of compression resistance and increasing of the difference between the strength of the coal seam C_5^v and the immediate roof and bed should be noted. Therefore, despite the maintenance of stoping works at smaller depth ($H = 230 \dots 380$ m) compared to Mine “Samarskaya”, the convergence of roof and bed, as well as wall of the working U_W are developed exceeding in certain areas such measurements on Mine “Samarskaya”.

Number of development features of displacement, compared with previous studies, is noted. Influence of the longwall starts to be visible at a closer to the face distance from 15 ... 20 m to 30 ... 40 m. In other words, zone of bearing pressure is shortened along the length of the extraction gallery, due to the higher stiffness ratio of the coal seam and weak roof rocks. On the other hand, outside the specified zone of influence of stoping works the ratio of the value of convergence of the roof and bed $U_{R,B}$, as well as walls U_W of the working is of interest. In the current regulations [39, 40] the ratio $\frac{U_W}{U_{R,B}}$ for flat seams is regulated in the range of 0,20 ... 0,39. Measurements of rock contour displacement of excavation galleries show that the ratio $\frac{U_W}{U_{R,B}}$ is not only approaching to unity (i.e., values $U_{R,B}$ and U_W approximately equal to each other), but in some areas of the convergence of walls exceeds the convergence of the roof and bed (see Fig. 3.5 and Fig. 3.6). This fact also requires some explanation in terms of justification of the loading mechanism of the fastening system outside the influence area of stoping works and on this basis the development of measures for limitation of manifestations of the rock pressure in walls of the working. Here is also manifested “stamp effect”, not regarding to soil heaving (as in Mine “Samarskaya”), but in relation to the active displacements of walls. It should also be noted that the magnitude of displacement $U_{R,B}$ and U_W is more significant and reaches values of 200 ... 280 mm outside the influence area of clearing works.

Another confirmation of “stamp effect” manifestation in extraction galleries outside the influence area of stoping works during the application of the anchor

support is the series of observations carried out on Mine “Yubileynaya”, the block №3. Results are shown in Figure 3.7.

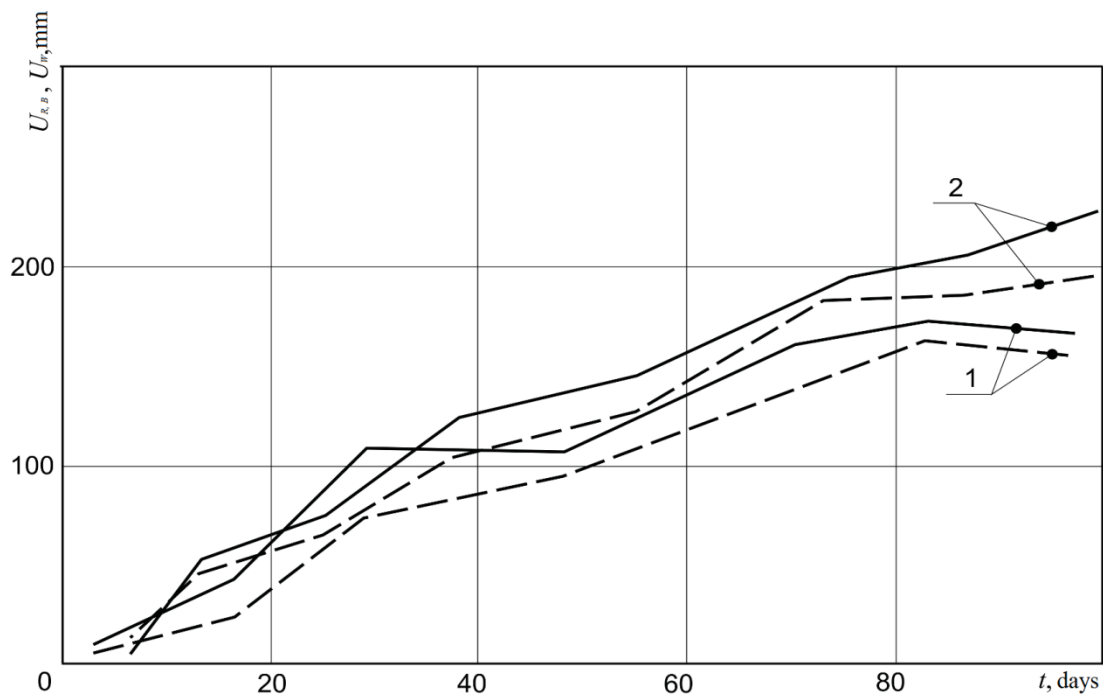


Fig. 3.7. Dependence of development in time of convergence of the roof, bed (—) and walls (---) of the working outside the influence area of stoping works on Mine “Yubileynaya” (139 main drift):
1 - benchmark station №1; 2 - benchmark station №3

The extraction gallery (139 main drift) has not yet got into the zone of influence of stoping works. However, the three-month observations have shown a relatively high value of the ratio $\frac{U_W}{U_{R,B}}$, which increases with growth of observation duration and displacement of rock contour. Thus, if in the first two weeks of measurements $\frac{U_W}{U_{R,B}} = 59 \dots 87\%$, whereas $t \geq 80 \text{ days}$ $\frac{U_W}{U_{R,B}} = 86 \dots 95\%$. It indicates growth of the plastic component in displacements value, whereby “stamp effect” is manifested more intensively.

At the approach of the longwall to the benchmark station in excavation workings the study area falls into the zone of the bearing pressure, where the process of rock contour displacement is significantly activated. For example, in excavation galleries of Mine “Samarskaya” the displacement is increased (at the section of conjugation

with longwall) up to 3.5...4 times in the roof (U_R) and up to 3...3.5 times in bed (U_B). At the same time, the ratio $\frac{U_B}{U_R}$ is sufficiently stable as it approaches longwall and is varied between 67...88% outside the zone of influence of stoping works and in the zone of the bearing pressure until conjugation with the longwall. Moreover, high values of 78...88% are characterized by the section up to 20 m from the conjugation. So, it can be argued that the process of growth of the heaving soil during the approach of the longwall is more intense than the increase of the roof lowering. Consequently, relative to bedrocks “stamp effect” is slightly increased in the zone of the bearing pressure that is quite explainable by activation of displacement of roof rocks.

Similar situation is traced in extraction workings of Mine “Ternovskaya”, where the ratio of convergence of the roof and bed $U_{R,B}$, and the walls U_W was tracked. Here is also observed quite predictable process of increase $U_{R,B}$ and U_W in the zone of the bearing pressure: the parameter of convergence of the roof and bed – in 3...4 times; the parameter of convergence of the walls – in 2,5...3,5 times. The relative stability of value $\frac{U_W}{U_{R,B}}$ at 75...105% is observed, which confirms the presence of the phenomenon of stamping of wall rocks (“stamp effect”) in the zone of bearing pressure.

On Mine “Yubileynaya” benchmark stations have not yet got into the zone of bearing pressure, but the previous two examples are enough to justify such feature of displacement of the rock mass in the vicinity of extraction workings of mines of Western Donbass as formulated “stamp effect”.

After the passage of the longwall the displacement velocity of rock contour of the working decreases, growth of the absolute value of displacements is slowed down and at some distance from the longwall the rock pressure manifestation is stabilized. Their specific development is caused by exceptionally long rheological state changing process of underworked rock mass over large areas. Behind the longwall attention is focused on the stability of berm rock of the extraction gallery, which is protected one way or another, and the process of its soil heaving. Because wall rocks

of the immediate roof are losing rigid base in the form of the coal seam and here on the berm of the working acts concentration of the bearing pressure that is transmitted through a security structure to the berm rocks and bedrocks. Here “stamp effect” manifestations are intensified that can be seen in Figure 3.4, when displacements of the roof rocks and bedrocks are approached to each other, and the ratio of $\frac{U_B}{U_R}$ is 93...95%. For the objectiveness should be noted that in another extraction gallery (see Fig. 3.3) the ratio $\frac{U_B}{U_R}$ is slightly lower (85...94%), but still equal to a significant value.

In such a way, by experimental researches were recorded the presence of active stamping of wall rocks and bedrocks in the working (during the use of the anchor support) on different mines of Western Donbass and in different mining and geological conditions. This phenomenon should be considered in complex with the process of displacement of above-coal rock mass at the face ends, which also has a significant impact on the state of extraction workings.

3.3. Features of the displacement of above-coal strata rocks at longwall ends

Experimental researches of contour displacement of extraction workings, which are supported with the use of combined roof support that composed of the frame and anchor support, have convincingly shown the dominant influence of coal-face works on the state of these development workings and prospects for their reuse. This statement is not new, it is well known and repeatedly confirmed by international practice of coal mining.

It has been established that regardless from the mechanical and technological parameters of the stoped excavation there is no any stable regularities of load changes on sections of the roof support along the length of the longwall and on its face ends. At various coordinates lengths of the extraction pillar maximum load on the section is manifested stochastically along the length of the longwall and it was repeatedly recorded at both the center and the end of the face. Obviously, this is due to changes in the structure of above-coal strata along the length of the longwall, which is extremely difficult to track by geological surveys because of long distances between wells, on the basis of drilling data of which is built mining and geological prognosis.

This proposal has been tested with variation of other parameters that affect the formation of the load on roof support sections. Regular ambiguity of load distribution along the length of the longwall only confirms the conclusion about the dominant influence of the rock mass structure during the almost equal conditions.

The process of above-coal strata displacement affects not only the end sections of the powered support, but also marginal rocks around the extraction working, including its support. By the essence of geomechanical processes a single system is considered, where the indicator of its load is pressure P in the piston cavities of hydraulic props of end sections.

It is generally accepted that the factor of sections loading of powered support is the weight of rocks volume of the immediate roof, which is caving immediately after outcrop. But the main factor is the lowering of rock layers of the main roof, where the load is directly connected with the length of hovering over a section rock cantilevers. This statement is also applied to end sections, but here it is necessary to consider rock cantilevers in the three-dimensional representation in the form of rock slabs with non-rigid fixturing in the depths of the coal seam and with the resting contact upon sides of marginal rocks of extraction workings. In this regard, it is of interest experimental studies of the formation of rock slabs in the main roof on the face ends of the longwall in order to use the results in the developing mechanisms of the formation of the load on the extraction working support in the area of the conjugation with the longwall and behind of it.

In order to perform these researches were selected five sections with a significantly different structure of above-coal rock mass by height in the roof of 11...13 m. Rock mass are changed from mainly thin-bedded to predominantly coarse-grained structures that are mostly represented by mudstones and siltstones. In this case, the sufficient stability of the pressure P in hydraulic props during the movement of the longwall under predominantly thin-bedded structure of roof rocks is clearly tracked. This indicates that the process of the collapse of rock layers along the height occurs stepwise and uniformly without manifestations of perturbations of rock pressure, which usually precede the so-called main roof caving. In this regard, on the

face ends of the longwall is logical to consider the rock slab (for each layer), which in cross-section of the extraction working on border with the goaf will also be collapsed without hanging of the elongated consoles immediately in marginal rocks of the working. Under the coarse-grained structure of roof rocks, despite the high strength characteristics of mudstones and siltstones, perturbations of the pressure are periodically observed in hydraulic props of face ends. This indicates on the hanging of rock consoles with their subsequent periodic collapse. It is logical to assume that on the border of the longwall and the extraction working (in its cross-section) is also formed the plate with overhang in the mined-out space on certain length. Recorded excess of the medium pressure in hydraulic props of end sections is up to 20 ... 25%. However, the overhanging length of rock slabs by the thickness of the main roof is able to substantially increase the load on the support of the extraction working and the security construction raised right after the longwall, which reduces the stability of the berm and intensifies heaving of bedrocks.

Experimentally was established that within a single extraction area the formation process of the load on the maintenance system of the extraction working may differ significantly. It is caused by the change in the structure of rocks of above-coal strata. Noted variations of rock pressure manifestations must be reflected in the mechanism of the formation of the load on the roof support and in the subsequent stage of geomechanical processes modeling for the more reliable prediction of the extraction working sustainability and the substantiation of measures to ensure its reuse.

Another established feature of rock pressure manifestations at face ends of the longwall is the independence of the load on the powered support section from the development depth H . Change in the depth from $H = 400\text{ m}$ to $H = 340\text{ m}$ is almost not affected the value of the pressure in hydraulic props of end sections. Data are intentionally given for the predominantly coarse-grained structure when the effect of the main roof caving is manifested in order to reflect the influence of the depth as much as possible. However, a greater fluctuation of P is caused by the location of end sections regarding to the section of the extraction working, where the caving of the main roof occurs. Noninteraction between parameters P and H indicates that the

formation of the load on end sections is connected with a certain volume of main roof rocks, which is associated with the above-coal strata structure and a number of other parameters. Can be assumed that the volume of deformable unstable rocks, which creates load on the fastening and security systems, also more particularly concerned with the structure of the above-coal strata and other parameters, and poorly related with the development depth. Explanation of this phenomenon lies in specific properties of rocks of Western Donbass. These rocks have low strength and deformation characteristics, weak coupling between layers and strongly pronounced rheological properties. Easily deformed rock mass is inclined at the greatest degree to the formation of the dome similarity of the natural equilibrium by prof. M. M. Protodyakonov. Well known that the volume of such a dome is not dependent from the development depth.

Technological parameters of extraction works, such as the average daily speed V_a of the face advance, the time t of its stoppage (down-time) and the distance δ of end sections from the wall face, play a significant role in the formation no of the load on the longwall powered support. These regularities are such that restrict or expand the dome of roof rocks, which forms the load on end sections. Similar processes affect neighboured roof rocks above the extraction working.

Therefore, the mechanism of the load formation on the fastening and security systems of extraction workings should consider mentioned technological parameters of coal-face works.

Summarizing results of the analysis of experimental researches in extraction workings and face ends of the longwall could be argued that the database of features of rock pressure manifestations in mines of Western Donbass was created. The given base is the foundation for the development of the formation mechanism of the load on the fastening and security constructions and for a further research on the development of recommendations for extraction workings reuse.

3.4. Conclusions

Basic features of the geomechanical displacement of the above-coal strata and the loading mechanism of the fastening and security systems of reusable extraction

workings in the conditions of coal seams mining in Western Donbass were disclosed. Main conclusions and directions of further investigations to achieve the goal - ensuring the resource saving conditions for the reuse of extraction workings, are as follows:

1) The experimental investigation of the displacement processes of the coal bed in extraction workings and face ends have formulated the updated database of features of rock pressure manifestations in Western Donbass mines. The database is the basis for the created loading mechanism of the fastening and security constructions for the application of resource-saving technologies of extraction workings maintenance during their reuse.

2) The effectiveness of extraction workings reuse is determined by the selection of parameters of their maintenance. Four stages of rock pressure manifestations during the maintenance were identified: outside the influence area of stoping works; in the area of the bearing pressure in front of the first longwall; behind the stoping face, including the area of rock pressure stabilization; in the zone of stoping works influence of the second longwall until the extinction of the extraction working.

3) Experimental studies and multiparametric modeling of geomechanical processes fixed the presence of active stamping of wall rocks and bedrocks of the working ("stamp effect") at different Western Donbass mines in different mining and geological conditions. The mechanism was developed to limit the negative impact of the "stamp effect" as the basis of design and technological solutions of the increase the stability of walls and bed of extraction workings in conjunction with the anchor support.

4) Recommendations about principles of creation geomechanical models of the rock mass displacement and the load of the fastening system in the zone of stoping works influence were formulated. Substantiated that the partial softening of rock layers of above-coal strata leads to an increase in the volume of rocks, which are exposed active deformations. And the separation of these rocks on the system of interacting plates intensifies the process of extrusion of the partially softening

marginal rock mass in walls and bed of the working. The roof lowering influence at stoping face ends on the process of the fastening system loading of extraction workings was proved. Also was established a connection of these processes with a number of technological parameters of stoping works that occur not only in space but also in time.

5) The stepwise improvement algorithm of geomechanical model was substantiated in the direction of the improve reliability of the reflection of the real process of origin and development of discrete displacements in the layered above-coal strata.

6) The regulation was developed on the basis of existing mining researches, results of computational experiments and modeling on equivalent materials. The regulation states that the load on the stoping face support and end sections is formed by the weight of unstable rocks inside the dome, which is limited by surfaces of hovering rock consoles and surfaces of the sign changes of curvature of the rock layers bending during its lowering to the underlying rock. Both contours of the dome (from the face side and mined-out space) are exposed significant influence of technological parameters as speed of working face movement and duration of its stopping. These regularities emphasize once again the need of the rheological problem formulation for the reliable representation of investigated geomechanical processes and the development of effective recommendations for the maintenance of reusable workings.

7) Stabilized area of rock pressure manifestations is characterized by partial leveling of geostatistical anomalies due to the combined action of factors of rheology, solidification and consolidation of previously softening rocks over the mined-out space that facilitates the work of the fastening and security systems. Therefore, it is extremely important to ensure the sustainability of extraction workings in the area of their conjunction with the longwall, which is guarantee the creation of resource saving prospects for the reuse.

4. ANALYSIS OF THE DEFORMATION MECHANISM OF ABOVE-COAL STRATA ROCKS DURING THE STOPPING OF FLAT COAL SEAMS

4.1. Analysis of the deformation process of above-coal rock mass

Specialties of zone I formation mechanism from rock layers of the above-coal strata that broken by vertical cracks in area of tensile stresses and form area of joint-block rock displacements are analyzed [41-43]. On Figure 4.1 shown quality picture of force interaction of first layer of zone, I rock block with thickness m_1^I and uncontrolled collapse zone II of second layer rock block with thickness m_2^I . On the basis of classical representations, deformation properties of rock layer do not allow to save wholeness during lowering to value of contact beginning with weakening collapsed rock of zone II. That is why in rock layer in crosscuts of maximum tensile stresses ($C_1 - C_1'$) and ($D_1 - D_1'$) appear normal tension cracks to stratification from the action of stretching horizontal stresses by means of low resistance of rock.

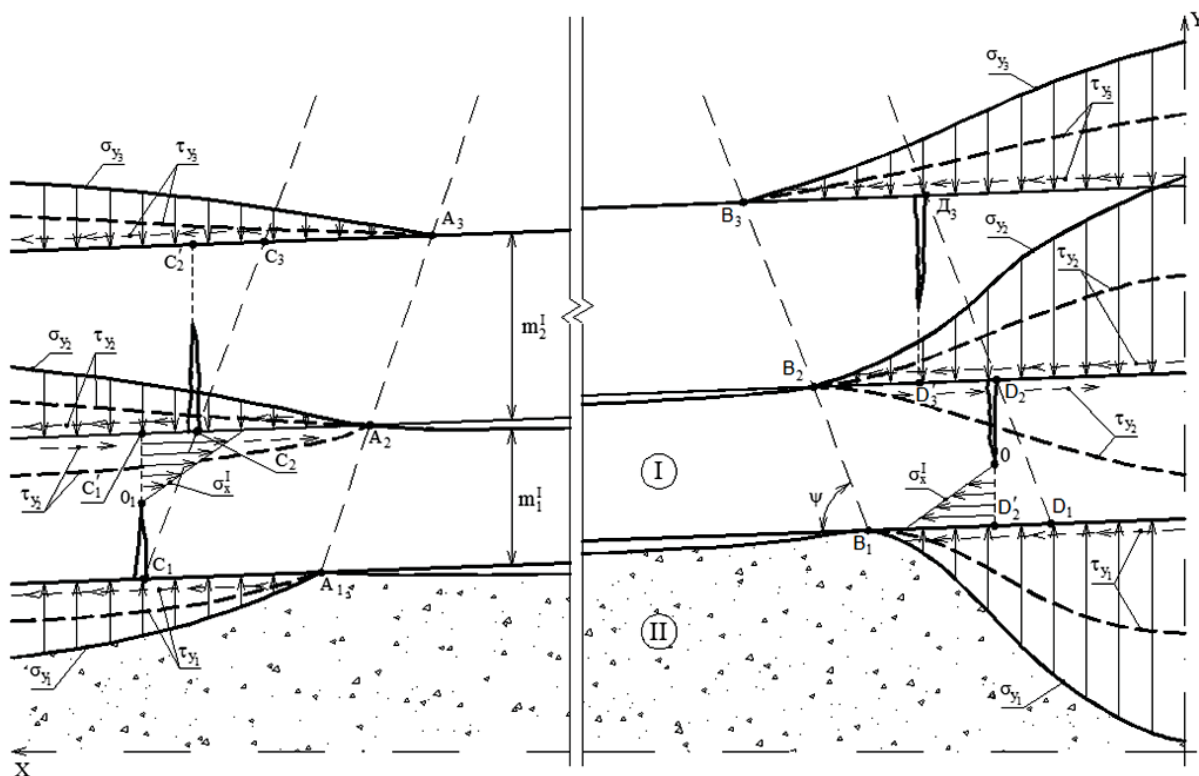


Fig. 4.1. Deformation mechanism of cross joint-block system of zone I rocks

In other (on rock layer thickness m_1^I) part of crosscut, compression stresses continue acting σ_1^I be means of which rock block saves resistance to the above-coal

strata displacement. During volume of rock breakage near considered crosscut it saves permanent resistance to rock pressure. Formed quasiplastic joint allows to displace for rock block on axis Y on necessary value. Therefore, from rock blocks form cross-hinged system. Its value depends upon local stresses field near quasiplastic joint.

Formation mechanism and quasiplastic joint work are following (Fig. 4.2). During zone I rock layer deflection with thickness m_1^I , its neutral axis X_n displaces to the side of tensile stresses actions σ_1^I by reason of existing elevation of deformation module on compression over a module of tension. On the unit $0 - Y_2$ stresses σ_x^I are absent because of tension crack formation; on the unit $0 - Y_1$ acting tension stresses σ_x^I that are apportioned according to linear law [44] on axis Y with maximum σ_{x1}^I on the rock seam surface Y_1 (diagram 1). On rock layer contact in vertical direction, acting stresses σ_{y1}^I from the side of collapsed rock of zone II. If we use strength theory of Kulon-Mor [45], maximum value of stresses σ_{x1}^I will be equal to

$$\sigma_{x1}^I = \sigma_{comp}^I + \frac{1+\sin\varphi^I}{1-\sin\varphi^I} \sigma_{y1}^I, \quad (4.1)$$

where σ_{comp}^I – ultimate resistance to compression of first rock layer of zone I;
 φ^I – internal friction angle of rock.

At further deformation of rock layer under the influence of rock pressure happens its breakage in the area of axis Y_1 . At first approximation of breakage process can be represented as chip prism P_1 [45], which under the influence of stresses σ_{x1}^I aspires to displace in the side of collapsed rock. Rock of zone II resists to this displacement and stresses on a contact with chip prism σ_{y1}^I increase (diagram 2) that leads to increasing of rock resistance to stresses σ_{x1}^I on condition (4.1). Simultaneously rock block of layer turns regardless to 0 point on angle that exceeds its possibilities at elastic-plastic deformations. At this, chip process of prisms will develop to the moment when in the other end of worked-out area lay on the collapsed

rock of zone II. Because of rock prisms displacements, stresses σ_{y1}^I exceeded and increased stresses σ_{x1}^I , but diagram σ_x^I form approximates to square shape (diagram 3). Coordinate Y_p characterizes moment of joint resistance and is determined by rock block turn regardless to 0 point before moment of its contact with rock of zone II.

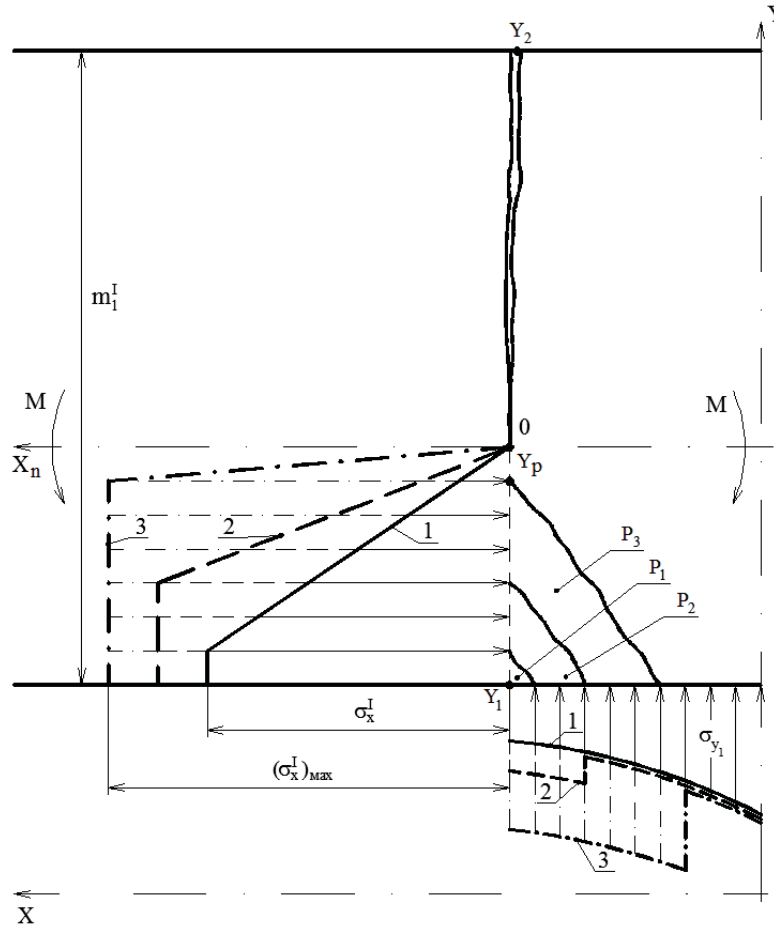


Fig. 4.2. Formation mechanism of quasiplastic joint on the border of rock block of zone I

Second parameter that characterized resistance moment M of quasiplastic joint is maximum of horizontal stresses σ_{x1}^I . It value can be determined by methodology [45] with taking into account conformity displacements of chip prisms P_i and contacting rock of zone II.

Second specialty of zone II is local contact of rock layers with each other with formation in central parts of block span of cavities between adjacent rock layers.

Local contact is interaction of first rock block with collapsed rock of zone II and interaction of all (on thickness of zone I) rock blocks between each other.

It is well known that formation and opening of normal cracks to stratification in area of tensile stresses happens under the angle of full displacements Ψ . Its value for flat seams of Donbass estimates in $70-75^\circ$ (on Fig. 4.1 shown by dotted graphs $C_1 - C_2 - C_3$ and $D_1 - D_2 - D_3$). Underlie to stratification Ψ is connected with parameters of roof layers jamming on the border of worked-out area.

Roof layers in zone I form cross joint-block system, consequently from rock layer of zone II to further on the zone II on the border with rock lowering zone. At this in each further of rock layer in crack crosscut that correspondence to maximum of tensile moment displaces to the worked-out area. It is caused by non-rigid rock layer jamming.

For more detailed considering of the process was analyzed formation of non-rigid jamming for first joint-block rock layer with thickness m_x^I (see Fig. 4.1). Rock layer is deforms under the influence of rock pressure from point B_1 has reaction σ_{y1} from the side of collapsed rock of zone II. This part of diagram reaction σ_{y1} declines to the side of worked-out area because of increased deformation of collapsed rock of zone II. Total action of stresses σ_{y1} create resistance moment that directed on increasing of rock layer steadiness and displace maximum of resistance moment to the side of worked-out area.

Second rock layer with thickness m_2^I during deformation with first one has same geomechanical processes. From the right side of point D_2 with help of combined vertical displacement of first and second layers acting normal stresses σ_{y2} . From the left side of point D_2 because of console turn of first layer, contact of two layers has to disappear and on the line $D_2^I - D_2$ in second block will form quasiplastic joint. However, mine research [43, 46] and finite elements modeling show that in second rock layer quasiplastic joint appears on the line $D_3^I - D_3$ to the side of worked-out area. From one side, turn of first layer rock block is performed not near point D_2 , but bear 0 point on neutral axis of layer (see Fig. 4.1 and 4.2). At this some part of surface of first rock layer on the left side from D_2 will displace up. From the other side, during formation of cross joint-block system in first rock layer happens load of

rock pressure which contributes to displacement of first rock layer surface towards deforming second rock layer.

All enumerated factors allow layers contact on unit $B_2 - D_2$ and normal stresses σ_{y2} action which transform diagram of compression moments in second layer so that maximum of compression moment displaces in the side of worked-out area, where happens formation of quasiplastic joint of second rock layer of zone I.

Analogous geomechanical process happen on other ends of rock blocks of zone I in worked-out area (see Fig. 4.1., lines $C_1 - C_2 - C_3$, $C_1 - C'_1$ and $C_2 - C'_2$). As a result, in zone I forms cross joint-block zone that is characterized by local contact of nearby layers near quasiplastic joints that situated on the height of zone I under the angle of full displacements Ψ . At first approximation cavity borders between layers (lines $A_1 - A_2 - A_3$ and $B_1 - B_2 - B_3$) also situated under the angle Ψ .

Third specialty of zone I of joint-block rock displacement is development of horizontal displacements of rock layers relative to each other. In works [41, 47] during simulate modeling and finite elements method were established regularities of horizontal displacements of the above-coal strata layers: immediate and main roof rocks displace to the side of coal seam, value of displacement decreases during movement to surface and on height $(6 \dots 15)m_y$ of coal seam thicknesses, direction of rock layers movement changes on opposite. It was established that horizontal displacements of first layer (with thickness m_1^I) of zone I to the side of coal seam is more than second one and is equal to $(0,05 \dots 0,30)m_y$ [41, 42, 47]. At such value of displacements of adjacent rock layers relative to each other lose their connection. They interact between each other on stratification by means of friction forces $\tau_{yi}(x)$ that appear because of normal stresses action $\sigma_{yi}(x)$ on area i . Because first layer of zone I displaces to the side of coal seam, from the side of collapsed rock of zone II acting reaction tensile stresses $\tau_{yi}(x)$. Total impact of these stresses $\tau_{yi}(x)$ on unit $B_1 - D'_2$ creates moment towards 0 point of rock block turn, which is directed on stoppage from further displacement to the side of collapsed rock of zone II. Thanks to tensile stresses τ_{y1} create "recreational" moment which increases steadiness of cross

joint-block system of first rock layer and increasing resistance to the above-coal strata movement.

On a contact with first and second rock layers of zone I also act tensile stresses of friction τ_{y2} . Their impact direction on first rock layer is determined by value of horizontal displacement of first rock layer relative to second one. Tensile stresses τ_{y2} act to the side of worked-out area. Their sum along the contact $B_1 - D_1$ length creates moment relative to 0 point, as called “tipping”. Proportion of “tipping” and “recreational” moments depends upon coordinate of 0 point and total values of tensile stresses that acting on lines $B_1 - D'_2$ and $B_1 - D_1$ (see Fig. 4.1).

For second rock layer (with thickness m_2^I) moment of tensile stresses τ_{y2} and τ_{y3} action is only “tipping” because of decrease and change of horizontal stresses displacements direction.

On the border of rock blocks of zone I from the side of worked-out area tensile stresses also create “tipping” and “recreational” moments relative to 0_1 point in first rock layer. In second one and further rock layers moment is only “recreational”.

Considered specialty of tensile stresses action is factor that taking into account contact length of rock blocks of adjacent layers. As a result, quasilastic joints (on height of zone I) situated under the angle Ψ to stratification and dome of zone is closed.

Summarize specialties of rock deformation mechanism of zone I we can make three main conclusions:

- 1) Because of quasilastic formation, rock layers as cross joint-block systems save part of bearing capacity and maintain resistance to rock pressure.
- 2) Formation processes of quasilastic joints, local contacts and cavities between rock layers have to be considered in correlation with taking into account influence of abutment pressure zone and unloading.
- 3) Limited heights of zone I are substantiated by formation of dome, which geometrical parameters appear from results of stress-strain state system calculation. On dome formation process has influence all specialties of zone I deformation:

moment from quasiplastic joint, dragging power by means of turn and unload of underlying rock block, moment from tensile stresses friction action.

4.2. Study of the formation of joint-block area of the displacement in above-coal strata

Analysis of stress-strain state of the above-coal strata rock layers with usage of finite elements method was executed on diagrams of horizontal stresses σ_x that were received at 30 calculation variants with deformation modules of rock layers changing in 10 times (from $0,2 * 10^4$ to $2 * 10^4$ MPa) and their different combination.

Firstly, diagrams σ_x in each rock layer of the above-coal strata illustrate deflection process with maximum of compressive and tensile stresses formation on surfaces of each layer in two zones: abutment pressure in behind stoping face and above worked-out area, where rock layers lowering reaches maximum.

Secondly, maximums of compressive and tensile horizontal stresses σ_x show on coordinates of vertical cracks appearing in rock layer and quasiplastic joints formation, which form joint-block movement zone. In this connection, common features are following:

- maximums σ_x are situated in area of maximums σ_y action and displace to the side of stoping face according to movement to surface in abutment pressure zone;
- above stoping face, changing line of rock layer deflection is clearly expressed, this line is a boundary of abutment pressure zone and has incline to the side of worked-out area;
- in worked-out area line of maximums σ_x position has some incline to the side of stoping face, this is evidences by decreasing of rock blocks span length and cavities width between layers according to movement to surface.

Thirdly, maximums of compressive and tensile horizontal stresses σ_x are far higher on absolute value in more rigid rock layers regardless of their position to coal seam.

Fourthly, rock layers of immediate roof and coal seam in front of stoping face are extremely sensitive to rigid vibrations of coal-bearing strata layers and its composition. It expresses in quantity changing of component σ_x and in quality distribution diagram.

Established total distribution regularities of horizontal stresses σ_x are supposed also significant differences. Maximums of tensile stresses σ_x are equal to 50-100 MPa with changing interval from 21 to 317 MPa. At such level of tensile stresses σ_x in rock layers happens tension cracks, which oriented perpendicularly to bed plane.

Maximums of compressive stresses σ_x are situated on opposite surface of rock layer and displaced towards maximum of tensile stresses σ_x . Value of maximum of compressive stresses σ_x changes from 64 to 299 MPa and is breaking for weak and average hardness rock that forming coal-bearing strata of Donbass. Therefore, in tensile stresses action area also is possible formation of quasiplastic joint. These joints divide rock layers on separate blocks in joint-block movement zone. That is why on maximums σ_x position lines we can destine about regularities of blocks length changing.

With help of laboratory and mine research was established event of rock consoles hanging above the worked-out area of stair form. Line of consoles incline to the side of worked-out area is connected with geometrical place of deflection changing points of rock layers. Therefore, in model forms two groups of rock blocks on axis X :

- above worked-out area from coordinate maximums σ_x to the end of rock layers (group A) deflection changing;
- in abutment pressure zone from coordinates of rock layers deflection changing to maximums σ_x (group B) coordinates.

Angle of inclination Ψ of changing line of rock layers deflection above stoping face and worked-out area more researchers estimate to $\Psi = 70 \dots 74^\circ$ on flat seams regardless of structure and hardness of above-coal strata rocks. On researched diagrams of horizontal stresses σ_x geometrical place of changing deflection sign can be established on axis X changing of horizontal stresses on the same surface of rock layer

zone. It was established that angle Ψ is characterized by enough stability: regardless of deformation modules of coal-bearing strata correlation, value Ψ within 72 to 76° . Therefore, we can accept angle of rock hanging consoles position above worked-out area equal to 74° .

Majority of researches stated that process of foliated formation of joint-block movement zone determines load formation on support of stoping face. That is why it is necessary to estimate sizes of blocks and regularities of their connection with deformation modules of coal-bearing strata correlation.

Analysis of similarities of distribution diagrams of horizontal stresses σ_x allowed to form scheme (Fig. 4.3) to calculation of first group length a_i of blocks A above worked-out area (length of first rock layer block a_i) and length b_i of second group B in abutment pressure zone near the longwall. Sizes of all blocks in joint-block movement zone would be known if would be established lengths a_i and b_i of blocks in first rock layer m_1^I and angles Ψ_1 and Ψ_2 .

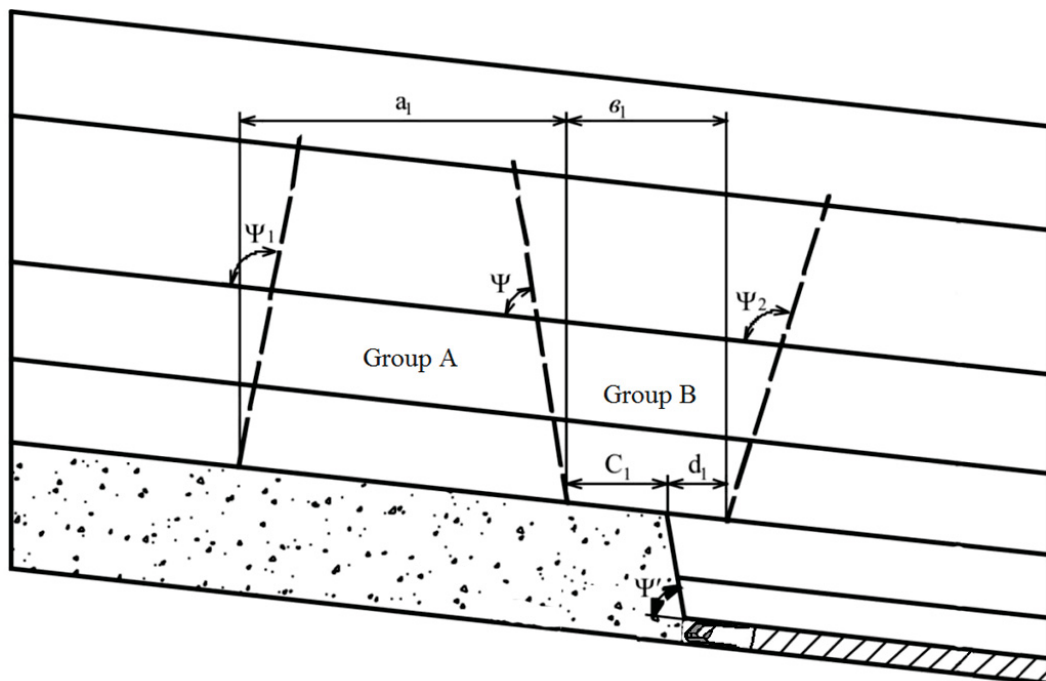


Fig. 4.3. Scheme to length determination of blocks in joint-block movement zone of above-coal strata rocks near stoping face

Therefore, task of regularities revelation of parameters changing from rock layers deformation modules correlation of coal-bearing strata is set.

Analysis of zone block sizes was executed beginning from angle Ψ_1 (see Fig. 4.3), which is their boundary from the side of worked-out area. Increasing of angle Ψ_1 is determined by closer position to stoping face of maximums σ_x that is connected with more intensive deflection of layers m_2^I and m_3^I , when both rock layers in immediate roof ($\Psi_1 = 103^\circ$) and in first layer of main roof $\Psi_1 = 105^\circ$ have lower rigidity. Thus, top m_2^I and m_3^I have layers that are more rigid during deformation in worked-out area and have less reaction from the side of lower ($m_1^I, m_1^{II}, m_2^{II}$) yielding layers. Load on rigid layers increases and they contorted more intensive that displaces maximum of resistance moment closer to the place of rigid rock layer binding.

Variants of uniform (on deformation modulus value) addition of above- coal strata layers have angle Ψ_1 which is almost perpendicular to bedding plane. For some calculation variants angle Ψ_1 exceeds 90° , which is connected with position of second layer m_2^{II} in immediate roof of lower rigidity. It decreases resistance to deflection of layers of main roof and maximum m_2^{II} displaces to the side of longwall. When in immediate roof layered second rock layer of main roof and decrease intensity of deflection and maximum σ_x displaces to the side of worked-out area. When in immediate roof possessed only rigid layers, their increased reaction on layers of main roof increases resistance to deflection and maximums continue to displace in the side of worked-out area ($\Psi_1 = 80 \dots 81^\circ$).

Further research on different depths of coal seam mining shown practically stable values Ψ . Angle Ψ_2 characterizes position of maximums σ_x on height of the above-coal strata of abutment pressure zone. It hasn't connection with rock layers deformation modules of correlation. For 30 calculated variants at height $H = 400 \text{ m}$ angle Ψ_2 has enough narrow fluctuations ranges from 98 to 104° without presence of steady tendency of changings at different parameters $E_i^{I,II}$. I connected with well-known factors of stable maximums displacement of abutment pressure to the side of coal seam due to surface. That is why accepted constancy of angle with average value of $\Psi_2 = 101^\circ$.

Diagrams analysis σ_x shown that (Fig. 4.3, group A, first rock layer of main roof) main influence is caused by correlation of main roof rock layers rigidity. At increased rigidity of two overlying layers (m_2^I and m_3^I) in comparison with first rock layer m_1^I of main roof, first layer m_1^I has increased load and is less subjected to deflection forces. That is why maximum σ_x in first yielding layer m_1^I moved to the side of worked-out area and length a_1/m_y of block in first rock layer increases to $a_1/m_y = 19,1$. At lower rigidity of layers m_2^I and m_3^I they recline on more rigid layer m_1^I . That is why more intensive deflection of main roof first rock layer is observed and maximum σ_x displaces to the side of stoping face: length a_1/m_y decreases up to 13,7.

Length b_1/m_y of second group B block of first rock layer consists of two parts (see Fig. 4.3): console length C_1/m_y that hanging above stopping face and length d_1/m_y of district from jamming point to maximum position σ_x point in abutment pressure zone.

It was established that console C_1/m_y length is practically doesn't depend upon correlation of rock layers deformation modules correlation of coal-bearing strata: average value C_1/m_y is equal to 4,36 with fluctuations range from 3,92 (-10,1%) to 4,67 (+10,7%). It is connected with stability of rock deflection changing angle Ψ , which during formation of quasiplastic joints characterizes angle of main roof rock console hanging near the stopping face.

Therefore, it was substantiated stability of console length C_1/m_y of first rock layer of main roof regardless of layers deformation characteristics of coal-bearing strata that changes within $C_1/m_y = 4 \dots 4,5$.

As mentioned earlier, length b_1/m_y of second block B in rock layer of main roof includes also district d_1/m_y . During analysis was established influence on parameter d_1/m_y of deformation characteristics of rock layers of the above-coal strata. Maximum value $d_1/m_y = 3,9$ observes two variants when in immediate roof layered only yielding layers, and in main only rigid ones. It explained by low bearing

reaction of yielding layers that contributes to intensive deflection of rigid layers of main roof in depth of coal seam.

In opposite variant of bedding in immediate roof only rigid layers, but in main only yielding, maximum σ_x is keen to the point of console jamming, but sometimes goes to $d_1/m_y = -0,3$. According to the results of these research were received following regression equations:

- angle of cracks position in rock block system of group A doesn't depend upon mining depth and is determined by the formula

$$\Psi_1 = 90^\circ + (0,36E_1^I + 0,31E_2^I + 0,23E_3^I + 0,10E_4^I - 0,36E_1^{II} - 0,66E_2^{II}) \cdot 8,6 \cdot 10^{-4}, \text{ degree}; \quad (4.2)$$

- relative length of lower rock block of group A

$$\frac{a_1}{m_y} = 12,6 + 0,75 \cdot 10^{-4} \cdot (E_2^I + E_3^I - E_1^I); \quad (4.3)$$

- relative length of lower rock console of rock blocks of group B

$$\frac{c_1}{m_y} = 4,5; \quad (4.4)$$

- relative length of non-rigid jamming of lower rock block of group B

$$\frac{d_1}{m_y} = 1,5 + \left[1,1E_1^I + 0,82E_2^I + 0,68E_3^I - 1,01E_1^{II} - 1,58E_2^{II} + 0,39(1,5E_1^{II} + 1,5E_2^{II} - E_1^I - E_2^I - E_3^I) \cdot 10^{-4}; \quad (4.5) \right.$$

- angle of full rock displacements is stable at different correlations of geomechanical system parameters and is equal to $\Psi = 74^\circ$;

- angle of cracks position in rock blocks system of group B is stable at different correlations of geomechanical parameters and is equal to $\Psi_2 = 101^\circ$.

Analysis of distribution diagrams similarities of stress field components allowed to substantiate building scheme of joint-block movement zone of above- coal strata layers near the stoping face that differs by separation on different groups according to abutment pressure distribution. Two blocks groups are underlined: above worked-out area and in abutment pressure zone. Tendencies of geomechanical parameters changing of rock blocks groups depending on mining depth and deformation modules correlation of coal-bearing strata are detected.

4.3. Rocks displacements of above-coal strata during coal seams mining in Western Donbass

Coal-bearing massif of Western Donbass is represented by weak lithological differences, hardness coefficient of which in few times lower than coal seams hardness coefficient. Besides, it is observed bottom and roof rocks fracturing with intensity from 1-2 cracks/m to 4-5 cracks/m that increases their calculation resistance to compression in 1,5-2,5 times [40]. Here it is necessary to take into account probability of water content in roof and bottom rocks that decrease calculation resistance to compression of siltstones on 40%, but mudstones on 50% [40]. Together action of given factors leads to situation when resistance to compression of roof and bottom rocks (represented by siltstones and mudstones) in 2-5 times lower than resistance to compression of coal seam.

Marked regularity concerns and correlation of bearing rocks deformation characteristics and coal seam. Here the main mechanical characteristics is deformation modulus, for siltstones is higher, but for mudstones relatively similar with coal seam. However, it is necessary to take into account weakened factors of fracturing and moisture content. Coal is also susceptible to fracturing, but coal seam situated in conditions of hard loading by compression forces, but rocks of immediate and main roof feel deflection deformations with impact of tensile stresses of vertical and horizontal direction. It is well-known that deformation modulus of rock during stretching frequently (to 8-10 times [42]) lower than during compression. Massif

fracturing of decreases its deformational characteristics at least by times [48, 49]. Consequently, on fracturing factor at deflection of roof and bottom rocks deformations decrease their deformation characteristics. Moisture content factor mostly influenced on siltstones and mudstones, inclined to slaking, than on coal seam, that more decrease their deformation characteristics of bottom and roof rocks. It is necessary to say that strongly marked rheological properties of Western Donbass weak rocks decrease longstanding deformation modulus in two times. As a whole impact of marked factors leads to situation of increased deformation of roof and bottom rocks in comparison with coal seam.

Given specialties of correlation of coal seam and roof rocks mechanical properties transform traditional scheme of the above-coal strata movement on flat seams of Western Donbass [41-43, 50-53]. According to the data of existing research, abutment pressure begins in extraction drifts on distance from 20...30 m to 150...200 m in front of stoping face, but maximal concentration of vertical stresses $\sigma_y = (1,5 \dots 8,5)\gamma H$ is possessed on distance $l_1 = 2 \dots 15 \text{ m}$ from stoping face depending on mechanical characteristics of coal-bearing strata and technological parameters of stoping operations (Fig. 4.4). Hard coals of Western Donbass are enough steady to abutment pressure perception. Their resistance to compression $\sigma_{comp} = 30 \dots 45 \text{ MPa}$ frequently exceeds initial vertical stresses $\sigma_y = \gamma H = 5 \dots 13 \text{ MPa}$ at existing depth H of mining operations conducting. In zone of abutment pressure, maximal action predicts seam entirety, because it situated in volume non-uniform stressed condition from tensile horizontal stresses action σ_x and σ_z , at which resistance to vertical stresses increases in 2,0...2,5 times and more. Even with taking into account high values of concentration σ_y in abutment zone next to impossible seam weakening. Substantiated combination of hard non-weakened coal seam and easy-deformed roof rocks lead that maximum and abutment pressure zone displace to stoping face. Mechanism of abutment pressure zone concentration near the longwall (see Fig. 4.4, puncture line) at hard seam and easy-deformed roof explained on Figure

4.5. Schematically underline deflection on neutral axis of some roof layer two variants are considered:

- rigidity of coal seam (deformation modulus E_y is the main value at same thickness) significantly lower than rigid (E_R) roof rock layer;
- rigidity of coal seam significantly higher than rigidity of roof rock layer.

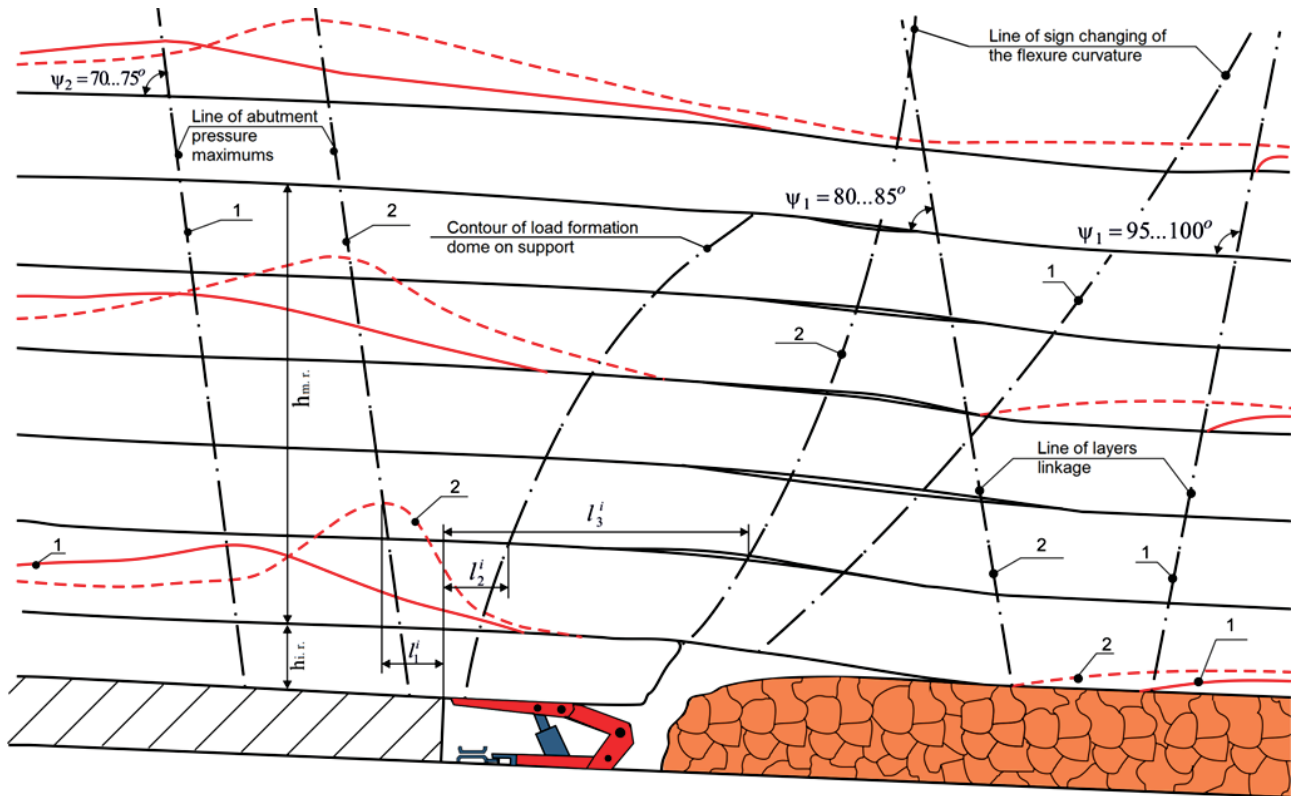
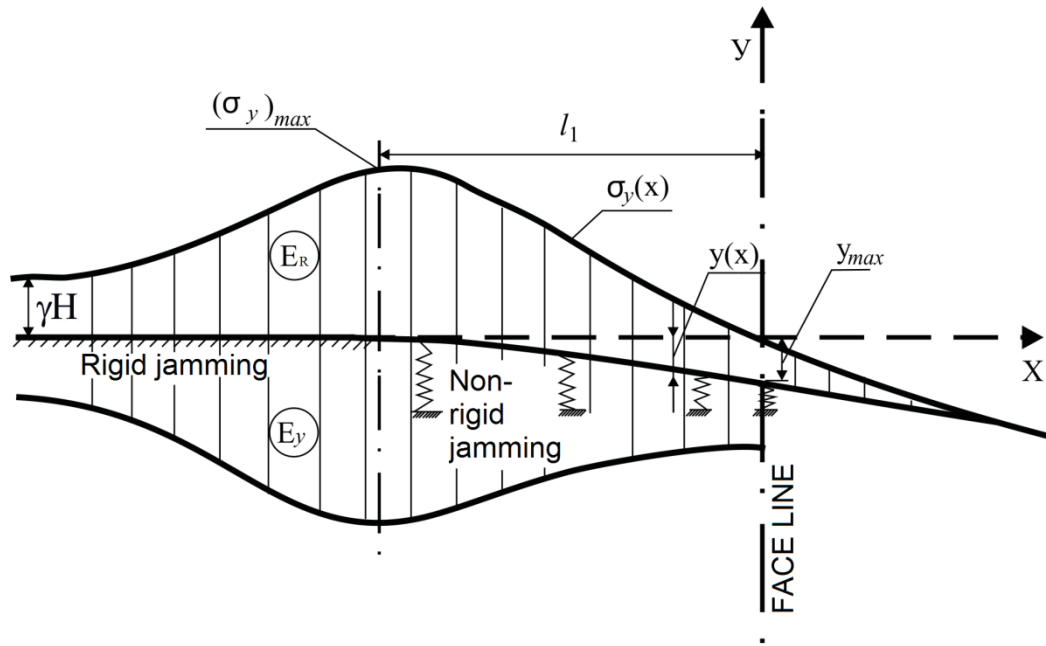


Fig. 4.4. Specialties of movement scheme of the above-coal strata of Western Donbass (line 2) regardless to traditional representations (line 1)

First variant: hanging rock console creates increased load in selvage of a seam where observe significant vertical displacements $y(x)$ because of lower deformation modulus E_y . At the same reason, vertical reaction $\sigma_y(x)$ near the stopping face will be definitely low. Balance weight from hanging rock console also insignificant because of increased rigidity of rock layer and increased rigidity of underlying layers. Load from these layers decreases because of the appearance of cavities on contacts of hanging consoles of rock layers.

a) $E_y < E_R$



b) $E_y > E_R$

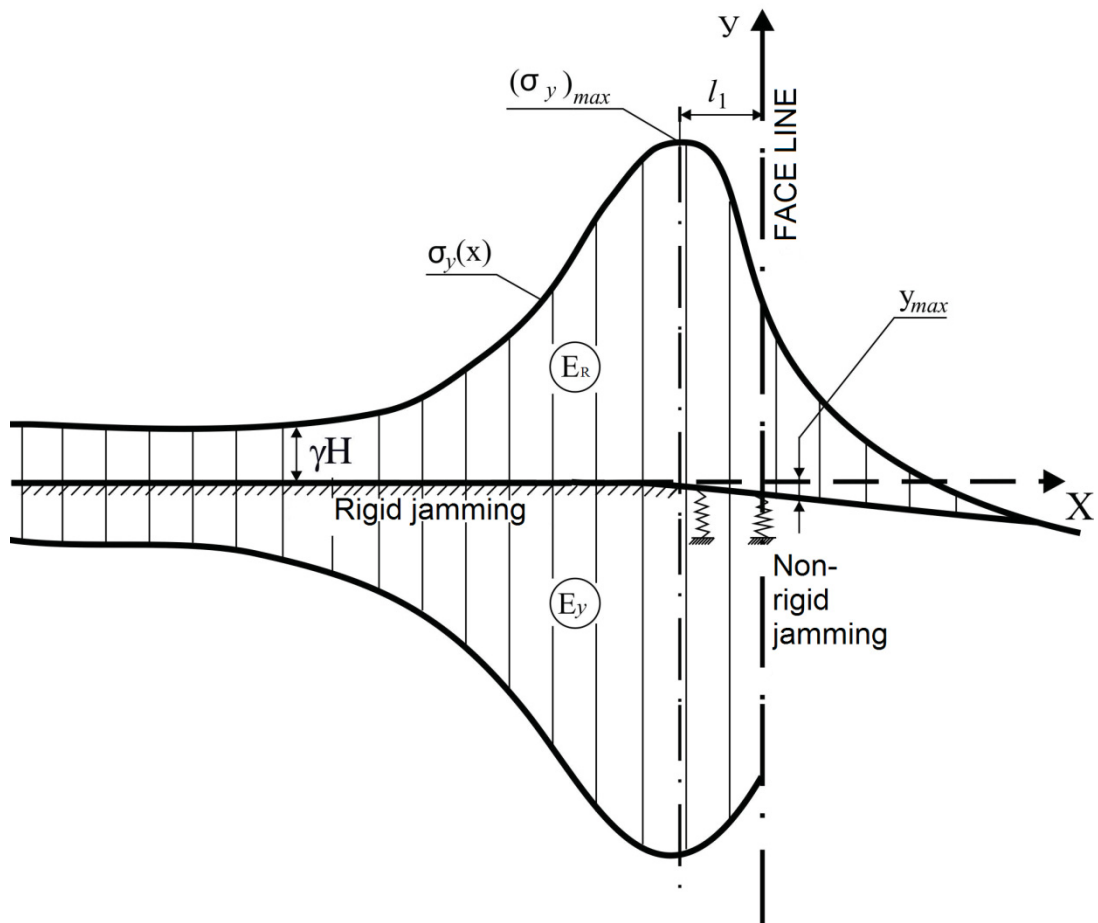


Fig. 4.5. To the mechanism of abutment pressure zone formation at correlations of coal seam rigidity and roof rocks: a) $E_y < E_R$; b) $E_y > E_R$

Yielding properties of coal seam of lower rigidity are forced by partial weakening in near-the-face part that is why mostly low reaction pressure $\sigma_y(x)$ decreases. Lower reaction in selvage of a seam has to be compensated increased reaction in deeper part because of massif balance condition. During sinking into the seam, stress condition transforms into volume non-component, at which deformation modulus and resistance to compression are repeated. At the same unit decreases roof rocks stratification, disappears cavities between interlayers and each of them get (on underlying massif) load from own weight and load from weight of underlying layers by means of contact between them. Therefore, vertical stresses $\sigma_y(x)$ increase to such value when total load in abutment pressure zone, accordance to initial distributed load γH with taking into account of worked-out area. Because abutment pressure zone distributes into the massif on enough great distance (on axis X), so its maximum $(\sigma_y)_{max}$ have to be definitely less because of condition of coal-bearing massif.

Here classical representations about console deflection [44] can be separated into two units. First – non-rigid jamming when abutment part in jamming has some yielding, but upper part just partially limited turn of layer crosscut. Value of jamming hardness increases during movement inside of a seam, because grows its hardness (volume deformation modulus) and decreases weakening. As a result, non-rigid jamming of rock layer gradually transforms into the rigid one. Beginning of second unit is maximum $(\sigma_y)_{max}$ of abutment pressure (see Fig. 4.5), but finish is area where vertical stresses $\sigma_y(x)$ stabilized on the level of initial condition γH .

Second, diametrically opposite variant of abutment pressure zone formation (see Fig. 4.5, b) is characterized by coal seam rigidity that significantly exceeds hardness of roof rock layers. Here in sewage of a seam concentrated increased load $\sigma_y(x)$, because hanging roof consoles lay one into another and have increased deformation, but vertical displacements $y(x)$ of seam are low on the reason of its increased hardness. Stratification in roof above the stoping face partially compensated by intensive lowering of underlying layers, vertical load in rock consoles of near-the-face area increases and is transmitted on sewage of a seam, which cannot distributes

concentration σ_y on width unit l_1 by means of abutment pressure transfer inside. Length l_1 of unit of non-rigid jamming is low and on it distributes man part of abutment pressure that causes high values of maximums $(\sigma_y)_{max}$.

Coal-bearing massif of Western Donbass mainly characterizes by position of abutment pressure increased maximums near face with increased load on powered support (see Fig. 4.4, puncture line). In overlying roof layers, abutment pressure zone changes: maximums $(\sigma_y)_{max}$ decrease; increased σ_y distribute on great distances X to the side of untouched massif in front of stoping face. Line that joins maximums $(\sigma_y)_{max}$ on the above-coal strata thickness inclined to the side of massif on $70...75^\circ$. It was established on the basis of laboratory research, models on equivalent materials and results of computer modeling.

During modeling intensive action of deflection moment in abutment pressure zone was educed. In upper part of crosscut of violent layer develops tensile horizontal stresses σ_x , but in lower part forms concentration of compressive σ_x that mostly exceeds initial non-hydrostatic condition of massif with side thrust $\lambda\gamma H$ efforts. For more detailed consideration of deflection mechanism violent roof rock layer was detached and diagrams of horizontal stresses σ_x were analyzed (Fig. 4.6).

In front of stoping face in abutment pressure zone deflection of rock layer develops deeper than axis l_1 of maximum $(\sigma_y)_{max}$ action of vertical stresses about which testify anomalies $(\sigma_x \neq \lambda\gamma H)$ of horizontal stresses diagrams. Axis X is the beginning of intensive deflection of roof layers and depends upon correlation of coal seam rigidity and roof layers:

- at $E_y < E_R$ (see Fig. 4.5) inside massif move away maximum $(\sigma_y)_{max}$ of abutment pressure and coordinate X of active deflection of roof layers;
- at $E_y > E_R$ (most specifically for Western Donbass) initial axis of active deflection of roof layers approaches to stoping face, but stands far away than maximum of abutment pressure.

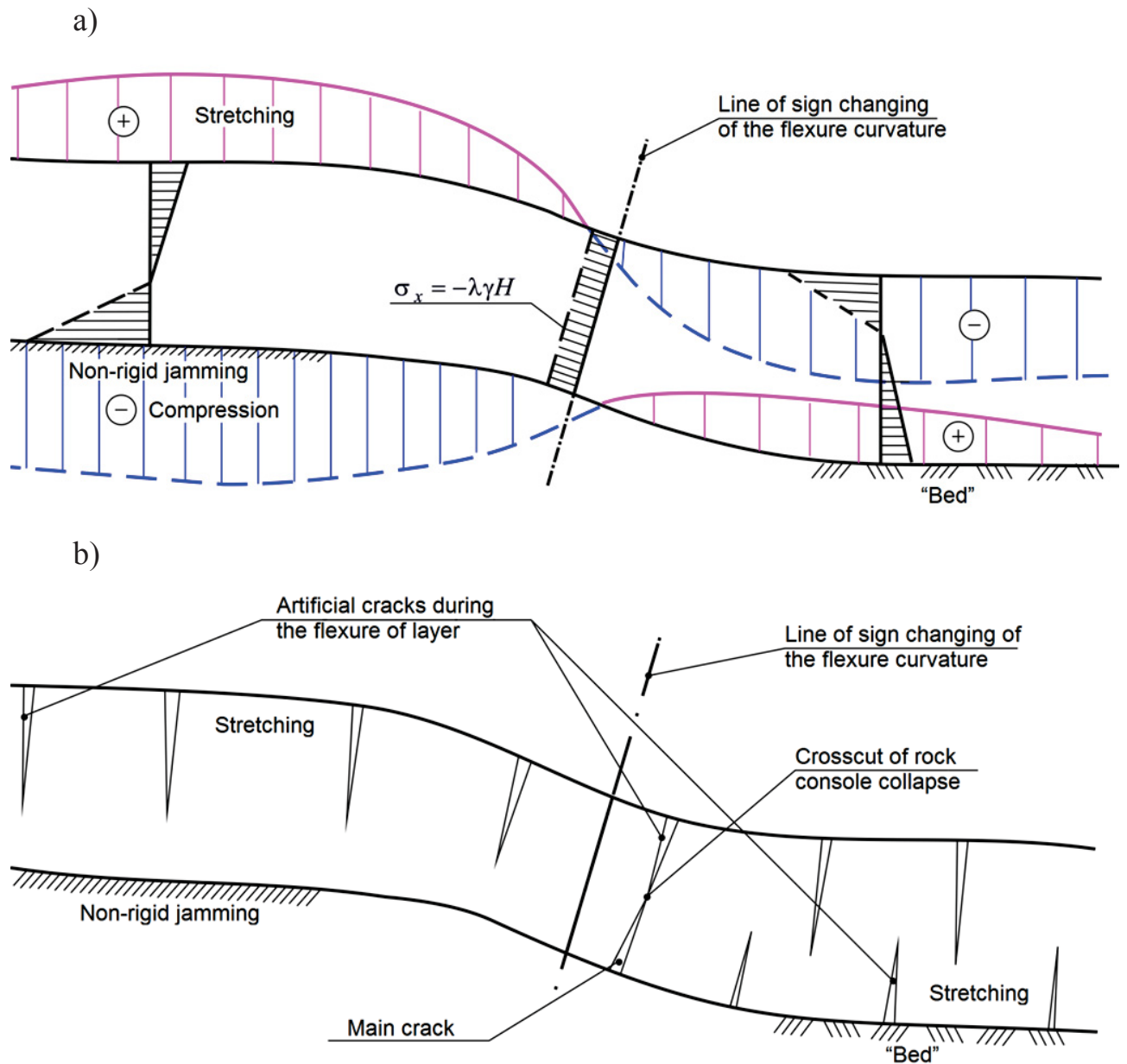


Fig. 4.6. Schemes of violent roof rock layer flexure:

- a) diagrams of horizontal stresses σ_x ; b) formation mechanism and connection of cracks in main line during flexure

Intensive deflection of roof layer begins on rigid jamming unit. Here it ought to be noted definitely small gradient of deflection moment value changing above the coal seam (flexure happens to the bottom side) and in worked-out area (layer contorted to the roof side). Only on small unit of layer bay happens intensive changing of deflection moment value and its mark.

Weak coal-bearing rocks of Western Donbass are characterized by low strength characteristics: resistance σ_s to stretching changes from 1...1,5 MPa in mudstones to 3...3,5 MPa in sandstones. Preliminary calculations of the above-coal strata stress-strain state shown that tensile horizontal stresses σ_x in roof rock layers during deflection can reach 10...15 MPa that predetermine appear and develop of vertically directed tension cracks in each roof rock layer. These cracks distributed on great part of rock layer thickness by means of force anisotropy of rock deformation properties. In work [42] was substantiated that neutral axis during rock layer deflection displaces to the side of compression area, because deformation modulus during stretching to 8...10 times less than during deflection. As it shown on Figure 4.6, a, area of tensile σ_x stresses during deflection distributes on 70...75% in upper part of layer thickness (see Fig. 4.6, b). From the other side, weakening factors of rheology and water content sharply decrease rock resistance to stretching and with some marginal stability during deflection, we can suppose absence of resistance to stretching forces. In addition, we can suppose appear of artificial system of vertical tension cracks with depth to 70...75% of layer thickness. If to artificial cracks system add nature roof fracturing, so will form slab with decreased resistance to deflection. This slab is easy-deformed during laying on "bed" from underlying rocks. On a unit from layer banding line before contact with "bed", tensile stresses σ_x form in lower part of rock layer and provoke tension cracks appearing. In a case of these cracks linkage with previously formed upper cracks, slab brakes on blocks that can save some steadiness (due to horizontal thrust forces), but more forced slab yielding during deflection.

Stated mechanism of rock layer deflection in stoping face area substantiated observed on practice events: low spacing of main roof breaks with blurred effects, layers deflection behind the longwall with limited length of hanging rock consoles. Line that characterizes deflection value changing has tendency (see Fig. 4.4) of inclination angle increasing with height of rock layer position growing. Such tendency is approved during preliminary modeling of the above-coal strata lowering. Previously pointed that tensile σ_x in lower part of layer thickness appear together with axis X of deflection value changing, do not reached "bed" and in lower part

form tensile cracks. At losses of beam system steadiness happens sliding of “gob” part of slab regardless to its near-the-face part, i.e. happens rock layer collapse, but at coincidence with cracks of overlying layers develops process of layer group simultaneous collapse, which is usually classified as main roof caving.

Generally, mechanism of main roof collapse is represented as linkage of some cracks in upper and lower part of each rock layer and main line crack formation on all thickness of main roof, horizontal axis X of which is situated near axis of layer deflection value changing. At hard coal seam and easy-deformed roof rocks (Western Donbass condition) of deflection value changing happens near from powered support protection. Logically suppose that line of deflection value changing on main roof thickness will limit (from the side of worked-out area) volume of rocks that create by own weight load on powered support. This load periodically changes from minimal value (after main roof caving) to maximal value (before main roof caving) that is caused by quantity of involving the formation on support rock layers.

From the side of stoping face border of rock contour that falling and loading powered support depends upon geomechanical and technological parameters of stoping operations conducting.

4.4. Conclusions

1) Analysis of the existing understanding of the processes of deformation of roof rocks in the area of the stope, as well as preliminary results of computer modeling and monitoring the pressure in the of powered support sections allowed to develop the scheme of the above-coal strata displacement applied to features of the layered massif of soft rocks of Western Donbass.

2) Was considered the mechanism of the load formation on the powered support, depending on main influencing factors, which is based on features of the mining of coal seams in Western Donbass. The loading of roof support is caused not by the hovering over it rock consoles with their subsequent caving, but by the subsequent lowering of mostly thin- and medium-layered rocks with strongly pronounced rheological properties and the almost complete lack of adhesion between layers.

3) According to the developed loading mechanism of the powered support was justified the position that the forming load on the support on the stoping equipment is defined by coordinates of two boundaries (in the cross-sectional view of the longwall): from the goaf it is the line of sign change of the flexure curvature of layers; from the wall face it is the dome contour of the load formation on the support.

5. DEVELOPMENT AND ANALYSIS OF LOADING MECHANISM OF THE ROOF SUPPORT OF THE LONGWALL SET OF EQUIPMENT UNDER THE DISPLACEMENT OF ABOVE-COAL SOFT ROCK STRATA

Methods analysis of load prediction on powered support led to outcome that they proceed from two basic propositions, which have temporality:

- 1) growth of rock consoles length within a face advance at on value of spacing of roof breaks;
- 2) process of overlying layers lowering on the deformed underlying layers of a roof.

Earlier it was noted that one of features of roof lowering in Western Donbass is small length of hanging. Therefore, the factor of length growth of rock consoles within spacing of roof breaks has secondary importance during forming of load on powered support. Then a major factor of loading is development of lowering process of rock layers over working space of a face. The size of load will depend upon the volume of the layers of a roof contacting with support through mutual transfer of rock pressure part. Here underline following parameters that having the most essential impact on process of loading formation:

- structure of roof rocks, thickness m_i of separate rock layers and friction forces c_i of their contacts with adjacent layers;
- strength (resistance to compression σ_{comp}^i and stretching σ_s^i), deformational (deformation modulus E_i and Poisson's ratio μ) and rheological (index $\left(\frac{x}{\beta}\right)_i$, that examined in Western Donbass in details) properties of each layer on main roof thickness;
- main roof caving step, which is function of the first two groups of parameters, and also other geomechanical factors (mining depth, influence of adjacent extraction units, overworked seams, their water content, etc.);
- advance rate V of stoping face and down-time t ;
- cutting width B of coal extraction equipment main element and intake V_k velocity;

- reaction $P(Z)$ of powered support units on face length Z at hydraulic props thrust and their maximal resistance (bearing capacity).

5.1. The mechanism of influence of above-coal rocks strata structure on the roof support loading

Mechanism of influence of the first group of parameters, main of which are structure of roof rocks and thickness m_i of separate layers is represented as follows. For its disclosure and more accurate understanding on Figure 5.1 scheme of loading of a separate roof layer, which is also used for an explanation of influence mechanism of other factors, is represented. On the scheme of separate i roof layer deformation with thickness m_i acted load is divided on three components: weight q_i of layer; load $q_{i+1}(x)$ from next overlying layer; load $q_{i-1}(x)$ in the form of reaction from the next underlying rock layer. Points 1 and 2 characterize border of contact disappearance between adjacent layers of a roof by means of their stratification; value of cavities $y_i(x)$ disclosure and their length on beddings of adjacent layers decrease in process of removal from longwall. Therefore, the point 1 is displaced to the right (towards worked-out area) concerning the point 2.

As it is known [44], deformation of any beam or plate at which section height (m_i) repeatedly is less than length of bay and generally defined by deflection moment $M_i(x)$ causing stresses on one-two orders surpassing those from action normal and cross-cutting efforts. Therefore, under the scheme of rock layer load, qualitative diagram of deflection moment (see Fig. 5.1, b) as main characteristics of condition and stability of i layer is given.

On a site of stratification and loss of contact with adjacent layers (to the point 1) deflection of i layer is defined only by its own weight q_i , and deflection moment $M_i(x)$ increases on classical parabolic dependence.

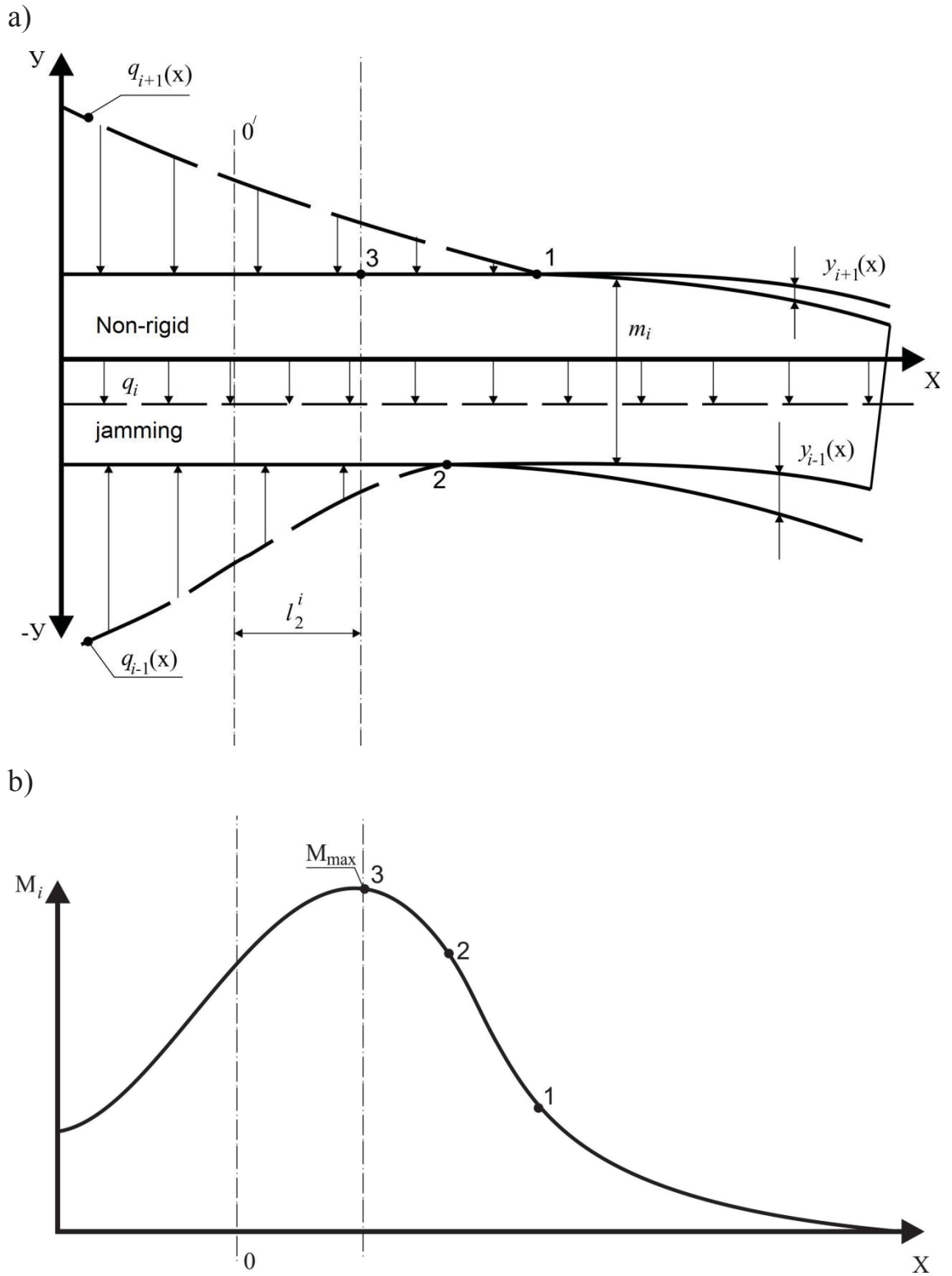


Fig. 5.1. Scheme of separate roof layer load (a) above stopping face and appropriate diagram (b) of deflection moment

Between points 1 and 2 balance weight $q_{i+1}(x)$ operates on a site from an overlying rock layer which increases a gradient of growth of deflection moment as

approaching to non-rigid jamming of i layer. On the left side from the point 2 on i layer starts working reaction $q_{i-1}(x)$ of underlying rocks, which exceeds load $q_{i+1}(x)$ on a balance condition. Then growth of $M_i(x)$ is slowed down and in the point 3 reaches a maximum M_{max} , and in process of deepening to the massif of effort $q_{i+1}(x)$, q_i and $q_{i-1}(x)$ compensate each other and the deflection moment asymptotically aspires to zero. The maximum M_{max} of deflection moment (the point 3) characterizes the most probable section of i layer break at which its right part completely lays down on the below-located rocks and eventually on powered support. In this regard the provision of the point 3 (axis X) will determine the volume of rocks by coordinate of i layer, creating load on support. The task consists in disclosure of influence mechanism of earlier grouped parameters on axis X of the point 3.

First group of parameters is structure of roof rocks and thickness m_i of separate layers, but forces of adhesion c_i on the planes of beddings for coal-bearing strata of Western Donbass can be neglected because of their small value or absence.

Influence of thickness of i layer consists that its deflection $y_i(x)$ is inversely proportional to the inertia moment, and the maximum of horizontal stresses $(\sigma_x)_{max}$ is inversely proportional to the moment of resistance of cross section of rock layer [44]. Opening this thesis at first we will address to communication of deflections of rock layer with its thickness m_i . The moment of inertia I_i is directly proportional to the thickness of rock layer in the third degree m_i^3 , therefore, at increase in its power the deflection $y_i(x)$ decreases on hyperbolic dependence and it means reduction of height $y_{i+1}(x)$ of a cavity on contact with overlying roof rocks and increase in height y_{i-1} of a cavity on contact with underlying roof rocks. Thus (see Fig. 5.1, a): the point 1 of contact loss with overlying rocks moves to the right, there is a growth of load $q_{i+1}(x)$; the point 2 of contact loss with underlying rocks moves to the left, reaction $q_{i-1}(x)$ decreases around the considered site on console coordinate X of i layer. The increase in loading from above and decrease in reaction from below in the area not a rigid jamming that rock console i leads to increase in the deflection moment M_i on a unit $1^I - 2^{II}$ (Fig. 5.2, a, puncture line) and its maximum M_{max}^I in the point 3^I . Along with

growth of deflection moment M_i at increase in thickness of i layer also the moment of resistance W_i of cross section of a layer that promotes increase of its stability increases, that is, two mutually opposite tendencies in an assessment of coordinate $(l_2^i)^I$ of the most probable collapse of the rock console and its lowering affect underlying layers with the corresponding loading of the powered support. Let's analyze the specified tendencies. Concerning the first one, dependence $M_i(m_i)$ can be claimed:

- firstly, growth of load $q_{i+1}(x)$ from above and decrease in reaction $q_{i-1}(x)$ from below are inversely proportional to heights $y_{i\pm 1}(x)$ of the corresponding cavities at a deflection of i layer which are inversely proportional to its capacities in the third degree (the moment of inertia of section I_i is directly proportional to m_i^3); therefore, on hyperbolic dependence (m_i^3) there is a growth of load $q_{i+1}(x)$ and decrease in reaction $q_{i-1}(x)$;

- secondly, deflection moment $M_i(x)$ in rock layer is directly proportional to the loads $q_{i+1}(x)$, q_i and $q_{i-1}(x)$ operating on it; thus it is necessary to consider increase in branch of the appendix of equally effective load $q_{i+1}(x)$ and decrease in length of branch of equally effective reaction $q_{i-1}(x)$ in relation to which deflection moment has [54] parabolic (square) connection;

- thirdly, imposing two separate regularities (cubic from deflections and square from lengths of branch equally effective) we come to the conclusion about extremely intensive growth of the deflection moment $M_i(x)$ at increase in thickness m_i of i layer (Fig. 5.2, a, puncture line).

Second regularity $W_i(m_i)$ is characterized by the growth of moment of resistance W_i of cross-section only on parabolic dependence on thickness m_i of i layer [44]. If compare two functions $M_i(m_i)$ and $W_i(m_i)$, so the first will increase more intensively in relation to the second, therefore, their relation $\frac{M_i}{W_i}(m_i)$ will increase.

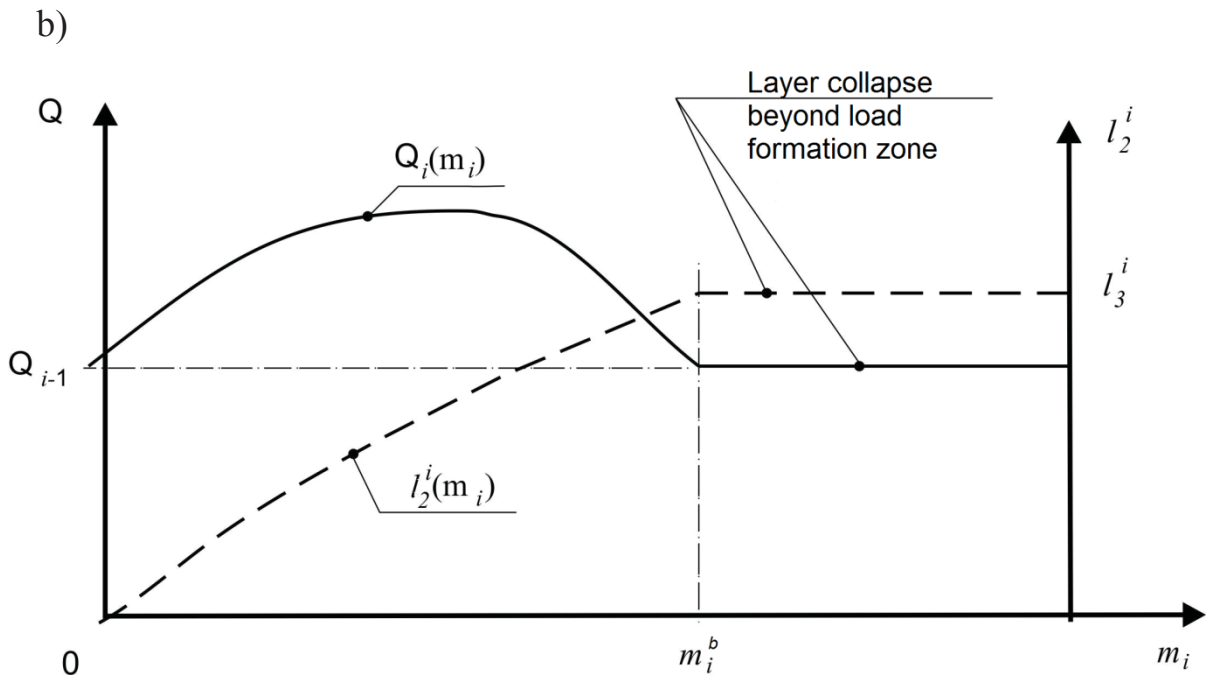
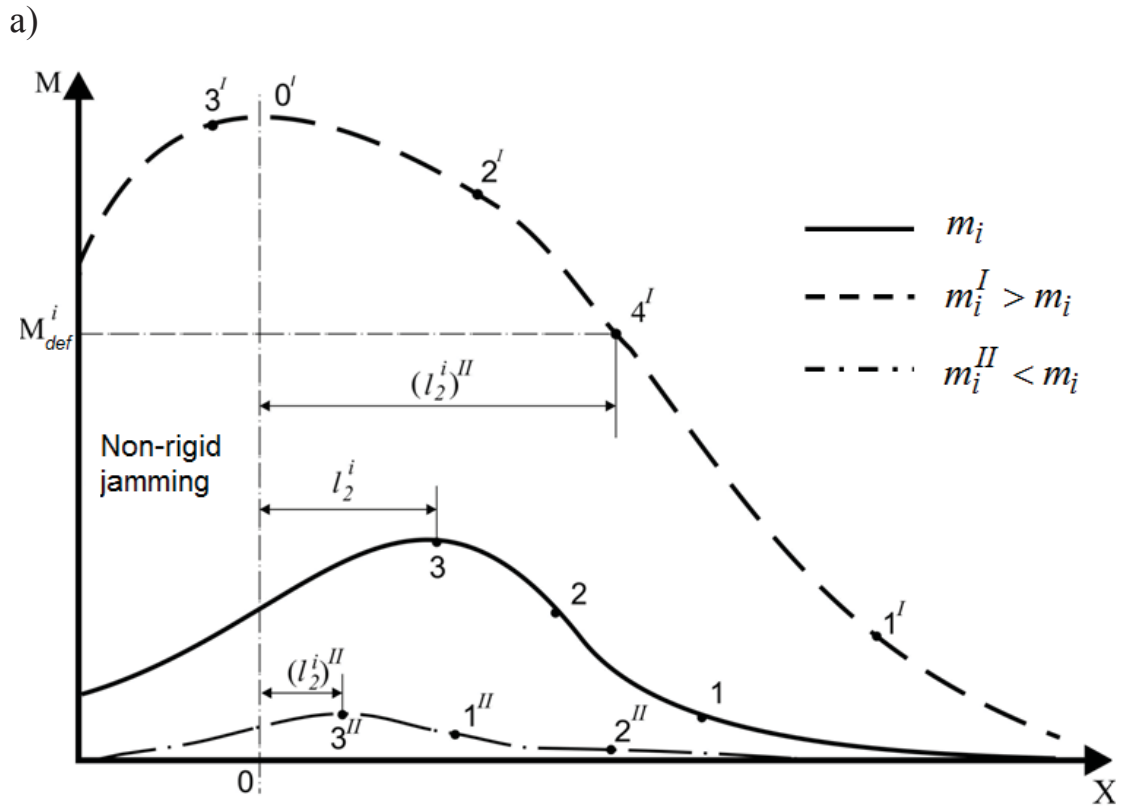


Fig. 5.2. Quantity regularities of thickness m_i impact on diagram (a) of deflection moment M_i in i roof rock layer and load Q_i (—) and length l_2^i (---) parameters (b) of rock console

On the other hand, the relation $\frac{M_i}{W_i}$ characterizes stresses σ_x of a rock layer deflection which maximum value in the field of compression is limited to the corresponding strength of rock σ_{comp} . Therefore, there is the maximum value of deflection moment M'_{def} , at which strength of rock on compression is reached and there steps the condition of a collapse of rock console and its lowering on underlying layers. As at the increased thickness m_i of i layer, deflection moment grows very intensively, a limit condition of the console (see Fig. 5.2, a, the point 4^I) comes long before achievement the deflection moment of maximum value M_{max} in the point 3^I and the distance $(l_2^i)^I$ from a non-rigid jamming increases, so, length of console hanging increases.

The increase in distance $(l_2^i)^I$ with growth of thickness of i layer leads (see Fig. 4.4) to decrease in volume of the rocks that created load on powered support. On the other hand, with weight increasing m_i of i layer that after lowering on underlying layers actively participates in formation of load on support. These two tendencies cause non-uniform communication of load $Q_i(m_i)$ with thickness m_i of i layer (see Fig. 5.2, b, continuous line). At the low thickness of i layer loading from a body weight is small and at total load $m_i = 0$ from weight of i quantities of layers of a roof Q_i corresponds to weight Q_{i-1} , created by underlying layers.

As the layer of low thickness m_i is characterized by low rigidity, its raised deflection provides movement of the point 2 with simultaneous increase in reaction $q_{i-1}(x)$ and a branch of its appendix of rather non-rigid jamming to the right (see Fig. 5.1, a). Low thickness of a layer m_i and increased reaction $q_{i-1}(x)$ provide a low gradient of growth of deflection moment, so that its maximum in the point 3^{II} (see Fig. 5.2, a) settles down near the coordinate of a non-rigid jamming and length of rock console $(l_2^i)^{II}$ will be small. Therefore, at the beginning of m_i growth the load Q_i will increase under the law close to linear as the weight of i layer from parameter m_i linearly increases (see Fig. 5.2, b). Further growth of m_i promotes increase in length l_2^i of the hanging console and the gradient of growth of load Q_i is

slowed down. In addition, depending on arrangement height of i layer from section fix, delay of growth of function $Q_i(m_i)$ can turn increase in this volume connected with growth into its decrease when growth of rock consoles $l_2^i(m_i)$ shown by puncture line reduces the volume of rocks m_i of i layer. The increase in growth of m_i to some boundary size m_i^b leads to that the hanging console l_2^i reaches border l_3^i of formation of load on support from worked-out area (see Fig. 4.4), it falls already by the brought-down rocks in worked-out area and doesn't participate in loading of section therefore, at $m_i \geq m_i^b$ (see Fig. 5.2, b) support perceives only weight Q_i of underlying rocks

Thus, influence mechanism on support load of a separate layer thickness m_i is considered. Their interaction between each other represents difficult process. In the qualitative view, it is possible to assume the following mechanism of influence of structure of the above-coal strata on condition of a separate layer of roof.

Let's consider option of thin-layer structure of roof rocks that lying above the considered rock layer at invariable structure of the below-located layers. Top low-power layers m_{i+k} (where $k = 1, 2, 3 \dots n$) the inertia I_{i+k} having the low moment and moment of resistance of cross section W_{i+k} , testing deflections and violations of uniformity fall by the considered rock layer thickness m_i . Load $q_{i+k}(x)$ including on contact $i + 1$, increases along with movement of the point 1^I of contact loss to the right (see Fig. 5.1, a). Thus, rock console m_i tests the high deflection moment M_i (see Fig. 5.2, a) which is repeatedly exceeding its bearing capacity that is caused by two factors:

- 1) on the one hand, joint action of the raised load (linear communication with deflection moment M_i) and increase in a branch of its equally effective (square communication with taking into account distribution of loading on console length);
- 2) on the other hand, moment of resistance W_i of cross-section of the considered layer remains to constants.

In total the power function of growth M_i takes place at invariable W_i that removes a console collapse point to the right (see Fig. 5.1, a) also promotes growth of

the load which is transferred on support. However, process of a collapse and lowering of a layer m_i doesn't come to an end (see Fig. 5.3, a): the layer break in the point 4^I sharply reduces deflection moment (continuous line) as the part of load $q_{i+1}(x)$ located more to the right of an abscissa of the point 4^I disappears and length of bay of the equally effective remained load of rock console of i layer, located more to the left of the point 4^I abscissa, decreases. Nevertheless, load $q_{i+1}(x)$ size on a unit $(l_2^i)^I$ such is that deflection moment M_i in rock console $(l_2^i)^I$ length exceeds the most possible size M_{def}^I and in the point 4^{III} there is its destruction and lowering on underlying rocks. The steady rock console by length $(l_2^i)^{III}$ repeatedly less initial length $(l_2^i)^I$ and a contour of the arch of formation of loading on support is as a result formed (see Fig. 4.4) comes nearer to border of a non-rigid jamming of a layer m_i . Thus, the thin-layer structure top (on the relations to a layer m_i) parts of a roof increases the volume of rocks that forming load on powered support.

In other option of presence of thin-layer structure is lower than a layer m_i , the last part of a roof, possessing the raised deflection that reduces reaction $q_{i+1}(x)$ value with movement of the point 2 to the left (see Fig. 5.1, a). Thus, deflection moment in a layer m_i increases even more and on described above to the mechanism of its deformation, the following process is supposed (see Fig. 5.3, b). Intensively growing deflection moment M_i in process of approach to a non-rigid jamming exceeds value M_{def}^I and in the point 4^I there is a destruction of the console. On the remained its length $(l_2^i)^I$ of deflection moment increases from zero (the point 1^{III}) to a maximum (the point 3^{III}) with excess in the point 4^{III} of limit value $M_{def}^{III} = M_{def}^I$, happens the following collapse of a layer to formation of the truncated console $(l_2^i)^{III}$. But reaction $q_{i+1}(x)$ of underlying thin-layer rocks is small because of their raised deflection and deflection moment M_i on this site again intensively increases, exceeding limit value $M_{def}^{III} = M_{def}^I$ in the point 4^{IV} ; here the small console is formed by length $(l_2^i)^{IV}$.

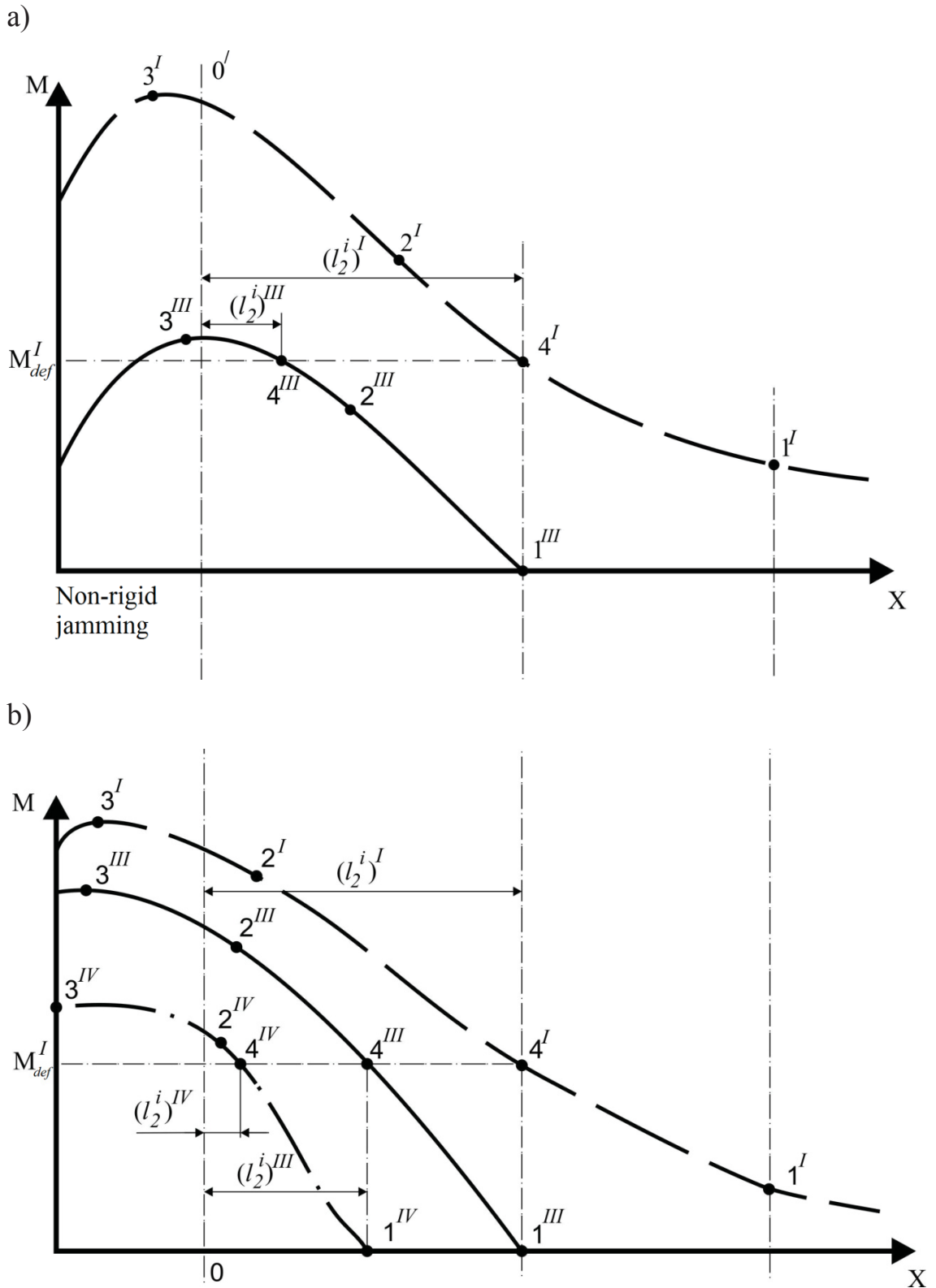


Fig. 5.3. Quality diagram of deflection moment M_i at variants of thin foliated roof rocks bedding a) above i layer; b) above and below i layer

Thus, the thin-layer structure of a roof rocks at the raised deflection promotes considerable loading of i layer with development of its stage-by-stage destruction and

lowering. Thus, rock consoles with a small length of a departure out of limits of a non-rigid jamming are formed, and the contour of the arch of load formation on support gets more vertical situation that increases the volume of unstable rocks over a stoping face and load on powered support. In this situation restriction of load possibly only at the very powerful, that layer which at the high rigidity (on condition of its integrity) completely perceives rock pressure from overlying layers and transfers it to the brought-down rocks in the developed space and the bottom hole massif ahead of a longwall. Such structure of the above-coal strata within the thickness of main roof in Western Donbass practically doesn't happened. Therefore, it is possible to predict increase in volume of unstable rocks with movement of a contour of formation of loading on support towards a face breast in the thin-layer massif. In the presence of middle not layered structure the contour of the arch of formation of load moves towards worked-out area and the volume of unstable rocks over a stoping face decreases. The mechanism of influence of structure of the above-coal strata and thickness m_i of its separate layer on development of loading on support and stoping complex are represented.

5.2. The mechanism of influence of deformation properties of above-coal rocks strata on the roof support loading

The following stage of research is disclosure of influence mechanism of rocks deformation properties of the above-coal strata on load formation on powered support of stoping complexes in relation to features of flat seams mining in difficult mine-and-geological conditions of Western Donbass.

In existing methods of forecast of load development on support where major influencing factor is length l_3^i of rock console, the main attention is paid to the module of deformation E_i of rock layer. Here specification of value E_i in respect of physical heterogeneity of rock depending on a sign of enclosed load is necessary. For example, in work [43] it is claimed that the module of deformation of rock on stretching E_i^S in 8...10 times less than module of deformation of rock on compression E_{comp}^S . Most of researchers connect this fact with existence of a jointing and lamination of coal-bearing

strata of Donbass rocks. The accounting of such “different modules” of rock (power anisotropy) is recommended to held introduction of given deformation module, which we will designate through E_i , and its size is equal to

$$E_i = \frac{4E_{comp}^i \cdot E_s^i}{\left(\sqrt{E_{comp}^i} + \sqrt{E_s^i}\right)^2} \quad (5.1)$$

will be, approximately, in four times less (at the above ratio $\frac{E_s^i}{E_{comp}^i}$ on stretching and compression), than E_{comp}^i defined in laboratory conditions.

Strength characteristics of rocks (resistance to compression σ_{comp}^i and stretching σ_s^i) are seldom considered at the description of processes of displacement and a collapse of layers of a roof over a stoping face and in worked-out area. Here at the insignificant resistance to stretching of lithological differences of poor rocks of Western Donbass it is expedient to be limited to only rocks resistance to compression σ_{comp}^i [55, 56]. In the same works rheological properties of Western Donbass rocks which data will be used during studying the mechanism of development in load time on powered support regarding to decrease in strength and deformation characteristics of rocks layers in the course of conducting stoping operations are examined in details.

Let's estimate influence of the module of deformation of i rock layer on process of its deflection and lowering with usage of scheme in Figure 5.1. Raised module of deformation E_i increases rigidity of i layer and reduces the value of its deflection $y_i(x)$; it causes reduction of height $y_{i+1}(x)$ of a cavity on the top surface of a layer and increase in height $y_{i-1}(x)$ of a cavity on its lower surface with other things being equal. As a result, the point 1 moves with increase in load $q_{i+1}(x)$ of rock console to the right, and the point 2 moves to the left, promoting decrease in reaction $q_{i-1}(x)$. As a result, deflection moment M_i increases (similar to the change M_i , which is qualitatively presented by lines in Figure 5.3, b), and its limit value M_{def}^i remains to constants because of invariable other conditions. Then there is a process which is

already described earlier: destruction and lowering of console length $(l_2^i)^I$, increase on its length M_i to limit value M_{def}^i with the subsequent collapse and stage-by-stage reduction of length of the console at first to $(l_2^i)^{III}$, and then to $(l_2^i)^{IV}$. Thus, at increase the increased rigidity E_i of a layer generates increase in its loading that causes reduction of length of the console. Such regularity $l_2^i(E_i)$ contradicts the dependence offered in work [42] for approximate determination of length of the console. However, this contradiction “seeming” as in work [42] means the console length $l_3^i - l_2^i$ falling on support, that is a difference according to the scheme upon Figure 4.4 and, if length of the console l_2^i decreases with increase E_i , the above difference will increase, eliminating a contradiction. Besides, formulated conclusion is in full consent with power representations in geomechanical systems: the increasing potential energy of a state of i rock layer (the increased tension and deformations) has to be compensated by increased kinetic energy in the course of destruction which is directly proportional to the volume of the falling rocks, therefore, is inversely proportional to length l_2^i of the hanging console.

Otherwise, lowered deformation module E_i of i rock layer and its increased deflection promotes growth of cavity $y_{i+1}(x)$ height on the top surface and decrease in height of a cavity $y_{i-1}(x)$ on the lower surface of a layer (see Fig. 5.1, a). It causes movement of the point 1 with reduction of load $q_{i+1}(x)$, and the point 2 to the left – to the right with increase in reaction $q_{i-1}(x)$. As a result, deflection moment M_i on length of the console grows less intensively that causes the following options of process development. First maximum deflection moment M_{max} in the point 3^{III} (see Fig. 5.3, b) is more than limit value M_{def}^I and destruction of the console happens in the point 4^{III} to formation of its departure length $(l_2^i)^{III}$; further growth of deflection moment M_i differs from the point 1^{IV} in low intensity and its maximum M_{max} in the point 3^{IV} less limit value M_{def}^I , then again formed steady console is characterized by the increased length $(l_2^i)^{III}$. The second option is deformation module E_i of i rock layer is very low and the gradient of growth of deflection moment is to such an extent

small that the console will collapse in the point 3 actions of a maximum of deflection moment M_{max} (see Fig. 5.1, b), which is settling down at the distance l_2^i determined by provisions of points 1 and 2. In our case, the point 2 will be more removed from non-rigid jamming, than the point 1; therefore, the point 3 maxima M_{max} will also move away from a non-rigid jamming and length l_2^i of the console increases. Thus, the conclusion is drawn on increase in length l_2^i of the console of i rock layer at decrease in its module of deformation E_i because more pliable layer relies on underlying rocks and thanks to their reaction steadiness of i layer increases.

Given conclusion (at low E_i) demands judgment regarding that, at first sight, easily deformed rock layer has to collapse at once with formation of the console of small length l_2^i , and a contour of the arch formation of load to adopt more vertical provision. However, it is claimed that at low E_i length of the console l_2^i increases. The seeming contradiction is explained by that at this stage deformation of a separate layer, but not roof rocks as the interacting system of these separate layers is considered.

In this plan, considering the above-coal strata as the interacting system of separate layers, some of the most probable cases of influence of the module of deformation E_i on formation of load on powered support through parameter l_2^i are allocated. Firstly, we will sort a “disputable” case of the low module of deformation E_i of i layer: it, falling by underlying layers, increases by them loading that can give to a collapse of rock consoles created below and reduction of their length; it is followed by decrease in reaction $q_{i-1}(x)$ and a beam is equally effective on i layer, increase in it deflection moment M_i and the subsequent collapse with reduction of steady length l_2^i of the console. Similar chain reaction can arise because of loss of stability of consoles of overlying rock layers, after all reaction $q_{i+1}(x)$ to them of i layer decreases because of its lowering. The collapsing overlying rocks increase load $q_{i+1}(x)$ on i layer, deflection moment M_i grows in it and it also collapses with reduction of length l_2^i of the console; this chain reaction can develop on underlying layers. Therefore, final conclusion of regularity of influence of the module of

deformation of layers can be drawn on the basis of research of their interaction by the most modern, considering a large number of factors and powerful tool – computer modeling of geomechanical processes.

Here attempt to open the mechanism of influence of deformation properties of roof rocks is made really proceeding processes were most objectively reflected in computer model. In this plan, influence of modules of deformation of overlying E_{i+k} and underlying E_{i-k} layers on a state of i rock layer is estimated. At the raised deformation characteristics of overlying layers, they due to higher rigidity test the lowered deflection $y_{i+k}(x)$ over a longwall and the adjoining to worked-out area. Parameters of i layer remain invariable therefore height of a cavity $y_{i+1}(x)$ increases (due to restriction of a deflection of overlying layers) and the point 1 (see Fig. 5.1, a) is displaced to the left – loading $q_{i+1}(x)$ on i layer decreases. At invariable parameters of underlying layers the point 2 remains on a place, also as well as value of reaction $q_{i-1}(x)$ of underlying layers doesn't change. Therefore, deflection moment M_i decreases in value, and its maximum in the point 3 moves away on bigger distance l_2^i from a non-rigid jamming. As a result, the tendency of length growth of the console l_2^i (it is more unloaded) with increase in the module of deformation E_{i+k} of overlying layers takes place.

In other variant of distribution of deformation characteristics on the power of main roof rocks when the raised modules of deformation E_{i-k} have underlying rock layers, the following mechanism is predicted. Here the deflection of underlying layers decreases and height $y_{i-1}(x)$ of a cavity decreases that involves (with other things being equal) movement of the point 2 to the right and the increase in reaction $q_{i-1}(x)$ – deflection moment M_i decreases, and its maximum in the point 3 moves to the right. That is, stability of the rock console that causes growth of its length l_2^i increases.

In the third version of the raised deformation characteristics both in overlying, and in underlying rock layers (in relation to E_i) both previous variants, that is, decrease in load $q_{i-1}(x)$ from overlying layers and the increasing $q_{i-1}(x)$ reaction

are combined from underlying layers that in total leads to substantial increase of stability of i layer and length l_2^i of its console growth.

Thus, excess of modules of deformation $E_{i\pm k}$ of any layers of a roof over the module of deformation E_i of a separate layer in all options leads to increase in length l_2^i of the console. Otherwise ($E_i > E_{i\pm k}$) the increased rigidity of i layer generates development on it the raised loading $q_{i+k}(x)$ from overlying layers and the lowered reaction $q_{i-k}(x)$ from underlying layers. It promotes decrease in stability of rock console (as it was already considered above) and reduction of its length l_2^i .

Now let us consider how change of deformation characteristics of layers influences formation of loading on powered support according to the scheme of displacement of the above-coal strata (see Fig. 4.4), where the main attention is paid to the provision of a contour of the arch over a clearing face. The schematic representation of process shown in Figure 5.4 is developed for this purpose.

At first connection of loading Q on powered support with the module of deformation E_i of i layer taking into account earlier given features of the mechanism of development of multistage loading and a collapse of layers of a roof is considered (see Fig. 5.4, a). At invariable modules of deformation $E_{i\pm k}$ of overlying and underlying rock layers will be tracked high-quality change of loading Q on powered support, since extreme value $E_i \rightarrow 0$ (the point A_0), to which there corresponds some loading Q_{i+k} . In process of increase E_i , i layer due to the growing rigidity reduces transfer of load of underlying layers, perceiving on itself the increasing part of loading from overlying layers. Therefore, loading Q on support decreases (site $A_0 \dots A_1$). Such tendency cannot infinitely last as the dashed line, which asymptotically is coming nearer to loading size Q_{i-1} from underlying layers, shows it. At thin- and medium-layered structure of the roof in the point A_1 there is i layer collapse because of high loading from overlying layers and low reaction from underlying layers. The collapse of some length of the console of i layer in steps increases loading on support (site $A_1 \dots A_2$). The further increase of E_i again reduces loading Q (site $A_2 \dots A_3$), but less intensively, as hanging, again occurred, console

already has smaller length. Nevertheless, at its certain value again there is a console collapse for the above reasons (the point A_3) and loading Q in steps (site $A_3 \dots A_4$) increases by the weight of the corresponding volume of rocks. Here loading in the point A_4 can surpass initial value of Q_{i+k} (the point A_0) as escalating rigidity of i layer provokes a collapse of the truncated i layer consoles on earlier described mechanism. As a result, at $E_i \rightarrow \infty$ length of the hanging console all volume of i layer aspires to zero also, and also the raised part of overlying rocks participates in formation of loading on powered support, which comes nearer to theoretically maximum value Q_{max} .

Now let's move to consideration of regularity of influence of modules of deformation E of all rock layers of the main roof on formation of loading Q on powered support (see Fig. 5.4, b). At a uniform roof (the module of deformation of all layers is identical $E_{i\pm k} = E$, the continuous line) increase of deformation properties at the same time of all layers reduces intensity of their lowering and loading on support until a collapse of the lower part of layers – loading in steps increases. Thus, from underlying layers repulse disappears on a site of the fallen consoles and stability of overlying layers decreases. With a further growth of E , there is a collapse and overlying layers – again loading sharply increases and the cycle of regularity $Q(E)$ is repeated. In the second option of the lowered module of deformation of underlying layers ($E_{i-k} < E$, the dashed line) dependence $Q(E)$ decreases on the first site as less rigid underlying layers form the main share of the loading transferred by more rigid overlying layers more intensively. Having reached some limit length (bigger, than at uniform rocks at the expense of the pliability) consoles of underlying layers collapse first and the weight of part of length of rock console is transferred on support – loading Q in steps increases. Thus, reaction to overlying layers decreases, they pass into an unstable state and part from them collapses, provoking one more jump of loading on powered support. Further process can repeat, but the general tendency such is that loading on powered support at $E_{i-k} < E$ below, than at a uniform roof as the part of more rigid overlying rock layers does not collapse and thanks to the rigidity, these layers transfer the lowered loading on support.

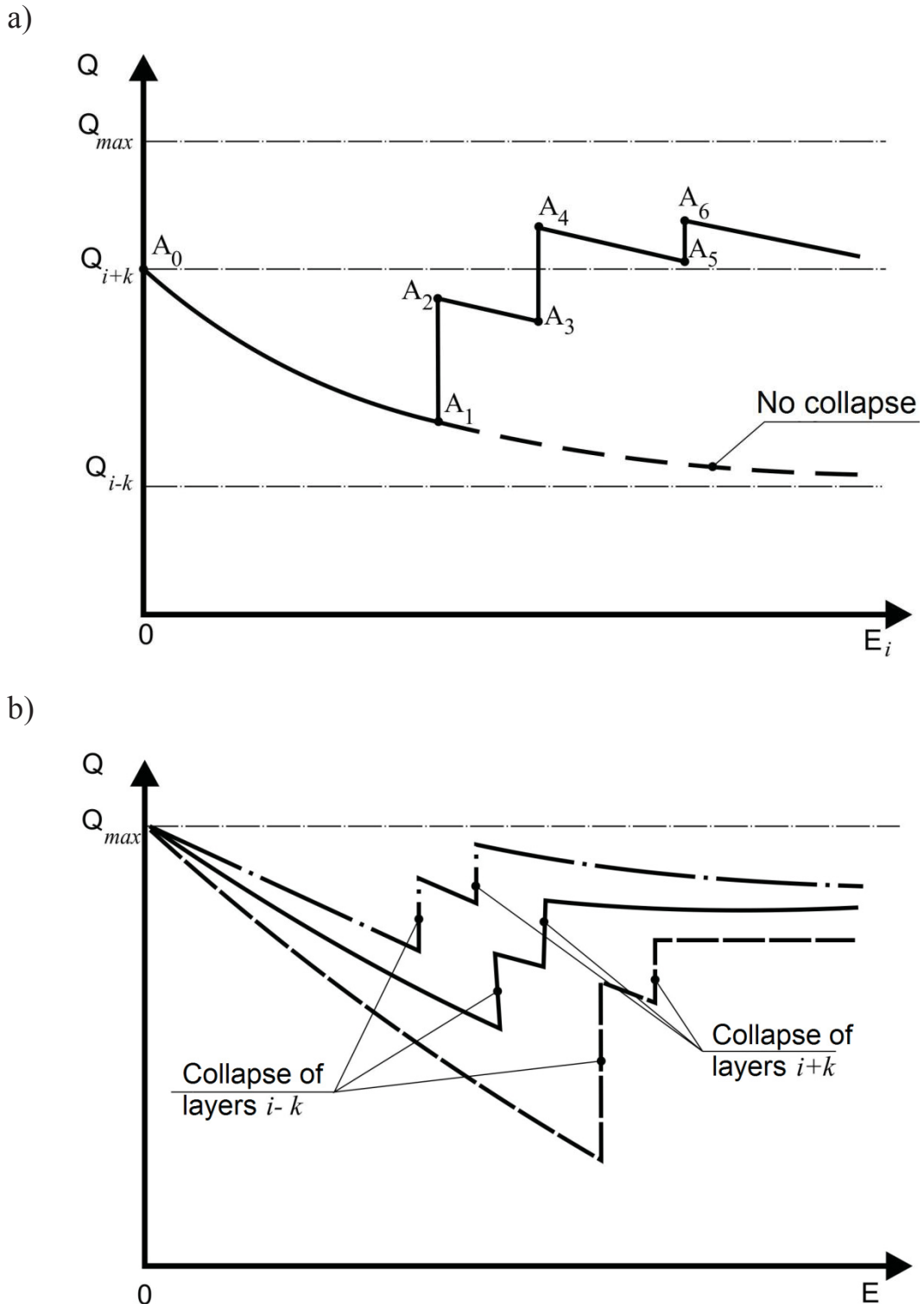


Fig. 5.4. Schematic representation of load development on powered support depending on deformation modulus of layer (a) and deformation modulus of roof in total (b) at: — $E_{i+k} = E$; - - - $E_{i-k} < E$; - · - · - $E_{i+k} > E$

In the third option ($E_{i\pm k} > E$, the dash-dotted line on the Fig. 5.4, b), the first site of dependence $Q(E)$ is similar to previous, but the line is located above other

options as more deformable overlying layers fall on underlying much more intensively, load the last and process of a collapse of rock consoles happens earlier.

The general conclusion about regularities of influence of the module of deformation E on loading Q on powered support is as follows:

- before and after stages of a collapse of rock consoles loading Q decreases with increase in their modules of deformation E because of restriction of deflections of more rigid rock layers;
- in the period of a collapse loading in steps increases due to the weight of volumes of caving rocks on support;
- process of a collapse has character of chain reaction and begins with underlying layers up to the height, when there comes the steady condition of the next overlying layer on length of its bay from a face breast before contact with caved rocks in the goaf;
- in this process the bedding in the main roof of more powerful lithological difference with the raised deformation module sharply increases rigidity of a layer and can lead to short circuit of the arch of a collapse with formation of the lowered loading on support and to sharp jump of loading at a layer collapse within working space of a longwall and to the adjoining site of worked-out area.

5.3. The mechanism of the influence of strength and rheological characteristics of above-coal rocks strata on the roof support loading

The next stage of the disclosure of the mechanism of development of the load on the powered support is devoted to the study of the influence of compressive strength of i rock layer and the main roof in total. Here should be noted the similarity of the mechanism of influence of the strength (σ_{comp}) and deformation (E) properties of rocks on the process of its lit-by-lit lowering on the roof support and in border goaf. The general trend is that with the increase σ_{comp}^i (other conditions being equal) the moment M_{hold}^i increases linearly, which holds the rock console from caving

$$M_{hold}^i = \sigma_{comp}^i W_i, \quad (5.2)$$

that causes an increase in its length l_2^i . However, with the l_2^i growth the load Q_i increases linearly under own weight of the console (see Fig. 5.1, a). In this case, considering the parabolic connection of the deflection moment M_i in the console with its length l_2^i [44], conclude that the ratio of “holding” and deflection moments $\frac{M_{hold}^i}{M_i}$ with increase in the console length will decline, despite the increase. That is why, the growth of the console length is limited by the more sharp decline of its competence. On the other hand, unlike the influence E_i , console rigidity with the growth of σ_{comp}^i is not changed, consequently the load $q_{i+1}(x)$ from above and the reaction $q_{i-1}(x)$ from below on i console remain the same and there is no significant redistribution of the load between the overlying and underlying rock layers during the growth of the console length until its caving.

Now let's analyze how the growth of σ_{comp}^i of i layer affects the load changes on the powered roof support. With very low $\sigma_{comp}^i \rightarrow 0$ the length of stable rock console also tends to zero ($l_2^i \rightarrow 0$) and almost the entire volume of i layer is lowered to the underlying layers - the load on the powered support corresponds to the value Q_{i+k} , the maximum for given conditions. With the growth of σ_{comp}^i the length l_2^i is increased and the volume of i layer is decreased. The trend is observed similar to the effect of the deformation modulus E_i (Fig. 5.4, a, the area A_0A_1). Here, however, the intensity of the decline of $Q(\sigma_{comp}^i)$ is much less, because the sustained console transfers part of its weight on underlying rocks within the limits of the dome of the load formation of the support. On the other hand, for the above reasons of active growth of the function $l_2^i(\sigma_{comp}^i)$ can not occur, since the deflection moment is increased much faster than the holding torque M_{hold}^i ; the function $l_2^i(\sigma_{comp}^i)$ is flattened, asymptotically approaching to some constant value $(l_2^i)_{max}$ when $\sigma_{comp}^i \rightarrow \infty$. Therefore, the function is stabilized and the function $Q(\sigma_{comp}^i)$ at a certain level smaller than the initial value Q_{i+k} . The main reason for this limited influence of

σ_{comp}^i is that this parameter does not make significant changes in the process of interaction between layers of the main roof and the diagram of the load distribution between them, and, consequently, in the position of the contour of the dome of the load formation on the powered support (see Fig. 4.4).

Similar trends are observed in the analysis of the influence of the compressive strength of rock layer groups of the entire main roof. Since strength characteristics of lithological differences of a coal-containing rock mass of Western Donbass as a soft with a relatively small range of change then their impact on load variations on the powered support should be assessed as insignificant. The exception can be only make conditions of occurrence in the main roof of sufficiently thick sandstone ($m_i > 4 \dots 5 \text{ m}$), which with the combination of high strength and deformation characteristics will close the contour of the dome of the load formation on the powered support. Such exceptions only confirm the regularity of weak influence of the compressive strength on the processes of displacement of the above-coal strata in the vicinity of coal-face works for conditions of Western Donbass.

At the final stage was considered the effect of rheological properties of rock layers of the roof on the mechanism of the load formation on the powered support. In studying the influence of rheological properties two trends are allocated:

- deformations development in time at a constant load, which is considered in the theory of elastic-hereditary creep by decrease of the deformation modulus depending from the time t of load application and the magnitude of the prolonged deformation modulus E_i^∞ (at $t \rightarrow \infty$) is associated with its conventionally instantaneous value by the ratio

$$E_i^\infty = E_i \left[1 - \left(\frac{x}{\beta} \right)_i \right], \quad (5.3)$$

where $\left(\frac{x}{\beta} \right)_i$ - rheology i layer of rock;

- reduction in time t of rocks resistance to compression, the long-term value $(\sigma_{comp}^i)^\infty$ of which is related to the conditional instantaneous value by the ratio

$$(\sigma_{comp}^i)^\infty = \sigma_{comp}^i \sqrt{1 - \left(\frac{x}{\beta}\right)_i} \quad (5.4)$$

The above relations are easy to use, and rheological parameters $\left(\frac{x}{\beta}\right)_i$ are studied in detail for rocks of Western Donbass on the basis extensive laboratory studies [55, 56].

Rheological factor is required to account by a number of reasons:

- 1) extraction works flow in time and the conditional instantaneous position of the face reflects only one from an infinite number of positions for a certain period of time;
- 2) all lithological differences of coal-containing strata of soft rock are characterized by clearly expressed rheological properties;
- 3) technological parameters (advance rate V of the stope, its down-time t , feed rate V_k of the stoping machine) are a function of time;
- 4) step of the main roof caving is manifested in a layered massif of soft rocks, which also has a time frame in the process of stope advance.

In the first place was analyzed the influence of rheological properties of rocks in terms of the development in time of their creep and related phenomenon of the reduction of compressive strength and deformation modulus. These trends contribute to the intensification of the lowering of the roof layers, which leads to change in the position of two contours (see Fig. 4.4), limiting the amount of rocks that load the powered support.

Previously, it was found that the growing deflection of layers, starting from their downstream portion that adjacent to the roof support, reduces the reaction $q_{i-1}(x)$ on i layer (see Fig. 5.1, a), its resistance is also reduced and it is collapsed closer to the nonrigid jamming. Moreover, all marked factors (creep, $E_i(t)$, $\sigma_{comp}^i(t)$) act in the

same direction: the length l_2^i of the console is decreased in time t of its loading and the position of the whole dome contour is moved from the conditionally initial position C_0 (time t_0) to the new position $C_1(t_1 - t_0)$, as it is shown in Figure 5.5.

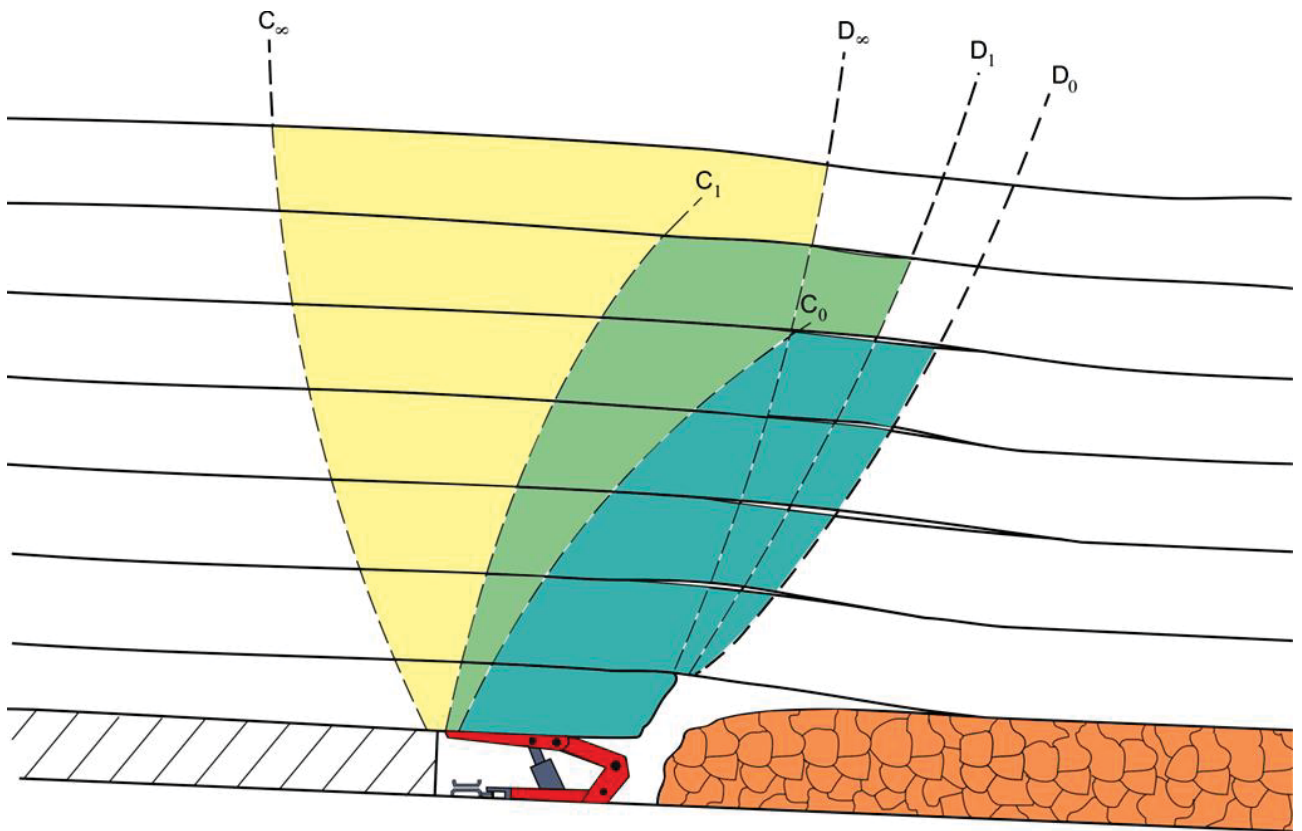


Fig. 5.5. Schematic representation of the formation of the load on the support in time

It should be noted that the change in the exposure duration of loads concentration is directly related with the stope advance rate V , because it determines the time of passage of a site with the length l_3^i (see Fig. 4.4). During the stop of the stoping machine the increase of loading time of the roof support corresponds to the down-time plus the previous time of passage the site l_3^i .

By practice was pointed out that with the increase in down-time of the stoping machine increases the load on the powered support. Consequently, the volume of unstable rocks is growing, which are concluded between lines C_j and D_j in Figure 5.5. In a flat cross-section of the above-coal strata the load on the support will be equivalent to the area bounded by the above-mentioned lines. During prolonged

down-time ($t > t_1$) of the face rock consoles l_2^i continues to be shortened due to the development of the creep process of the layer and increase its deflection over the stopping face. Here should be noted the regularity of the increasing creep strain with the growth of stresses intensity, especially when latest are approached to destructive values [55, 56]. Then, by decreasing the length of the stable console it is approached to the non-rigid jamming, where the rock pressure is increasing and rheological properties are manifested stronger (see Fig. 4.4). Moreover, deformations are amplified in the non-rigid jamming starting from the wall face (see Fig. 4.5), and the dome contour of the load formation take up the position close to the vertical. Changing the position of the dome contour is limited (at $t \rightarrow \infty$) by the line of bearing pressure maximums (see Fig. 4.4), where rock layers creep process is most intense; this position is reflected by the line C_∞ in Figure 5.5 and according to various estimates is characterized by the initial slope of the contour in the direction of the untouched massif up to $10 \dots 15^\circ$. In such a way, there is tendency of the position changing of the dome contour with the time of loading of rock consoles l_2^i . The duration of loading period is much higher while stopping of the working face than while it is advanced with different speeds. However, significant differences in the advance rate V can lead to the significantly different position of the dome contour of the load formation on the support, because the time of passage of the site l_3^i by the longwall can be up to several days, in other words, the period when creep deformations are growing the most intensive [55, 56].

Let us consider in time the position of the line change of the sign of the flexure curvature of layers (see Fig. 4.4) over the goaf (lines C_j and D_j , Fig. 5.5.). With the increase of the advance rate V of the stope increases the length of l_2^i of the stable rock console. Due to the reduction of the time t of passage of the site l_3^i , its length is increased and the line of change of the sign of curvature of the flexure is displaced toward the goaf (position D_0). Reduction of the time of deformation of i layer above the longwall causes a decrease in its deflection and removal of the inflection point

from the stope. The mechanism of the effect of rheological properties is not fundamentally different for lines C_j and D_j , but there are a number of features:

- 1) for the majority of lithotypes the creep deformation are damped in the time t of load application;
- 2) the load is reduced at the site due to the stratification of the roof; the creep deformation are reduced too;
- 3) change of the sign of curvature takes place in the area of the maximum stratification of the roof (see Fig. 4.4 and Fig. 5.5), and the resulting cavities with the height $y(x)$ partially compensate deflections; loading conditions of layers are changed slightly.

By the above reasons, changes of the position of lines D_j in the course time of the process of displacement are less significant. With the growth of t the area that enclosed between lines C_j and D_j , is increased - the load on the powered support is also increased.

5.4. The mechanism of the influence of the main roof caving step and the reaction of the roof support resistance

Let us consider the mechanism of influence on the formation of the load on the support of parameters: step L_{cav} of the main roof caving, the feed rate V_k of the stopping machine and the web width B of its actuating element.

Observed in practice cases of the active influence of the main roof caving can be explained as follows. In the area of the change of sign of the flexure curvature of the layer (see Fig. 4.6) appears the main crack. These main cracks for different layers do not coincide with each other in the coordinate - appears a likeness of the spacer-block system with supporting on caved rocks and rocks that hangs above the roof support. This support transfers part of the weight of rocks on the powered support and the load on it increases until the merger of separate main cracks. Collapse of the main roof occurs. In all other cases, the caving of the main roof occurs with insignificant variations of the load on the powered support. In such way, it can be considered to

justify the exclusion of the step L_{cav} of the main roof caving from the list of main factors.

Influence of parameters V_k and B on the process of the above-coal strata displacement and the formation of the load on the support is observed and is divided into two positions:

- 1) the combination of parameters V_k and B with the length of the longwall and execution time of auxiliary operations determines the advance rate V of the stope;
- 2) there is local variation of the stress-strain state of rock layers in the roof in the part of the face, where is produced coal extraction.

The second position is closely related to the structure and mechanical properties of rock layers. The process of the collapse of the immediate roof and stratification of the overlying rocks occur over time. Considering the existing feed rate of stopping machines, only for the low V_k (with the web width equal to $B = 0,63 \dots 0,8 \text{ m}$) will be observed significant caving of the immediate roof. The process is enhanced with the stopping of the miner. Perturbations of the stress field in the local area along the length of the longwall up to 15 ... 20 m act a small period of time (up to 10 minutes) and processes of formation of the load on the support do not have time to change significantly. For ploughs with the small web width ($B \leq 100 \text{ mm}$) and high rates of movement of the actuating element the influence of considered parameters much less.

The final stage of studies of the mechanism of the load development on the powered support is devoted to the analysis of influence of the reaction P of its resistance to the process of the roof displacement above the face. Substantiated statement about the layerwise lowering of the roof, starting from the immediate roof and spreading to the main roof up to the stabilization of the formation of the load Q (closure of contour formed by lines C_j and D_j in Fig. 5.5). The reaction $q_{i-1}(x)$ on the first layer ($i = 1, q_0(x) = P$) increases its stability, the deflection moment in it is decreased (see Fig. 5.1). This increases the length l_2^1 of the hanging console. The reaction $q_1(x)$ is increased on the second layer ($i = 2$), the increased resistance of which leads to elongation the console l_2^2 and so on through the overlying layers.

Thus, with the increase of the resistance reaction of the roof support the line P is moved toward the goaf (see Fig. 5.5). Weight of rocks inside the dome of the load formation on the roof support is reduced. On the other hand, the position of the line D_j of changes of the sign of flexure curvature is changed to a lesser extent. Furthermore, the resistance reaction P has little effect on the process of closing of cavities between the layers. Then with the increase of the resistance reaction P of the powered support the load Q on it by weight of the downgoing roof rocks is reduced, which is schematically depicted in Figure 5.6, a. With such regularities the rational value of the resistance reaction P_r of the roof support will be the one at which in the roof is formed the dome with the weight of rocks Q_p equal to P_r . In any other situation will be higher or the reaction of the support ($P > P_r$ with $Q < Q_p$ - the right part of lines from points 1, 2, 3) or the load on the support ($P < P_r$ with $Q > Q_p$ - the left part of lines), for the balancing of which still will be occur the growth of P .

On the mentioned regularity also affects the stope advance rate V . In the end, there is regularity (see Fig. 5.6, dashed line) of changes of the rational value of reaction $P_r(V)$ from stope advance rates. This regularity will be modified from the degree of influence of other factors (structure and properties of roof rocks, the depth of the development and the like). Within the limits of certain area of conditionally stable geomechanical situation the regularity $P_r(V)$ allows operatively adjust technological parameters of extraction works in order to prevent the accidental position of overloading of the powered support and the landing of the longwall set of equipment on “hard base”.

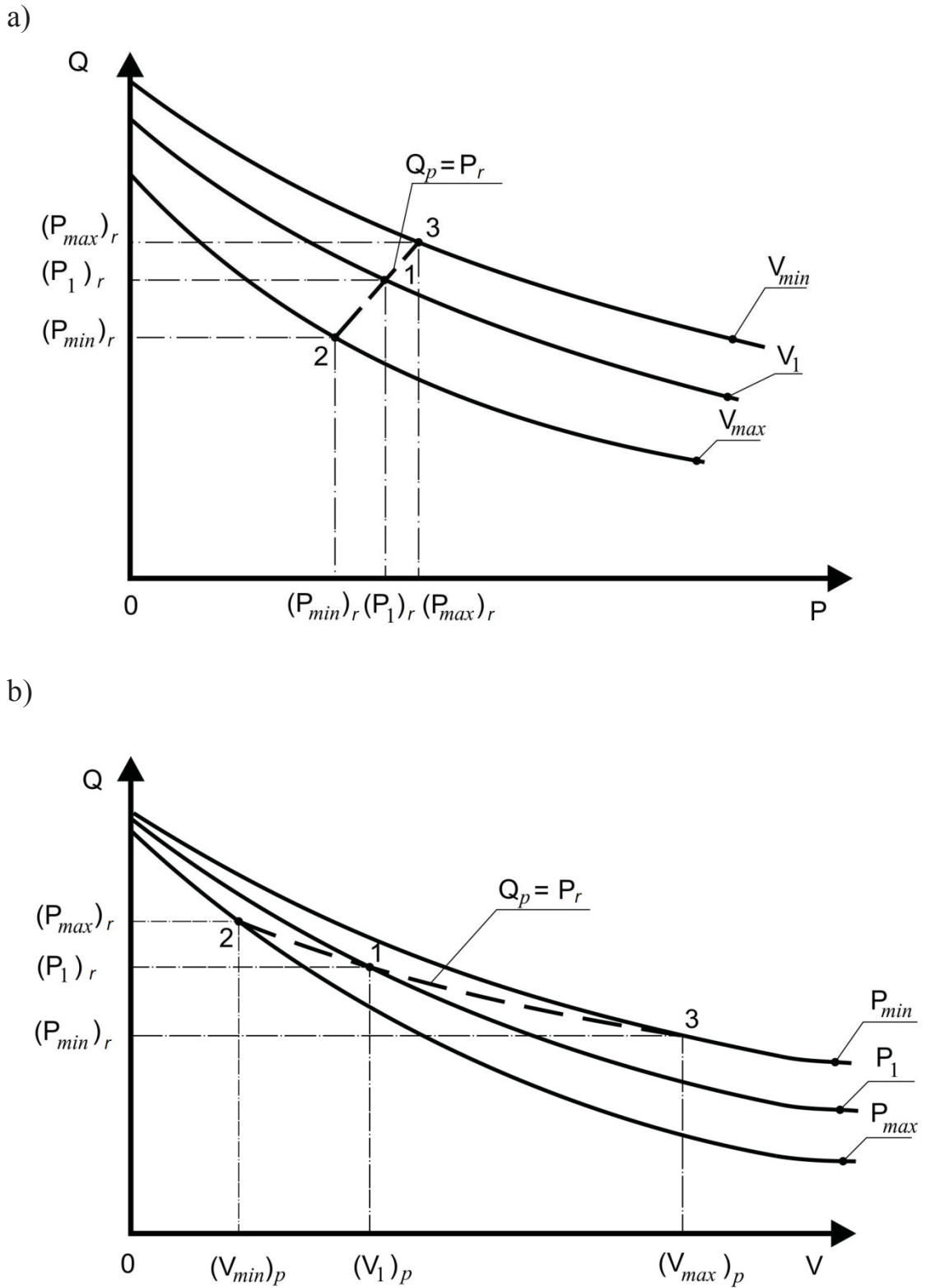


Fig. 5.6. Qualitative regularities of the influence of the resistance reaction P of the powered support on the load Q formation of downgoing rocks by the weight (a) and changes of the rational reaction of the resistance P_r , depending on the stope advance rate V (b)

This section were discussed general trends and qualitative regularities of the influence of main parameters on the loading of powered support. Each of the above regularities require the development and deepening into specific mining, geological and technical conditions of the stoping of flat coal seams. However, the tool of system planning of studies is received.

5.5. Conclusions

1) Was uncovered the mechanism of influence on the load formation on the support of geomechanical (structure and mechanical properties of roof rocks, their rheological properties) and technological (stope advance rate and its down-time, feed rate of the stoping machine and web width of its actuating element, step of the main roof caving and resistance reaction of the roof support) parameters of extraction works.

2) Considered general trends and qualitative regularities of the influence of geomechanical and technological parameters on the powered support loading are represented the tool of system planning of studies in mining conditions by the monitoring of the stoping equipment work and mathematical modeling of processes of extraction works with application of the modern computer technology and the latest software.

6. INVESTIGATION AND ANALYSIS OF THE INFLUENCE OF TECHNOLOGICAL PARAMETERS ON LOADING OF POWERED ROOF SUPPORTS

6.1. Regularities of pressure changes in hydraulic props of the support during different advance rates of the stope

The collection of information was carried out on indications of the pressure P_i in hydraulic props of each section along the entire length L of the longwall at different average daily advance rates V_a of the stoping face. General requirements are in need of the concretization in terms of significant restrictions (or complete elimination if it possible) of the influence of other factors under establishing regularities $P_i(V_a)$ for a separate section and $P(V, Z)$ for the powered support along the entire length L of the longwall ($0 \leq Z \leq L$). Here are considered the following: it is necessary to use such pressure indications P_i for different rates V_a at which all other parameters in the multiparameter connection equation will be constant or varies insignificantly. Multiparameter connection represented by the function:

$$Q = \Phi_1(V_a, Z, X, C, l, H, \delta_i). \quad (6.1)$$

Then it is possible to determine the true relation between the pressure in hydraulic props and the advance rate of the stoping face. In order to satisfy the formulated assumption, possibilities of the influence limitation of each of the parameters in equation (6.1) were analyzed: X - coordinate of the position of the longwall in the extraction area range, from which depend the structure C of above-coal strata (with almost constant mechanical characteristics of the lithological variety), the depth H of the development and the distance to the last main roof caving (the influence of spacing l of the main roof caving); δ_i - distance of the section to the working face, which depends from the value of the spreading action of the hydraulic jack rod of the support unit. The position of the support unit towards to the working face is repeated frequently (from 6-8 to 14-17 times per day). Therefore, during the

one day is selected three identical provisions of units and is calculated the average value of three measurements of the pressure P_i in hydraulic props for a given average daily advance rate V_a of the stoping face. Then, in the next few days for a different value V_a sampling of the pressure data P_i is repeated. The change in the depth H of the longwall location is insignificantly (about 0,2-0,5%), the structure of above-coal strata is also changed unessential, and the distance to the last main roof caving is magnitude of the same order with the value of the daily face advance. Therefore, the compromise solution is accepted for the parameter l . If the structure of the above-coal strata in the range of two or three steps l of the main roof caving is relatively constant then pressure readings P_i are used for those days when the longwall backward movement from the previous main roof caving was approximately similar with previous measurements. If the structure of the above-coal strata varies significantly then the measurement P_i is not taken into account during the construction of functional dependence $P_i(V_a)$, but will be used during the construction of the function $P_i(C)$. In the last case other data measurements P_i are selected at the approximately constant structure of the above-coal strata and the identical longwall backward movement from the previous main roof caving.

By the condition of the minimum influence of semi-constant parameters C, H and l were constructed a family of graphs of dependencies $P_i(V_a)$ and $P(Z, V_c)$, during the analysis of which were identified a number of features. Figure 6.1 shows the dependence of pressure variation in hydraulic props of sections along the length of the longwall, where at first glance, there are no clear regularities of the influence of any parameter. The dependence $P(Z)$ must be converted minimizing the impact of all other parameters (except Z or, as shown in Fig. 6.1, numbers of sections along the length of the longwall), transferring them into the category of semi-constant by the selection of appropriate periods of the time of readings P_i sampling. Here, the first stage is the division of all sections into groups according to their position δ in relation to the working face (the value of extension of the hydraulic jack rod, plus the constructive distance in 317 mm between the face and the edge of the console of the

section canopy). In this case, the minimum distance between sections and the working face is $\delta_{min} = 317 \text{ mm}$ and the maximum is equal to $\delta_{max} = 1067 \text{ mm}$. In

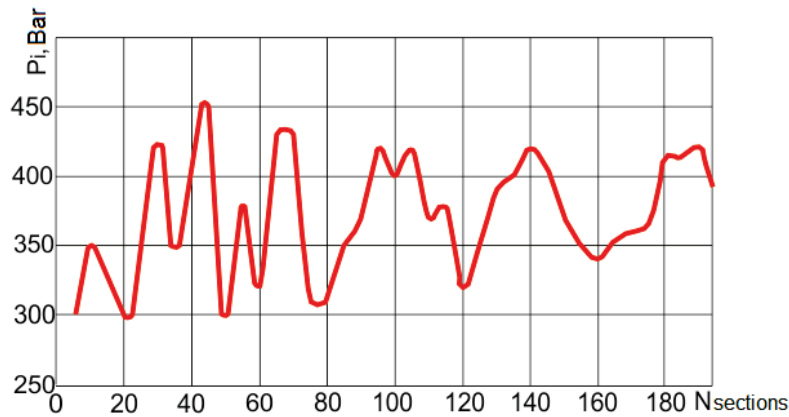


Fig. 6.1. Pressure variation P_i in hydraulic props of sections along the length of the longwall

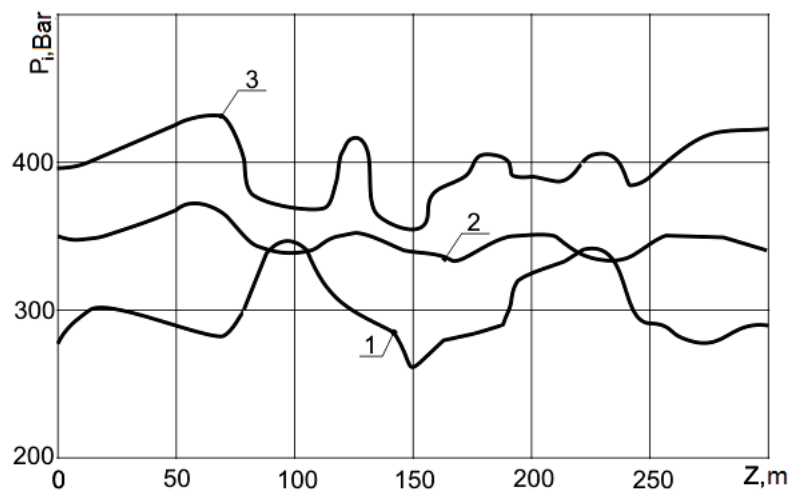


Fig. 6.2. Pressure variation P_i in hydraulic props of sections along the length Z of the longwall depending on their distance to the working face:

$$1 - \delta \leq 0,5 \text{ m}$$

$$2 - 0,5 \text{ m} < \delta \leq 0,8 \text{ m};$$

$$3 - \delta > 0,8 \text{ m}$$

this range sections is subdivided into three groups: $\delta \leq 0,5 \text{ m}$; $0,5 \text{ m} < \delta \leq 0,8 \text{ m}$; $\delta > 0,8 \text{ m}$. With such division of the function $P_i(Z)$ the length of the period is increased, the vibration amplitude along the length of the longwall is reduced and the function becomes smooth (Fig. 6.2) and informative for the analysis of the connection of parameters P_i and Z at different values of δ . Analysis of the dependencies $P_i(Z)$ gave results, which differ somewhat from existing concepts about features of the formation of the load along the length of the longwall [41-43].

Traditional schemes of the displacement of above-coal rock mass are based on the representations about unequal distribution of the

load Q_i on the roof support along the longwall. In the vicinity of extraction drifts from the side of the tight rock (the side edge parts of the seam) is formed the bearing

pressure area ($\delta_y > \gamma H$), which is significantly exceed the initial value of the vertical stress $\delta_y = \gamma H$ (γ - average volume weight of above-coal strata rocks). At face ends of the longwall rocks is unloaded ($\delta_y < \gamma H$). The load is increased, while the longwall is moved to the central part, due to the heightening of the area of complete displacement. Such representations were formed many decades and generally reflect the mechanism of rock pressure manifestations. However, in the specific case of the plough face with high speeds of its advance a number of features are arisen, which are caused by two reasons.

Firstly, all anomalous zones of rock pressure are formed during a certain period of time. For example, the dome of complete displacement of the above-coal strata is closed at a distance from the stoping face and the higher the advance rate V_a the farther from the face this area is finally formed. In any case, in the classical scheme of the above-coal strata displacement are considered anomalous zones already behind the longwall (excluding the advanced zone of the bearing pressure) in the goaf. In the studies was measured the pressure P_i in the hydraulic props directly in the workspace of the longwall, where flows only the second period (the first - the formation of the zone of the bearing pressure in front of the longwall) of lowering of the above-coal strata and its stratifications. Here probably will be ripping of the immediate roof, and in the main roof will be cracks nucleation and caving that form the swivel-block structure. In addition, should be considered a restraining influence of the working face on the lowering of rock layers of the roof. Therefore, the pressure distribution $P_i(Z)$ in the hydraulic props of the roof support along the longwall will differ from the existing concepts of the nature of the change in vertical rock pressure $\delta_y(Z)$ along the width of the extraction area.

Secondly, the face line during the plough winning has a convex shape, which is more stable in terms of resistance to the increased rock pressure in the central part of the longwall. Here the increase of the dome height of complete displacements increases the rock pressure, to which is successfully resisted the most convex part of

the face that pushes the end of the formation of the dome of complete displacements in the depths of the goaf.

Special attention is paid to mentioned factors because of the heterogeneity of load distribution $Q(Z)$ along the length of the plough longwall. Example of such a distribution is shown in Figure 6.1. Here could be argued that the maxima and minima of the pressure in hydraulic props of sections can be located on any parts along the length of the longwall. The division of powered support sections into groups according to the position in relation to the working face (see Fig. 6.2) in some extent smoothes the function $P(Z)$. However, stably repeating regularities of the pressure variation were not revealed. But in the parameter δ is detected constantly operating regularity of the low pressure P_i in removed (to the working face) sections and the pressure growth with an increase of the backwardness δ of sections from the face. For the first group of sections ($\delta \leq 0,5 m$) the range of pressure variation is $P = 262 \dots 345 \text{ Bar}$, for the sections that occupy an intermediate position ($0,5 m < \delta \leq 0,8 m$), the pressure increases to $P = 336 \dots 374 \text{ Bar}$, and its maximum $P = 353 \dots 432 \text{ Bar}$ is observed for lagging behind the working face sections at a distance of $\delta > 0,8 m$. This trend points to an active process of rock layers lowering (not only the immediate, but also the main roof) in the working space of the longwall, where a small backlog of sections from the working face causes significant growth of the load. The main reason for this phenomenon is weak, mostly thin-bedded, rocks of the above-coal strata with intense fracturing in the presence of watered lithological differences that must be considered in the developed geomechanical model of displacement of the rock mass during the stoping.

Disturbances of the function $P(Z)$ from the average value are 5-14% and are caused by changes in the structure of the above-coal strata along the longwall. Therefore, it is impossible to specify on the most dangerous areas of rock pressure manifestations along the longwall and in this sense the Z coordinate of its length can be excluded from the group of influencing factors, and geomechanical model can be simplified by the reflection of an arbitrary site of the length of the longwall.

The second part of researches of this stage has been devoted to the identification of regularities of pressure changes P in head ends of hydraulic props of sections, depending on the average daily advance rates V_a the stoping face. In order to obtain

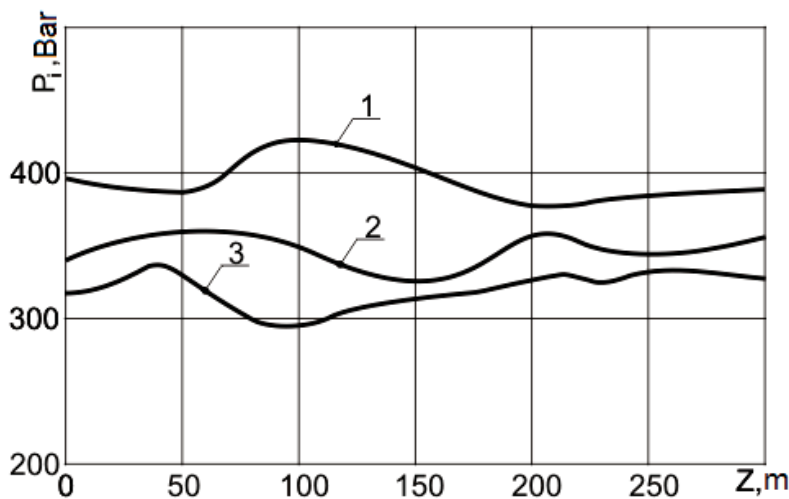


Fig. 6.3. Variation of average values of pressures P in hydraulic props of sections along the longwall length Z from average daily advance rates V_a of the stoping face:

$$1 - V_a = 3,2 \text{ m/day};$$

$$2 - V_a = 7,7 \text{ m/day};$$

$$3 - V_a = 11,4 \text{ m/day}$$

general regularities $P(V_a)$ was carried out averaging of the pressure P_i by groups of adjacent sections in the number of 12-15 units. These groups include sections with a different position relative to the working face, which eliminates the pressure jump P_i (on a short site along the Z coordinate) and in the integral form gives an indication of the degree of influence of the loading rate V_a on the supports. In Figure 6.3 are shown graphs of changes in

the averaged pressure P (by groups of sections) along the length Z of the longwall, depending on its average daily advance rate V_a , which was managed to record by three values (3,2; 7,7; 11,4 m/day) under the minimal influence of other factors. The regularity is such that with the V_a increase the pressure P in hydraulic props of the sections groups is reduced along the entire length of the longwall. Moreover, in the interval $V_a = 3,2 \dots 7,7 \text{ m/day}$ takes place more significant reduction of P than in the interval $V_a = 7,7 \dots 11,4 \text{ m/day}$. In that way, in the central part of the longwall is observed the reduction of P up to 15-27% in the interval 3,2-7,7 m/day and around 2-4% in the interval 7,7-11,4 m/day. But such an overwhelming excess of the gradient of P changes at relatively low advance rates V_a is observed not in all areas along the length of the longwall. In the area of sections № 25-60 and № 120-150 the gradient

of the pressure reduction P approximately the same on the entire interval $V_a = 3,2 \dots 11,4 \text{ m/day}$ and equal to $7,0\text{-}14,8 \text{ Bar}$ per 1 m/day of the increase of advance rates of the longwall.

The stable regularity of the pressure reduction P under the increase of V_a is still has a tendency of flattening with the growth of V_a . In other words, the function $P(V_a)$ displays the properties of the asymptotic approach to some constant value at high speeds, which can be observed in Figure 6.4. In Figure 6.4 is allocated several sites along the length the longwall (the central part and face ends), where the dependence $P_i(V_a)$ is divided for the removed to the working face sections ($\delta \leq 0,5 \text{ m}$) and unremoved sections ($\delta > 0,8 \text{ m}$). Here the influence of the parameter δ takes place in a part of different intensity of the reduction P_i with increasing velocity V_a - the gradient of the pressure reduction at unremoved sections is significantly higher.

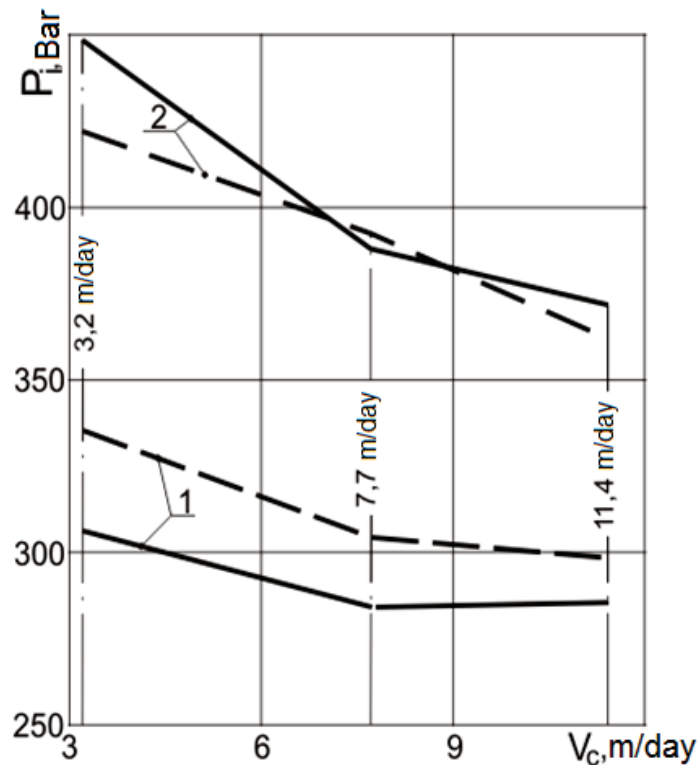


Fig. 6.4. Pressure variation P_i in hydraulic props of sections, depending on the average daily advance rate V_a of the stoping face:

- the longwall middle;
- - - face ends;
- 1 - $\delta \leq 0,5 \text{ m}$;
- 2 - $\delta > 0,8 \text{ m}$

face. The increase in speed V_a does not allow to realize this lowering in the fullest extent, because the section of the roof support manages to move to a new site. This regularity is confirmed by concepts of existing ideas about the processes of the above-coal strata displacement in terms of development through time and space of

roof rocks subsidence and their stratification during the longwall withdrawal. Indeed, with a small difference (0,3-1,0 m) of the lagging behind from the working face, the pressure reduction gradient of unremoved sections is 7,3-9,3 Bar per 1 m/day. The same gradient for removed sections is reduced to 2,6-4,8 Bar per 1 m/day, that is on average in 2,0-2,5 times. This experimentally established regularity must be considered in the computer modeling of geomechanical processes of the above-coal strata displacement. It is also necessary to take into account the fact that while the advance rate V_a of the stoping face is increased, the intensity of the load reduction on the sections is decreased. If this regularity is extrapolated, then, obviously, there is a value V_a , at which the advance rate of the stoping face is no longer affect the formation of the load on the support.

In conclusion of the analysis of the results of pressure monitoring in hydraulic props of the roof support should be evaluated the degree of their stress loading in relation to the value of the bearing capacity in accordance with the characteristic of DBT roof support. For the predominantly removed sections ($\delta \leq 0,5 m$) the pressure P_i is ranged within 58-77% from the maximum (relief valve operating pressure of hydraulic props). For the predominantly unremoved sections ($\delta > 0,8 m$) pressure is 78-96% from the maximum at achieved sufficiently stable velocity of the face advance $V_a = 7,7 m/day$. Reducing of the advance rate of the longwall leads to the relief valve action on the individual unremoved sections and they turn into the yield mode, loading the adjacent sections. Under the averagely-stable (7,7 m/day) and high (11,4 m/day) advance rates of the working face is remained the reserve of the roof support increasing resistance since the average pressure in hydraulic props is accordingly 71-80% and 66-75% from the maximum.

6.2. Monitoring of the interaction of roof support sections with rocks of the roof along the length of the longwall

Analysis of the pressure P_i in hydraulic props of the powered support along the length Z of the longwall was performed under different operating modes of the stoping equipment and the mining and geological situations in the longwall vicinity

that are changed over time and space. The objective was to establish regularities of pressure variation P in hydraulic props along the length Z of the longwall. For this purpose was gathered an extensive database, which is structured as follows:

- Detailed analysis of indications for a full day of work of the longwall set of equipment includes at least eight of its provisions (every three hours). This allows to limit the influence of changes of geomechanical factors: the structure and properties of the above-coal strata, the degree of its water content, depths of the longwall location and the like.

- During each month of the work on the working area was carried sampling of the information for a full day. At the same time, there is a periodic change in the structure of the main roof and some stable decrease in depth of the longwall location in relation with the panel development to the rise of the seam. In addition, technological parameters of the longwall set of equipment (the average daily advance rates of the stope and duration of its stops) and the position relative to the time of main roof caving were varied in a wide range.

With this approach to indication reading of pressure P is excluded the possibility of subjective evaluation of sections interaction along the length of the longwall due to the neglect of the influence of any significant factor. However, data processing and analysis of the pressure distribution in the hydraulic props along the length of the longwall have not revealed any stable regularities of the function $P(Z)$, which is associated with the coordinate Z of roof support sections location along the longwall under variations of the geomechanical and technological parameters of the stoping. In addition to what has been said previously possible to add:

- Firstly, the restraining influence of the convex shape face affects the value of the roof lowering in the workspace of the longwall and the diagram of rock pressure distribution along its length. The function $\delta_y(Z)$ is characterized by known areas of unloading in the central part; but, due to the working face restraining effect these differentials of the diagram $\delta_y(Z)$ have much smaller amplitude than in the goaf.

- Second, despite the influence of the face, lowering the immediate roof in the work area of the longwall reaches 100-150 mm or more. So, sections of powered support running on full capacity regardless of their position in the coordinate Z . Indeed, constant resistance mode of any hydraulic prop is designed in such a way that the section has been endowed with a certain yielding property (eliminate the lowering the roof is technically impossible), but at the same time its reaction of the resistance was close to the maximum. During the yielding, the load balancing Q occurs between adjacent sections that increases their resistance to the rock pressure. Operation characteristic of hydraulic props (the connection of pressures in the head end with the value of yielding) and sections generally indicates a sharp rise in pressure during the exhaustion of technical and technological gaps, cavities and the like. Usually, the total yielding of the hydraulic prop by increasing the density of the working fluid in the initial period of thrust, compensation of gaps and irregularities on the contact of the canopy with the roof and the foundation with the bedrock is up to a few tens of millimeters. After that pressure relief valves are triggered and the section turns into yielding mode, which close to the mode of constant resistance. Regardless from the value of the subsidence of immediate roof rocks along the length of the longwall (the lowering will be intermittent with the maximum in the middle of the longwall) the transition of sections into the yielding mode with the resistance maximum is possible on any section. From here there is a phenomenon of the relative independence of increased resistance reaction in sections from their location coordinates Z under the expressed non-uniformity of the roof lowering along the length of the longwall. Pressure differentials in hydraulic props of sections are connected with the process of the advance during the stoping and are caused by technological parameters. Therefore, the reduced resistance reaction (immediately after the advancing) is not connected with the coordinate Z of the position of sections along the longwall.

- Third, the analysis of emergency situations related to landing on “the rigid base” of powered supports has shown that the beginning of this process is not related to the length of the longwall. It is caused by other geomechanical and technological reasons: the bearing pressure from the previously worked-out

extraction sites, more intensive inflow of the water, changes in the structure of the roof of the seam, the down-time of the complex, fault conditions of the support, caving of the main roof and the like.

On the basis of wide-ranging observations the conclusion was made that the load on the powered roof support sections is independent from the coordinates of their location along the length of the longwall, and is determined by geomechanical and technological parameters, which includes the technical condition of the hydraulic equipment of each section.

6.3. Investigation of the influence of speed and width of coal shearer cutting on roof supports loading

Mining experience of 161st longwall of mine “Samarskaya” recommends to set penetration depth of knives (cutters) in the range of $B = 50 \dots 70 \text{ mm}$ for ensure reliable and non-stop operation of the plough. Thus, the web width of the plough is captured and its stabile and narrow range of vibrations can not affect the change of pressure P in hydraulic props of the roof support. Then, for the experimental study remains one parameter, it is feeding speed V_k of the plough. However, the identification of dependence $P(V_k)$ is principally possible only under the ensuring variation of the speed V_k of movement of the plough during the coal cutting process. Under our conditions on the technical characteristic, the plough has only two constant feed rates (0,66 and 0,92 m/s). Moreover, throughout almost the whole period of the mining of 161-st longwall was used only the first feed rate. Its average value (including stops) for any period of time does not allow objectively evaluate the influence V_k on the wave of stresses in the working area of the plough. Consequently, both studied parameters B and V_k are fixed values and do not allow experimentally to identify regularities of their influence on the pressure P in hydraulic props of the roof support. Here it is necessary to use a second direction of research - computer modeling of the mining processes of 161-st longwall.

Methodology of uncovering stresses wave is structured as follows. In the first stage is analyzed the influence of position of the plough under various structure of the

roof, stope advance rates V and position δ of sections. At the same time graphics $P(Z)$ of pressure changes along the length Z of the longwall are compared with those in the area of the plough movement with the objective to determine the extent of the anomaly of rock pressure by analogy with existing studies [42, 50, 53, 57] of stresses wave during the operation of the cutter-loader. Indications of pressures in hydraulic props of sections are averaged (under fixed parameters of the structure C of roof rocks, the advance rates V of the stope and the position δ of the section) by two variants: by the length of the longwall except the working area of the plough; by the nearby areas of the longwall with length of 15-20 m on both sides from the working area of the plough for the reduction of influence changes in the structure of the roof along the length of the longwall. At the second stage, in case of detection of significant pressure P perturbations in the working area of the plough, is formulated the problem of the detection of regularities of the influence of advance rates V of the longwall on rock pressure anomalies parameters: ΔP value of the average deviation of the pressure from the total average value and the width Z_{pl} of the longwall section, where the influence of the plough is manifested. It is obvious that along the length Z_{pl} of the longwall pressure P_{pl} anomalies must also be averaged over extreme values δ of the position of support sections relative to the wall face. Then in the interconnected parameters P_{pl} and Z_{pl} there is a tendency: the more sections are involved in the site Z_{pl} of the longwall length, the smaller will be the deviation ΔP , which is equal to the difference between the average pressure P_{pl} in the plough area and the average pressure P_a along the longwall. Here is reached the decision to include in the zone of plough influence only those groups of sections (or individual sections), where the deviation ΔP will be exceed the value P_a not less than on 40 Bar. That is approximately on 10% higher than average pressure readings over an extended period of measurements. In further studies, the value ΔP of the deviation may be adjusted.

According to the described methodology on the first stage the analysis of stresses wave parameters gave following results. Identify the stress wave in the working area of the plough actuating element has been very difficult for two reasons:

- 1) weak influence on pressure P changes of the stress-strain state of the longwall roof under the plough web width $B = 50 \dots 70 \text{ mm}$ and the feed rate $V_k \approx 40 \text{ m/min}$;
- 2) pressure variations in the hydraulic props along the length of the longwall that arising from the other factors

This situation is repeated consistently in all studied areas by varying the geomechanical parameters, but there is one exception. At the mid and large layered structure of the main roof (areas №1, №3 and №5, see Fig. 7.1) during the time that preceding its caving, the wave of stresses in the working area of the plough actuating element is still manifested. In Figure 6.5 are shown graphs of pressure changes $P(Z)$

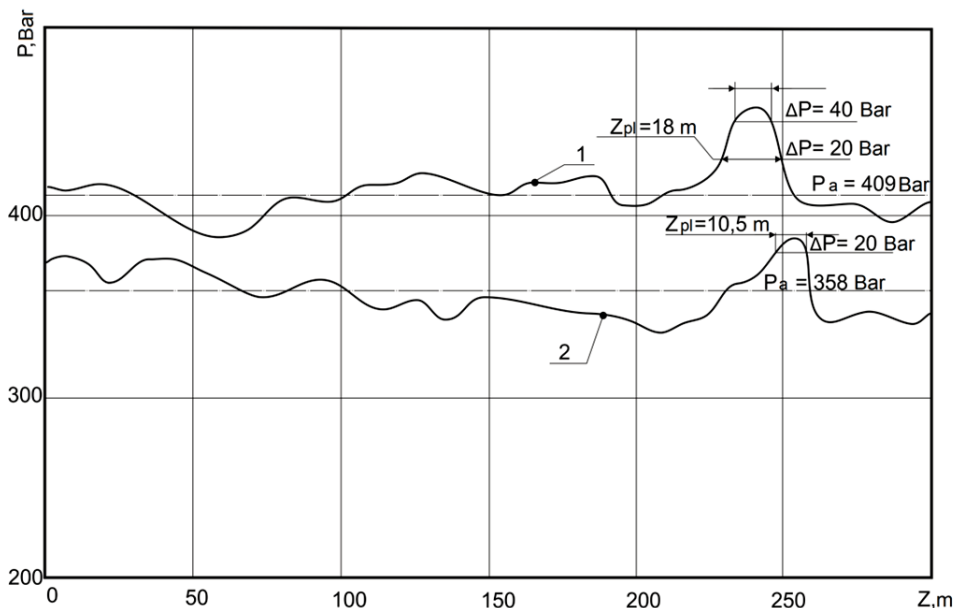


Fig. 6.5. The pressure change in the working area of the plough actuating element on the site №5 of the extraction panel length before the main roof caving:

$$1 - \delta \leq 0,5 \text{ m}$$

$$2 - \delta > 0,8 \text{ m}$$

parameters ΔP and Z_{pl} of the stress wave in the working area of the plough actuating element for sections, which is distant from the wall face. Here at the average value of

along the length Z of the longwall on the site №5 before caving of the main roof. The diagram $P(Z)$ is divided into two extreme groups by the position δ of sections relative to the face and averaged by groups from 3-4 adjacent sections. With this approach, quite clearly are evaluated

the pressure $P_a = 409 \text{ Bar}$, that is $\Delta P = 48 \text{ Bar}$. The area of overpressure $\Delta P = 40 \text{ Bar}$ is distributed on the length $Z_{pl} = 12 \text{ m}$, and at $\Delta P = 20 \text{ Bar}$ on $Z_{pl} = 18 \text{ m}$, which is characterized by attenuation of the stresses wave. For sections that pulled to the wall face (graph 2 Fig. 6.5) the average pressure for the longwall is $P_a = 358 \text{ Bar}$, and the maximum $P = 386 \text{ Bar}$ has approximately the same coordinate Z as for unremoved sections. Excess of the pressure $\Delta P = 28 \text{ Bar}$ is reduced that is quite explainable by the influence of the face, which restrains of the roof lowering. So here it is possible to evaluate only the length of the area the overpressure $\Delta P = 20 \text{ Bar}$, which is $Z_{pl} = 10,5 \text{ m}$ and significantly lower than for unremoved sections by the above mentioned reasons.

6.4. Conclusions

At high feed rates and small web width of the plough actuating element the wave of stresses in the area of its work is significantly affected the pressure change in hydraulic props and the load increase on roof support sections. This fact is consistently manifested in a wide range of variation of geomechanical and technological factors.

The exception is non-dominant condition of the main roof caving period that is provided by medium- and large-layered structure. Here was recorded overpressure up to 10-12% in the short segment of length up to 12 m (up to 8 sections) and only in unremoved to the working face sections.

As a result, it could be argued that constructive and technological parameters of the plough ensure the stability of manifestations of the rock pressure without the occurrence of any significant anomalies in the area of operation of the actuating element. Influence of the feed rate and web width of a coal-mining machine, as well as other parameters in a wide range of changes are the most appropriate to study by numerical methods of computer modeling. Often, computational experiment is the only means of the analysis of geomechanical processes in terms of reasonable and authentic division of the influence of individual parameters on the development of the load on the powered support.

7. RESEARCH AND ANALYSIS OF INFLUENCE OF GEOMECHANICAL FACTORS ON THE LOAD OF POWERED ROOF SUPPORTS

7.1. Influence of the structure of coal contain rock strata on roof supports loading

In accordance with the outlined methodology of the pressure data sampling in hydraulic props of roof support sections were selected five sites along the length of the extraction panel, which are most appropriate for the detection of the influence regularities of the roof rocks structure and the depth H of the longwall on the formation of the load on the powered support. From the point of identifying of the separate influence of the roof structure were selected four sites, which are located pairwise close to each other (with the exception of H influence), but have a significantly different structure of the main roof up to the seam C_6^1 . The given capacity of the main roof by results of preliminary modeling of its displacement put the main contribution to the formation of the load on the powered support. For illustrative purposes, all five structures are shown in Figure 7.1. Here it is seen approximately constant thickness of researched the main roof, which is enclosed between seams C_6 and C_6^1 . Identification of the influence degree of the main roof structure is performed by pairwise comparison of the pressure P values in hydraulic props for sites №1, №2 and sites №4, №5. Sites №1 and №5 are characterized by more heavy layers (medium- and coarse-layered structure), and compared neighboring sites №2 and №4 are composed mainly of less heavy layers (mostly medium- and sometimes thin-layered structure). Determination of the influence degree of the longwall depth H is produced on three sites - №1, №3 and №5, where the main roof structure is characterized as medium- and coarse-layered with occurrence of the same lithological differences with approximately the same mechanical properties. It should be recalled that readings at selected sites were produced not less than 8 times per day with fairly constant ($\pm 10\%$) average daily advance rate stope.

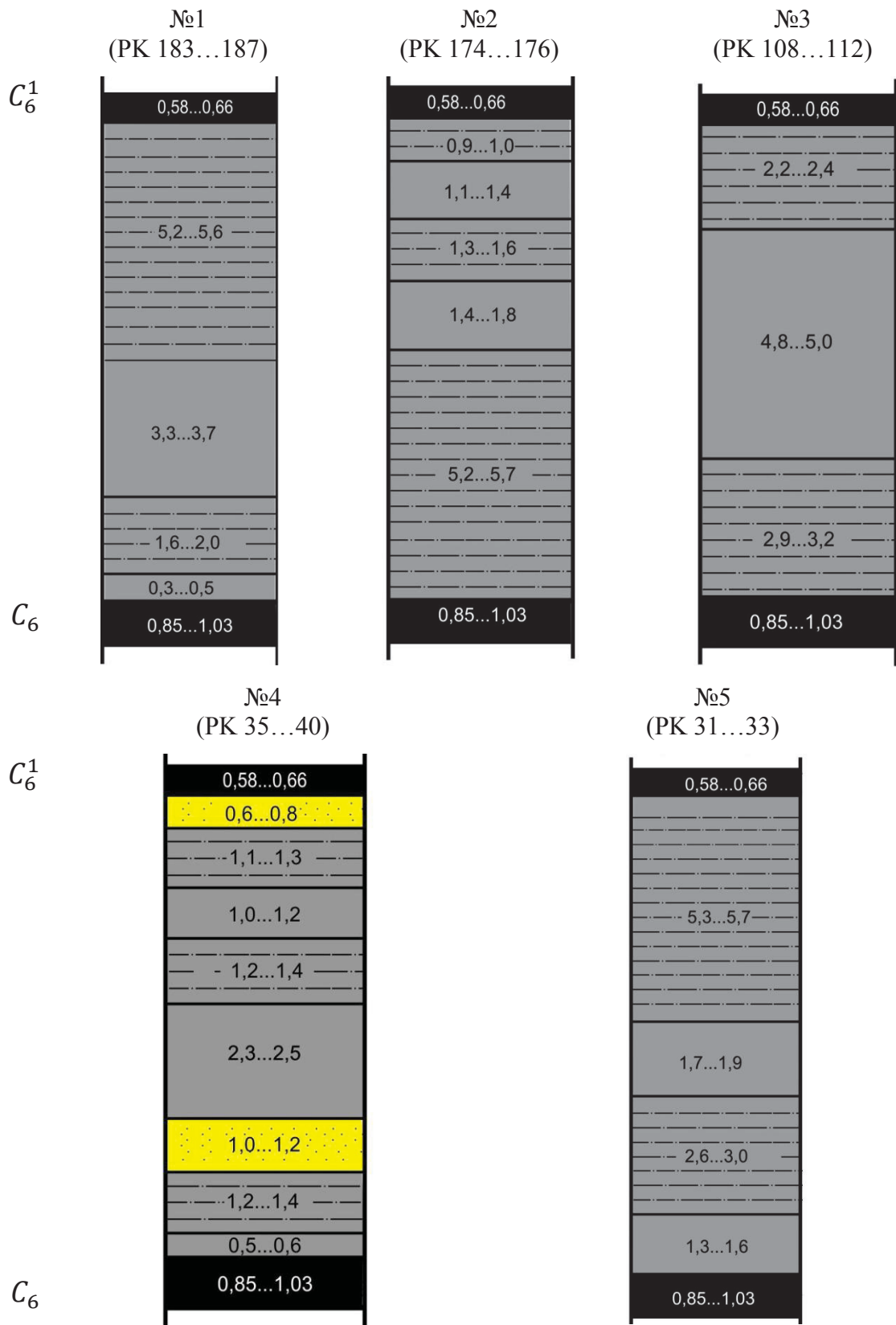
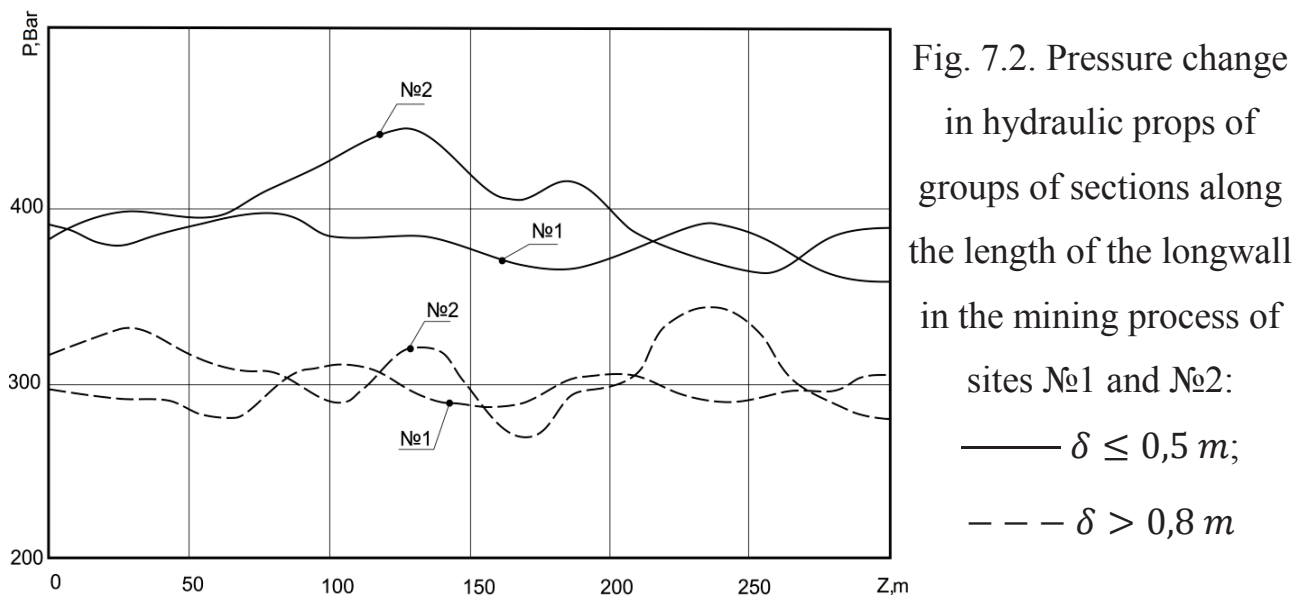


Fig. 7.1. Combined lithological columns of monitoring sites of the influence of geological environment on the formation of the load on the powered support

Analysis of indications $P(Z)$ for the first couple of sites №1 and №2 is made by the graphs in Figure 7.2 at the depth of the longwall $H = 429 \dots 437 \text{ m}$. Here, in order to exclude the effect of the position of sections relative to the wall face by the parameter δ graphics are built for two extreme values of $\delta \leq 0,5 \text{ m}$ and $\delta > 0,8 \text{ m}$.



For sections that are pulled to the wall face, the average pressure P_a along the longwall length in the structure №1 lower than in the structure №2 only per 3,7%, while the difference in the structure of the main roof at sites №1 and №2 is extremely essential (see Fig. 7.1). The value of the maximum P_{max} for the structure №2 exceeds thereof for the structure №1 per 10,6%, but the minimum P_{min} was lower on 3,9% for the structure №2. These data indicate that in the region of the wall face the difference in the structure of the main roof at the height of 10 m has no significant influence (for the plough longwall) on the formation of the load on the support. Let us analyze the pressure distribution P in hydraulic props of sections that is distant from the wall face not less than by 0,8 m (see Fig. 10.2, solid lines). Here there is the situation similar to the previous. The average deviation of the pressure along the length of the longwall higher per 5,6% for the more compact structure №2. For the same structure the maximum P_{max} higher per 11,3%, and the minimum P_{min} is almost identical with the structure №1.

In such a way, there is an increase of the load on sections of the powered support for the more layered structure №2 (where the main roof is composed of

mudstones and siltstones with thickness of 1,0-1,5 m), relatively small thickness of the layers determines their more intensive caving on the support, and the height of the contour of the load formation is distributed slightly higher in roof. Nevertheless, the difference in pressure readings is not so significant as the difference in the structure of the main roof. To confirm this fact was carried out similar comparative analysis of pressure values for hydraulic props of sections on sites №4 and №5 that are located at depth $H = 342 \dots 350 \text{ m}$. Results confirmed the insignificant influence of the main roof structure on the formation of the load on the support.

During carrying out of above studies has been noted some feature of pressure changes P along the coordinate $X = 20 \dots 25 \text{ m}$ of the length of extraction area in conjunction with manifestations of the impact of the main roof-caving increment l (Fig. 7.3). In the description of the interaction mechanism of roof rocks with the powered support was noted that in the predominantly thin- and medium-layered soft roof rocks there is no significant in length hanging rock consoles, therefore

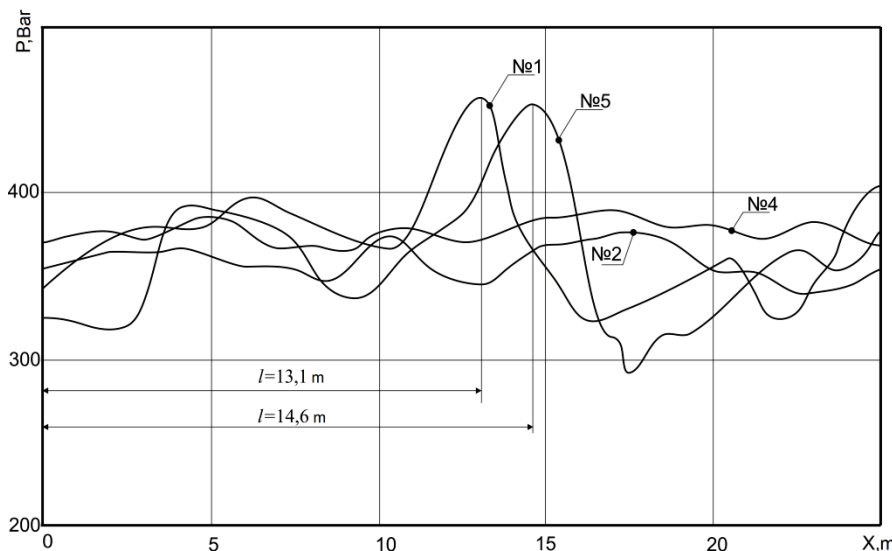


Fig. 7.3. Pressure change in hydraulic props of groups of sections during the mining of sites №1, №2, №4 and №5 with the corresponding structure of the main roof

manifestations of the roof caving are nearly invisible. The maximum amplitude oscillation of the function $P(X)$ is 10,1% for the site №2 and 6,6% for the site №4 at the length of indication reading $X = 25 \text{ m}$, so it can be argued about the practical absence of the phenomenon of main roof caving in conditions of studied structures №2 and №4. In the analysis of oscillations $P(X)$ within the detected range of amplitudes can be noted a certain periodicity of this process with the period length 6-7 m. By comparing this value with the length of the roof support section plus its

distance to the wall face (5-6 m in total), it can be argued that the collapse of the main roof (and the preceding to this load growth) occurs immediately after the section that confirms the developed scheme of the displacement of rocks of the above-coal strata in mines of Western Donbass.

However, in the medium- and coarse-layered structures of the main roof №1 and №5 are observed perturbations of the load on the section that are identical to the rise of the pressure function $P(X)$ in proportion to the mining of the studied sites. This “anomaly” is associated with the step l of the main roof caving, which is represented by relatively strong siltstone (hardness coefficient up to 4) with almost equal thickness from 5,2 m to 5,7 m for both sites. These characteristics of the siltstone allow to predict the hanging of the extended rock console, when above the siltstone is not expected substantial volume of falling rocks. Therefore, the intensive growth of the pressure P in hydraulic props of sections is explained by the increase of the length of the hanging console of the heavy layer of siltstone in the period before the main roof caving. Here at the site $X = 3 \dots 5$ m before the collapse, the rock console is almost completely lowered to the underlying rock and transfers through them the load on the powered support. In such a way, the combined action of the factor of increasing the length of the console (during the stope advance) and the factor of its continuous lowering with the load of the underlying rocks leads to a sharp increase in pressure in head ends of hydraulic props, which even by averaging of readings reaches the value $P = 450 \dots 460$ Bar of the relief valves action of hydraulic props - the support works with the maximum load. The transition of hydraulic props in the yielding mode does not cause complications, if the sufficient hydraulic props extension is provided. Otherwise, there is the danger of caving of sections on the “rigid base” with all following consequences.

After reaching the maximum $P(X)$ at the end of the main roof caving the pressure is sharply reduced and during the subsequent face advance up to 3 m reaches the minimum $P_{min}(X)$, value of which up to 25% less than $P_{min}(X)$ in sites №2 and №4 of the main roof of increased stratification. At the further face advance there is a gradual growth of the pressure, and the cycle is repeated. But because of the limited

length of sites structures №1 and №5 traced the stability of loading cycles of sections is not possible. Here it is necessary to note the following:

- The recorded step of the main roof collapse was $l = 13,1 \text{ m}$ for structures of the site №1 ($H = 434 \dots 437 \text{ m}$) and $l = 14,6 \text{ m}$ for structures of the site №5 ($H = 342 \dots 345 \text{ m}$).

- During the substantial change of the depth H of the longwall location, the difference in roof-caving increment was 11.5%, but the existing growth l points to the small effect of the development depth in terms of the impact reducing of the vertical rock pressure $\sigma_y = \gamma H$ on the console of the thick siltstone, which is due to the deflection “goes away” from high loads.

- The studied sites №1 and №5 are less common in a structure of above-coal strata, but they require special attention during the passage of the longwall: the development of a persistently high advance rate of the stope and the prevention of its prolonged stoppages.

- Fixed features of pressure changes $P(X)$ in the process of the extraction panel mining are comported with provisions of the developed mechanism of interaction of the above-coal strata with the powered support and are pointed to the connection ambiguity of the main roof structure with the process of the load formation on the support. This is strengthens the role of modeling of geomechanical processes as a tool for establishing these regularities.

7.2. Influence of the depth of stope location on roof supports loading

The ambiguity of the influence of the main roof structure is superimposed on the ambiguity of the influence of the depth H of placement of longwall. In Figure 7.3 in case of the comparison of average pressure readings P at sites №1 and №5 before and after the caving of the main roof their difference is set up to 6% during the depth reducing H up to 27%. For thin- and medium-layered structures №2 and №4 are observed “illogical” tendency of growth of the average pressure P (along the length of sites) up to 4.5% during the decrease in the depth of placement of the longwall up to 22%. This is connected with the existing differences between structures of the

roof: placement in the immediate roof of the site №2 the siltstone with thickness of 5,2-5,7 m; placement in the main roof of the site №4 two layers of the sandstone with thickness up to 1,2 m and the ability to form consoles of increased length.

In order to clarify the presence (or absence) of the connection of P and H , as well as its importance has been analyzed the influence of the depth of placement of the longwall with the assistance of the site №3 (see Fig. 7.1), which is located at the depth of $H = 385 \dots 386 \text{ m}$ and has the medium- and coarse-layered structure for the reliable comparison with indicators P on sites №1 and №5. According to results of the data sampling and their averaging by the length of the longwall and the position δ of section relative to the wall face graphs (Fig. 7.4) are given for various locations X

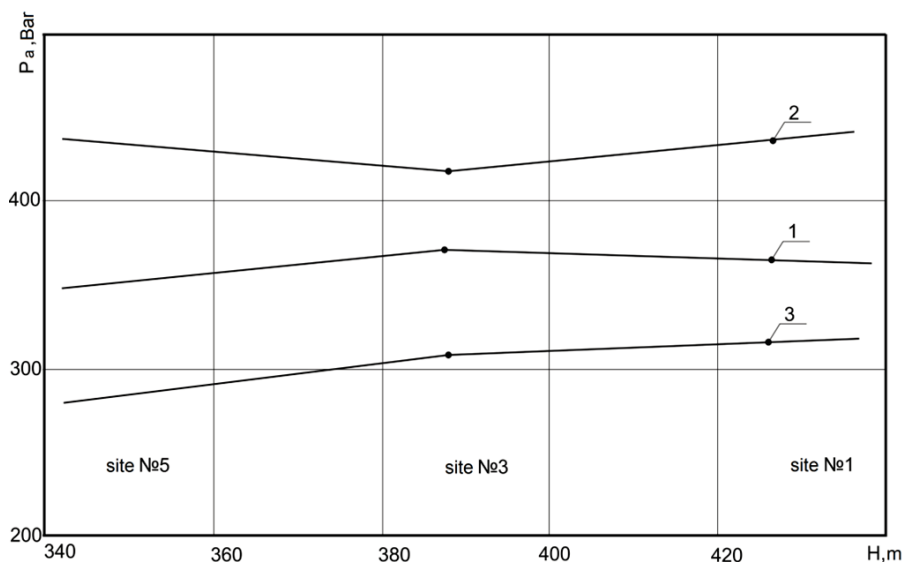


Fig. 7.4. Changes of the average pressure P_a in hydraulic props depending on the depth H of the stoping at the position X of the longwall relative to the step l of the main roof caving, which is represented by the medium- and coarse-layered structure: 1 - $X < l$; 2 - $X = l$; 3 - $X > l$

of the stope along the length of the extraction area. Graphical representation of trends of the pressure P connection in hydraulic props with the depth of the development underlines once again their ambiguity:

- The position of the longwall before the main roof caving ($X < l$) is characterized by a certain stability of the pressure in hydraulic props, followed by its increase in the period before caving (see Fig. 7.3). This provision reflects the line 1 of

Figure 7.4, where on the site №3 ($H = 385 \dots 386 \text{ m}$) is observed the growth of the average pressure P_a by 77% in comparison with the site №5 ($H = 342 \dots 345 \text{ m}$). However, during the further growth of location depth of the longwall (site №1, $H = 434 \dots 437 \text{ m}$) the pressure P_a is reduced by 3,0%, and in general with the increase of H on 92 m, it grows only by 4,6%, while the relative increase in the depth of extraction works is 26,7%.

- At the time of caving of upper layers of the main roof ($X = 1$) the regularity P_a from H is also practically not observed. At the site №3 there is the decrease of the average pressure by 4,2% (line 2) compared with the site №5. The further growth of the depth H has shown that at the site №1 the pressure P_a is increased by 5,8% in comparison with the site №3, and in general the difference between these parameters on sites №1 and №5 was 1,4%.

- After the main roof caving ($x > l$, line 3) there is the steady growth of the average pressure P_a with the increase in depth of the development H , however the gradient of this growth is quite low, so that the difference of P_a between sites №1 and №5 is 10,5%.

From the above analysis it is possible to make the main conclusion about the insignificant influence of the development depth H on the formation of the load on the support of the plough set of equipment in the studied of mining, geological and technical conditions. The explanation for this fact lies in the plane of the mechanism of layered lowering of the above-coal strata (see Fig. 4.4), when the load on the support is determined by the volume of rock inside the contour of the dome above the support. Here, the influence of H on the size of the dome is not so important.

7.3. Regularities of the influence of main roof caving step and stopping time of the stope

Main attention is given on areas with medium- and coarse-layered structure of the main roof, since a significant amount of researches on thin- and medium-layered structure has been executed earlier. The information in the process of the data gathering was generalized for all three sites (№1, №3 and №5) with medium- and

coarse-layered structure (see Fig. 7.1). As the result, the significant variation interval of average daily advance rates V_a from 2,6 m/day to 10,8 m/day was received. Also on graphs in Figure 7.5 was produced the separation of the regularity $P(V_a)$ on two groups of sites: immediately before caving of the main roof and on the rest of the length of the roof caving step with medium- and coarse-layered structure. As it is visible, both regularities tend to decrease pressure P in hydraulic props with the

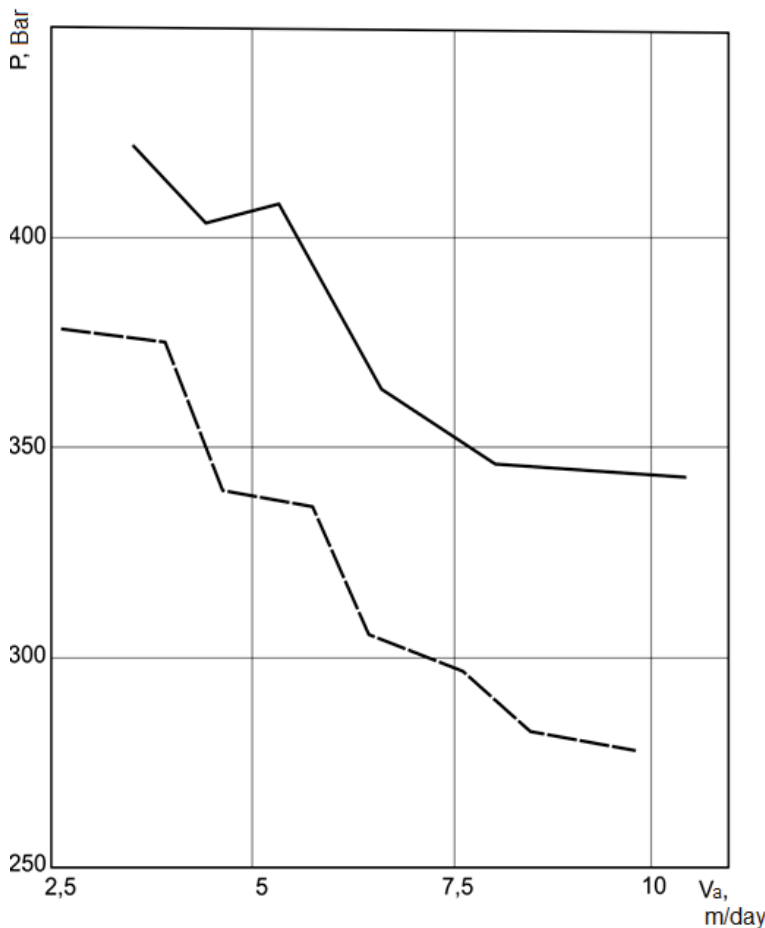


Fig. 7.5. Pressure change P in hydraulic props of adjacent sections depending on the stope advance rate V_a in areas with the medium- and coarse-layered structure of the main roof:
 — before its caving;
 - - - on the rest of the caving step length

increase of the stope advance rate V_a . Thus, on sites in front of the main roof caving the average pressure in the hydraulic props is reduced to 19% in the range of growth $V_a = 3,5 \dots 10,8$ m/day. On the rest length of sites is indicated the load decrease on sections (as the equivalent of the pressure P in hydraulic props) up to 27% in the range of increase of $V_a = 2,6 \dots 9,7$ m/day (Fig. 7.6). Obtained data correlate well with previously established trends of the variation $P(V_a)$ for the thin- and medium-layered roof, since the formation mechanism of this regularities is the one and caused by “retardation” in time of the caving of rock layers on the powered support under high feed rates, where the important role is played by the restraining influence of the

wall face. The difference is contained in the formation of the increased load up to 13-24% in the period that preceding the caving of the medium- and coarse-layered main roof. But regularly manifested fact of the degree reducing of V_a influence on the value of load on the support from advance rates of the stope $V_a \geq 8,0 \dots 8,5 \text{ m/day}$ focus the attention. Therefore, the most favorable mode of operation of the longwall set of equipment (in terms of reducing the load on the support) occurs in studied mining and geological conditions under the stope advance rate not less than 8 m/day.

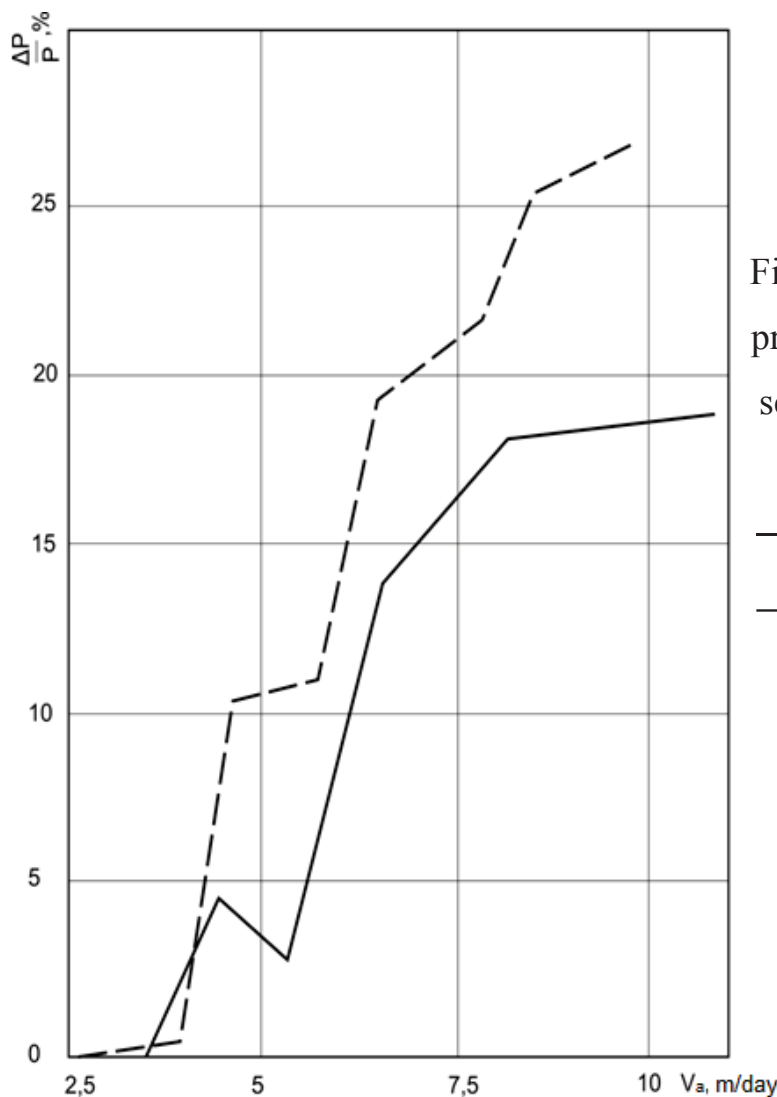


Fig. 7.6. The relative decrease of the pressure $\Delta P/P$ in hydraulic props of sections with increasing velocity of the stope advance rate V_a :
 — before the main roof caving;
 - - - on the rest of the caving step length

It is established that the greatest impact on the increase in pressure $P(t)$ during the down-time provide (in addition to geomechanical factors) the position δ of the section relative to the wall face and its advance rate V_a .

In Figure 7.7 are shown graphs of the growth of the average pressure P in hydraulic props during the down-time t of the longwall, which are divided by two parameters: the distance δ of the section relative to the wall face and the position of the face relative to the step of the main roof caving. Functions $P(t)$ increase with increasing of the down-time t : the most significantly - in the first 6 hours of the face

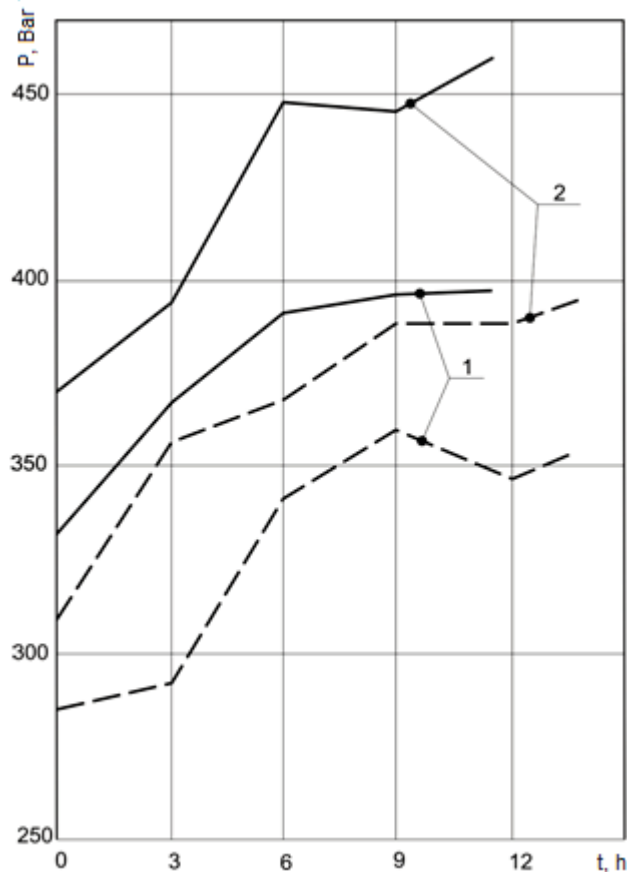


Fig. 7.7. The growth of the pressure P in hydraulic props of removed(1) and in unremoved (2) during the face down-time t :

—— before the main roof caving;
 - - - on the rest of the caving step length

stoppage, then the intensity of rise of the pressure (equivalent load) falls. The highest growth of the load is observed for unremoved sections in the period preceding the main roof caving. During this period, the relative difference between P for removed ($\delta < 0,5 m$) and unremoved ($\delta > 0,8 m$) sections increases from 8,0-12,1% at $t \leq 3 h$ up to 12,8-15,3% at $t \geq 9 h$, due to the more intense deflection of the roof layers. Also, in time t is increased the relative difference in the magnitude of the pressure caused by the main roof caving in comparison with the rest of the site length for unremoved sections. It varies from 12,7% at $t \leq 3 h$ up to 15,2-18,8% at $t \geq 9 h$, due to the more intense lowering of extended rock consoles and the development of this process in time until the moment of they caving.

Among removed sections, where a restraining influence of the wall face is large, the relative difference of the pressure P growth during the down-time t is stable enough: from 8,4-26,0% at $t \leq 3 h$ up to 10,9-14,7% at $t \geq 9 h$.

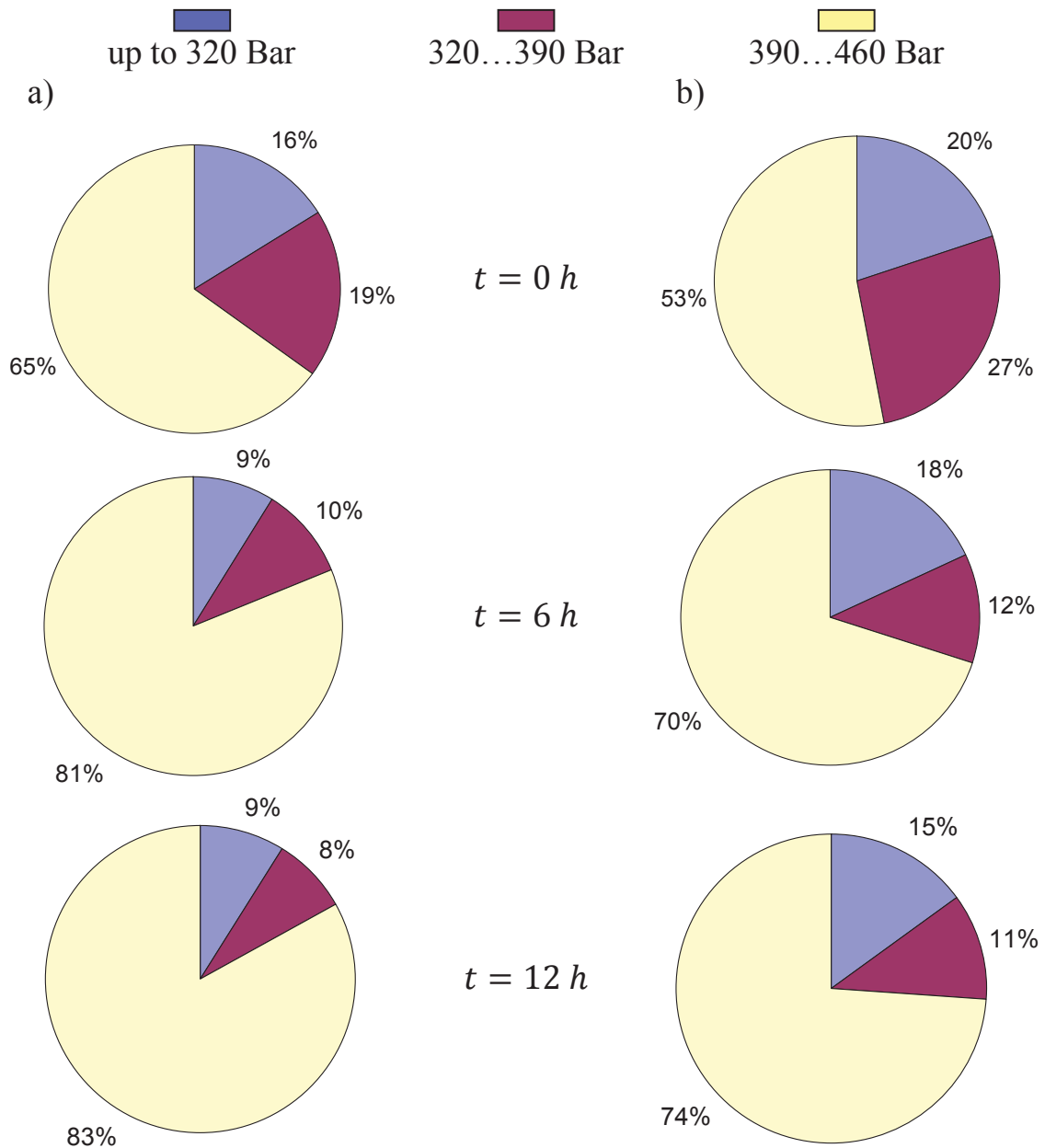


Fig. 7.8. The diagram of the pressure P rise in hydraulic props of sections during the down-time t of the longwall at two positions of its stoppage: a) before the main roof caving; b) on the rest of the caving step length

In the same way, interest is represented the statistical information regarding the distribution of P along the length of the longwall for two cases of its stoppage: before the main roof caving and on the rest of the caving step length. The trend of more

intense growth of the pressure in the first 6 hours of the longwall down-time is confirmed (Fig. 7.8). As you can see, with increasing duration of the down-time t of the longwall set of equipment is reduced the percentage of underloaded sections and is increased the proportion of sections, which are loaded over than 390 Bar (more than 85% of the maximum of resistance reaction). In this situation, at a more prolonged down-time can be expected the mass conversion of sections in the yieldable operation mode with decreasing of height of the longwall working space. In this scenario the landing of the longwall set of equipment on “rigid base” is possible. With the passage of these areas it is necessary to provide measures for the accident-free operation of the longwall, the main of which is to maintain the stable advance rate V_a of the stope before the main roof caving and on the rest of the caving step

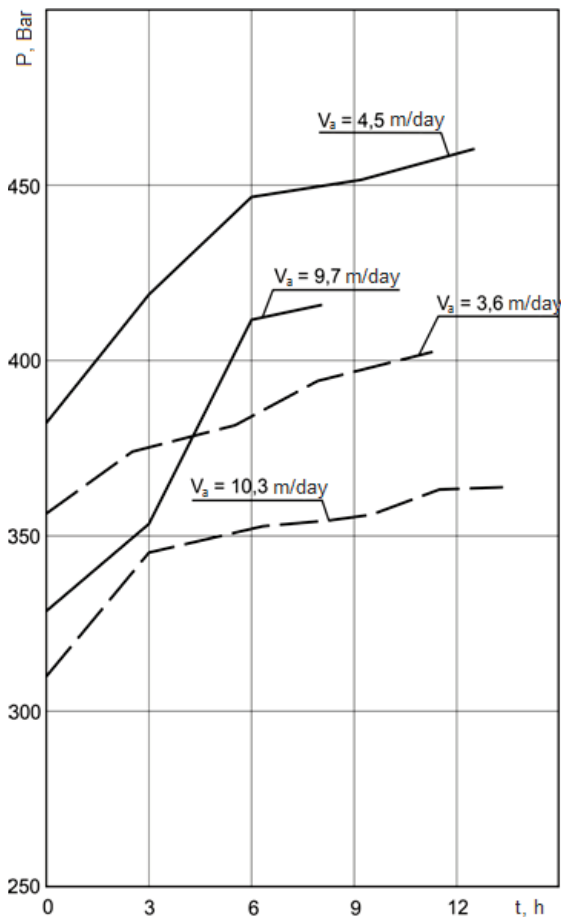


Fig. 7.9. Pressure P change in hydraulic props of sections depending on the stope down-time t in the medium- and coarse-layered structure of the main roof:
 — on the site before its caving;
 - - - on the rest of the site

length. As a result was established that in both cases the speed V_a significantly affects the process of the pressure P rise during the down-time t (Fig. 7.9). Thus, before the main roof caving the reduction of pressure P (at $V_a = 4.5$ m/day) was 16,1-18,7% in the first three hours of the down-time and about 8,5% at $t \geq 6$ h. As it is visible, is

observed the tendency of the reduction of the effect of V_a with the growth of duration of the down-time t , due to the influence of rheological factors. On the rest of the caving step length of the main roof of the considered structure the influence of stope advance rate V_a is changed. Comparing the two longwall advance rates (3,6 m/day and 10,3 m/day), is balanced the degree of influence of V_a during process of the face stoppage: in the first three hours of the down-time this influence was 9,0-14,8% and after 9 hours of the down-time was 11,4-11,7%. The given fact also is caused by the development of the process of rock consoles lowering on the support and with their relatively short length the load reduction (due to the increase of V_a) is compensated by the increase in the load during the down-time, which confirms the basic principles of the developed mechanism of the load formation on the powered roof support.

The general conclusion from the results of this part of researches is that the most dangerous situation is prolonged stoppage of the longwall just before the main roof caving with medium- and coarse-layered structure. It is recommended to pass these sites with a high advance rate of the stope for ensuring the reserve of the resistance reaction of the roof support in the case of down-time.

7.4. Conclusions

According to the results of the monitoring stage of operating modes of the plough longwall following conclusions were formulated:

- Information, from the workflow management system in the longwall, requires differentiated analysis. For this was developed the methodology of the influence separation of the main geomechanical and technological factors, grouping and systematization of parameter readings that characterizing the modes of operation of the equipment.

- Was established the feature of the pressure distribution P_i in hydraulic props of sections along the length of the plough longwall, which is expressed in the absence of steady regularity of the variation of function $P_i(Z)$ regardless from the rest influencing factors. Oscillations of average values of the pressure are 5-14% and are caused by changes in the structure of above-coal strata along the longwall.

- The division of sections on groups by the value of lag δ behind the wall face has allowed to establish the steady regularity of pressure increase P_i in the unremoved sections in comparison with sections that were moved to the wall face: the difference in magnitudes of pressures is about 20% over the entire length of the longwall.

- The average daily advance rate V_a of the stope consistently affects the pressure P_i in hydraulic props, regardless of the location coordinate Z of sections along the length of the longwall and its position δ relative to the wall face.

- In the studied range (3,2-11,4 m/day) of the average daily advance rate V_a of the stope is observed regularity of the influence attenuation of the parameter V_a on the pressure P_i in hydraulic props of sections at increased values V_a , and in general in the range of V_a the value of the pressure is reduced from 15 to 43 %.

- In order to establish the most objective regularities of load formation on the support, depending on the major influencing factors, the constant monitoring of the longwall set of equipment is required, which expands the database for future researches.

8. COMPUTER MODELING OF THE DISPLACEMENT OF COAL CONTAINING MASSIF OF SOFT ROCK IN THE VICINITY OF THE STOPE

8.1. Development and justification of computer model parameters

Accumulated results of the monitoring of the plough complex work in terms of the analysis of factors affecting the development of the load on the powered support, and fundamental principles of the developed mechanism of displacement of the above-coal strata indicate the need for a series of multivariate computational experiments on the basis of computer programs of the modeling of behavior of the geological environment. This will allow fundamentally expand the range of possible combinations of the geomechanical, structural and technological factors affecting the maintenance of the coal-face work since mine observations are characterized by extremely limited combinations of the above parameters and the difficulty of separation of their influence under the identification of a particular regularity.

Development of the model of geomechanical phenomena accompanying the stoping in the longwall includes the justification of parameters of the main components of the process that were previously noted during the disclosure of the mechanism of interaction of above-coal strata rocks with the roof support of the longwall set of equipment:

- structure and mechanical properties of the coal-containing rock mass within the spread of rock pressure anomalies caused by coal-face works;
- structure and mechanical of the mining out coal seam C_6 ;
- parameters of displacement areas of the above-coal strata: thickness and properties of the rock of the zone of the disordered collapse in the goaf, structure and characteristics of the contact elements of the constitution of the zone of the joint-block displacement of formed rock consoles outside the safe-guard of the powered support;
- geometrical and load-bearing parameters of the powered support;
- technological parameters of the mining, which are caused by the technical feature of the stoping equipment: the length of the longwall, its average daily advance rates, the feed rate of the coal-winning machine and the web width.

For each of these positions was carried justification of parameters of elements of the considered geomechanical system for the most adequate displaying of real conditions in the created model of the stopping processes.

According to the first block of positions the following researches were carried out. Results of the preliminary modeling of the coal-containing massif showed that by

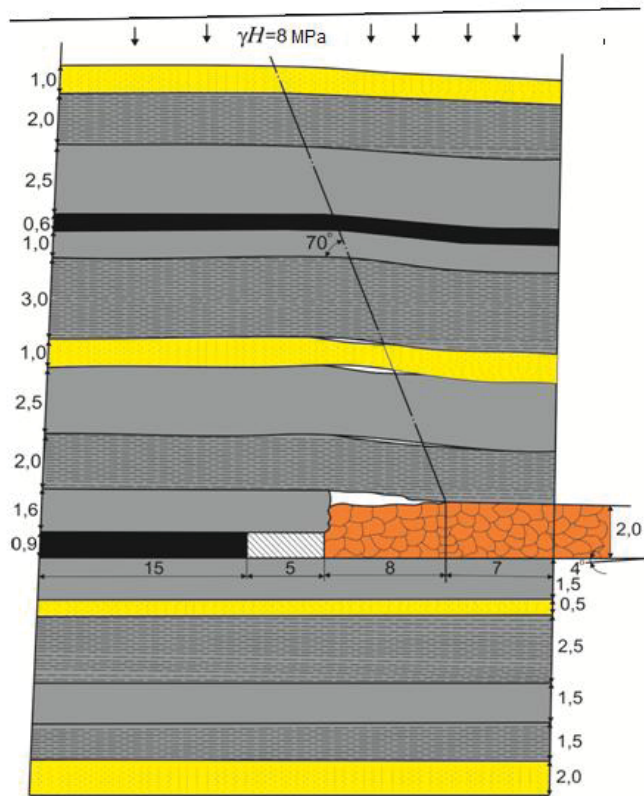


Fig. 8.1. Vertical section (plane XY) of the scheme of the stress-strain state calculation of the coal-containing massif during the stopping of 161st longwall of mine "Samarskaya"

its thickness the process of displacement is localized in the roof before the seam C_6^1 , and above it there is a smooth lowering of layers without discontinuity. However, in order to improve the reliability of results of the stress-strain state calculation was decided to include in the model of the roof three layers of rocks (mudstone, siltstone and sandstone) that occur above the seam C_6^1 , as it is seen from the calculation scheme in Figure 8.1. In the bed of the seam C_6 were modeled six lithological differences on the total depth of 9,5 m, which is sufficient to account the perturbation of the rock pressure that is manifested in the vicinity of the stope.

Now let us consider the range of mechanical characteristics of each lithological difference, which is required for the most reliable modeling of geomechanical processes in the vicinity of the stope. It is generally accepted [47, 58-60], that the state of the rock most objectively reflects the complete diagram of its deformation, which includes following stages: elastic-plastic, softening and loosening. Furthermore, all stages are developed in time and must include rheological component of deformation process. Attempt to perform the stress-strain state

calculation of the model within the software package Solid Works was unsuccessful due to technical reasons (limited computing resources) and opportunities of the programs themselves, which have the potential for solving individual simplified tasks with considered limited numbers of parameters.

Prospects for the objective reflection of geomechanical processes are connected with the software package Ansys, the potential of which allows to simulate geomechanical system in the full extent. But there are also technical and technological challenges:

- technical - required considerable computing resource;
- technological - the development of modeling techniques in relation to the characteristics of geomechanical problems for the reflection of the rock mass state considering technological parameters of extraction works and technical characteristics of the applied equipment.

Taking into account mentioned factors was accepted the methodological decision by the stepwise complication of the physical essence of the model (elastic and elastic-plastic performances, complete stress-strain diagram, consideration of the rheological properties) of technological parameters of extraction works and technical characteristics of the applied equipment. Therefore, at the first stage of modeling the stress-strain state calculation of the rock mass in the vicinity of the stope is carried out in the elastic formulation. But considering the prospect of complications of the physical side of the model have been collected, analyzed and systematized comprehensive data on the mechanical properties of all lithological differences, which are typical for coal-containing thickness of Western Donbass rocks. Justification of modeled values of mechanical characteristics was performed according to mining and geological prognosis of 161st longwall of mine “Samarskaya” and studies [55, 56, 61] of mechanical properties of Western Donbass rocks, which are summarized in Table 8.1.

The coal seam C_6 of simple structure within the extraction area has thickness of 0,85-1,03 m with the sufficiently large range of variation of the hardness coefficient ($f = 2,9 \dots 5,1$). During the modeling is accepted the average value of compressive

strength $\sigma_{cs} = 40 \text{ MPa}$ and tensile strength $\sigma_{ts} = 3,0 \text{ MPa}$ according to mining and geological forecast. Deformation modulus $E = 0,3 * 10^4$ is taken from [62] for the working capacity of coal seams of Western Donbass. Nesh character of coal destruction and averagely expressed rheological properties allowed to substantiate averaged values of other characteristics of the full stress-strain diagram of the coal and its rheology.

In the third block of positions are justified following model parameters.

By the practice of extraction operations in the Western Donbass was determined that the immediate roof collapses immediately outside the safe-guard of sections of the powered support. The main roof also collapses lit-by-lit, but with some lag from the powered support. Since the increase in the volume of the collapsed immediate roof is not enough to create the sustained main roof, then on some distance from sections the cavity is formed between the collapsed immediate roof and the downgoing main roof. With the retreat of the longwall, this cavity is reduced in height due to the flexure of main roof layers and with the collapse of lower layers disappears entirely. Rocks of Western Donbass are characterized by increased deformability. The value of their flexure without the collapse can be estimated up to 60% of the extracting seam thickness [91]. Then the length of the cavity (in the coordinate X) will be quite limited and in the developed geomechanical model it is accepted 8 m. By the monitoring of the collapse process of the main roof in Western Donbass was detected that the caving of the main roof in the traditional sense is usually not observed, but there is a smooth flexure of its layers with the lowering on caved rocks without the expression of dynamic phenomena.

Table 8.1

Mechanical characteristics of modeled lithologic differences of coal-contained thickness within the extraction area of 161st longwall of mine “Samarskaya”

Mechanical characteristics		σ_{CS} , MPa	σ_{ts} , MPa	$E \cdot 10^4$, MPa	$\frac{\sigma_{CS}^0}{\sigma_{CS}}$	$\frac{M}{E}$	$\frac{x}{\beta}$
Lithological difference	1. Sandstone	50,0	3,3	3,0	0,1	4,0	0,20
	2. Siltstone	35,0	2,3	1,0	0,15	2,5	0,25
	3. Mudstone	21,5	2,0	0,5	0,20	1,0	0,35
	4. Coal seam C_6^1	35,0	1,0	0,3	0,10	3,0	0,20
	5. Mudstone	23,0	1,5	0,5	0,20	1,0	0,35
	6. Siltstone	29,0	2,3	1,0	0,20	2,0	0,30
	7. Sandstone	50,0	3,3	2,0	0,10	3,0	0,20
	8. Mudstone	25,0	1,5	0,5	0,20	1,0	0,35
	9. Siltstone	33,5	2,8	1,0	0,20	1,5	0,30
	10. Mudstone	26,5	1,5	0,2	0,25	0,5	0,40
	11. Coal seam C_6	40,0	3,0	0,3	0,10	3,0	0,20
	12. Mudstone	24,5	2,5	0,2	0,25	0,5	0,40
	13. Sandstone	50,0	3,0	2,0	0,10	3,0	0,20
	14. Siltstone	30,0	2,5	1,0	0,15	2,0	0,25
	15. Mudstone	21,5	1,5	0,3	0,25	0,8	0,40
	16. Siltstone	31,5	2,5	1,0	0,15	2,5	0,20
	17. Sandstone	55,0	3,5	2,0	0,08	4,0	0,15

Further, in the goaf layers of the main roof lie on caved rocks and deform them to the initial value of the vertical rock pressure $\sigma_y = \gamma H$ at a certain distance from the longwall. In vicinity of the longwall the efforts on the contact of the main roof and caved rocks are determined by the deformation modulus E_{cav} . In order to determine its value were used experimental studies [63, 64] by the loading of the gob protective

pack in mines of Western Donbass. Processing of results of these studies to determine the value E_{cav} has shown the nonlinear increase of E_{cav} due to the compression (which is consistent with the provisions of the mechanics of flowing mediums and rocks). However, the initial part of the contact (after the cavity) is more interesting, where the most active flexure of the main roof layers occurs. Here on the contact length 7 m (total distance from roof support sections 15 m) the calculated modulus of the deformation of caved rocks is $E_{cav} = 3 \text{ MPa}$. In such a way, the total length of the model in the goaf is 15 m, as it is shown in Figure 8.1 (vertical cross-section in the XY plane) and Figure 8.2 (top view of the formation in the XZ plane). The preliminary testing of the model has shown that this distance along the X coordinate is quite enough to describe main processes of the main roof lowering in the goaf. The distribution of efforts at contacts of layers of downgoing main roof is determined in

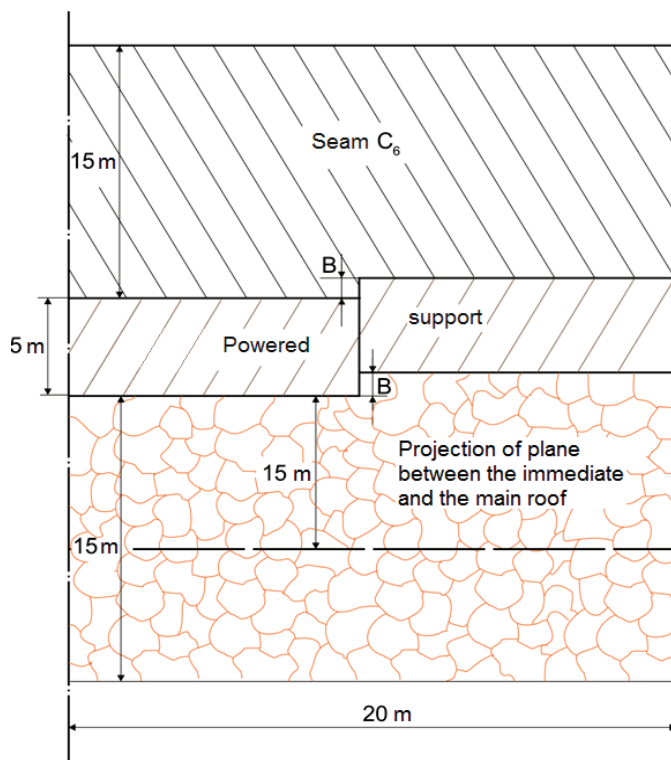


Fig. 8.2. Calculation scheme of stress-strain state of geomechanical systems in the formation plane (the plane XZ)

the calculation process of its stress-strain state depending on the thickness and deformation characteristics of the composing layers.

In front of the stope is researched the distance of 15 m of the virgin coal, which is caused by the fact that at this distance is almost completely located abutment pressure zone in front of the longwall, since the strong coal seam C_6 (relative to roof rocks and bedrocks) concentrates abutment pressure near the wall face.

In modeling of geometrical and power parameters of the powered support following assumptions are accepted. At the first stage of the stress-strain state calculation of the geomechanical system in the elastic formulation to reflect the real operating characteristic of

hydraulic props and sections of powered support generally is not possible, because after the linear (elastic) stage of resistance should be “the sawtooth” mode of yielding property, when periodically are triggered the safety valve and damper of hydraulic-cylinder rods occurs under the influence of rock pressure. However, this mode is quite extreme, when the powered support operates at the limit of its supporting strength with a probability of subsequent landing on the “rigid base”. Our task is to justify parameters of mining operations which do not allow this situation. Therefore, the work is accepted at the site of the linear resistance P of hydraulic props, when the lowering the roof corresponds to a linear increase in resistance P . Therefore, it is necessary to determine the coefficient of proportionality between these parameters, which is similar to the stress-strain modulus (Young modulus) E_{rs} of the roof support.

In order to simplify the process of the stress-strain state calculation of geomechanical system “massif-roof support” is accepted the uniform reaction of the roof support on immediate roof rocks and bedrocks that is achieved by replacing the real construction of the section to the parallelepiped with the stress-strain modulus E_{rs} . The value E_{rs} in the rigid mode is calculated from following considerations. At the linear connection fo P of the reaction of the hydraulic prop and its yield ΔU the relation is correct:

$$E_{rs} = \frac{P_{max}}{\varepsilon_{rs}}, \quad (8.1)$$

where P_{max} - supporting strength of the section (533...629 kPa);

$\varepsilon_{rs} = \frac{\Delta U}{m}$ - relative yielding of the hydraulic prop;

m - extraction height of the seam (0,9...1,0 m).

Concerning to the yield ΔU of the hydraulic prop in rigid mode of its operation it can be noted that it is caused by some compression of the power fluid in the head end of the hydraulic prop and backlashes in bracing joints of the hydraulic prop. In a first approximation, it can be assessed in $\Delta U = 10 \text{ mm}$; then $E_{rs} = 48 \dots 63 \text{ MPa}$. For the

calculation of the average value is first assumed $E_{rs} = 55 \text{ MPa}$, although subsequently there is no difficulty to vary the parameter E_{rs} over a wide range until the yielding mode ($E_{rs} = 2 \dots 6 \text{ MPa}$).

The last block of positions in the justification of the computer model regards to technological parameters of stoping works: the length of the longwall Z , its average daily advance rates V_a , the feed rate V_k of winning machine and the web width B of the actuator.

Preliminary test computations of the stress-strain state have shown that at least in the central part of the longwall (where the maximum height of caving zones and joint-block displacements are formed; the largest of the roof lowering is developed) the influence of its length is not observed. Similar conclusions are made during the monitoring of the plough operation and analysis of pressure readings in head ends of hydraulic props of sections along the length Z . Therefore at the first stage of the stress-strain state calculation of the geomechanical system is excluded from variable parameters the length of the longwall, and in order to save computational resources is considered the central part with the width of 20 m (see Fig. 8.2).

Average daily advance rates V_a of the stope and the feed rate V_k of winning machine are connected with the modeling of time parameters of extraction works (the rheological problem), which are slated to carry out stepwise after the conditionally instantaneously elastic formulation of the problem that is studied in the first stage in accordance with the developed methodology of the sequential complication of geomechanical model. Therefore, in this block of the stress-strain state calculation parameters V_a and V_k are assumed constant, and their influence is considered by the results of experimental studies of the load development on the powered support.

Web width B of the actuator is determined by the coal winning machine design and varies from the minimum of 50 mm for the plough (the lower limit) up to 700-800 mm for a drum head type miner. Preliminary computation tests have shown that a disturbance wave of the rock pressure in the operating area of the actuator at $B = 50 \text{ mm}$ is insignificant and has very limited sizes. Therefore, for the detection of a significant change in the stress-strain state in the operating area of the actuator its

web width is modeled on the first stage with the value of $B = 700 \text{ mm}$. If for this value will become apparent insignificant change of the stress-strain state, then at $B = 50 \text{ mm}$ is especially possible to exclude the influence of the web width of the actuator in the operating area of the miner in comparison with the rest of the longwall length. Otherwise, it is necessary to identify a range of significant influence of the parameter B .

As a result, the computer model for the stress-strain state calculation of the geomechanical system in the longwall area was developed and justified, which is the first stage for the numerical identification of regularities of the displacement process connection of the coal-containing rock mass and technological parameters of the stoping.

8.2. Calculation and analysis of the stress-strain state of rocks in the vicinity of the stope

According to the developed geomechanical behavior model of the coal-containing rock mass in the vicinity of the working face was carried out the calculation of its stress-strain state and developed the system of visual stress-strain state representation of main model elements. The complexity and objectivity of this representation was achieved by the combination of two directions associated with the spatial formulation of the problem.

Firstly, five components of the stress-strain state are subject to the subsequent analysis:

- 1) vertical stresses σ_y ;
- 2) horizontal stresses σ_x , which are perpendicular to the wall face and due to the small seam inclination ($\alpha = 4^\circ$, see Fig. 8.1) are nearly parallel to the plane of the formation;
- 3) horizontal stresses σ_z , which are parallel to the wall face;
- 4) intensity or reduced stresses σ , expressing the combined result of the action of above components, by which is estimated the limiting state of the massif in the compression area;

5) full movements U of the massif, characterizing both the magnitude and the direction of rocks displacement.

Secondly, the spatial model causes necessity to reflect component distribution diagrams in different cross sections of the geomechanical system. The most informative for estimating the system state will be following cross sections (under its minimum number):

- view №1 - isometry that passes along the coal seam at the distance 0,1 m from the face for the group of pushed to the wall face sections (see Fig. 8.2), then the cross section perpendicular to the face (starting from the coal bench) with the cut of the nearest unremoved section and the mined-out space behind the longwall, and then along the mined-out space parallel to the longwall (example of such cross section is presented for the diagram σ_y in Fig. 8.3, a);

- view №2 - isometry that passes in the goaf near (distance of 0,1 m) from the safe-guard of pushed to the wall face sections, then the cross section passes perpendicular to the face along the nearest unremoved section and the mined-out space and then along the mined-out space parallel to the longwall face (example of such cross section is presented for the diagram σ_y in Fig. 8.3, b);

- view №3 - flat section XY perpendicular to the wall face that passes along the outer unremoved section (example for the diagram σ_y is presented in Fig. 8.4, a);

- view №4 - flat section YZ along the coal seam at the distance 0,1 m from the wall face (example for the diagram σ_y is presented in Fig. 8.4, b).

These cross sections complement each other and provide the most complete picture of distribution fields of each component of the stress-strain state, the analysis of which is made by diagrams of vertical stress isolines σ_y that are shown in Figure 8.3 and Figure 8.4.

On the site of pushed to the wall face sections (distance of 0,1 m) operates the bearing pressure (see Fig. 8.3, a and Fig. 8.4, b) of very significant quantity:

- vertical stresses concentration in relation to the initial state ($\sigma_y = \gamma H$) is distributed along the entire length of longwall and its minimum value is $\frac{\sigma_y}{\gamma H} = 2,5 \dots 3,4$;

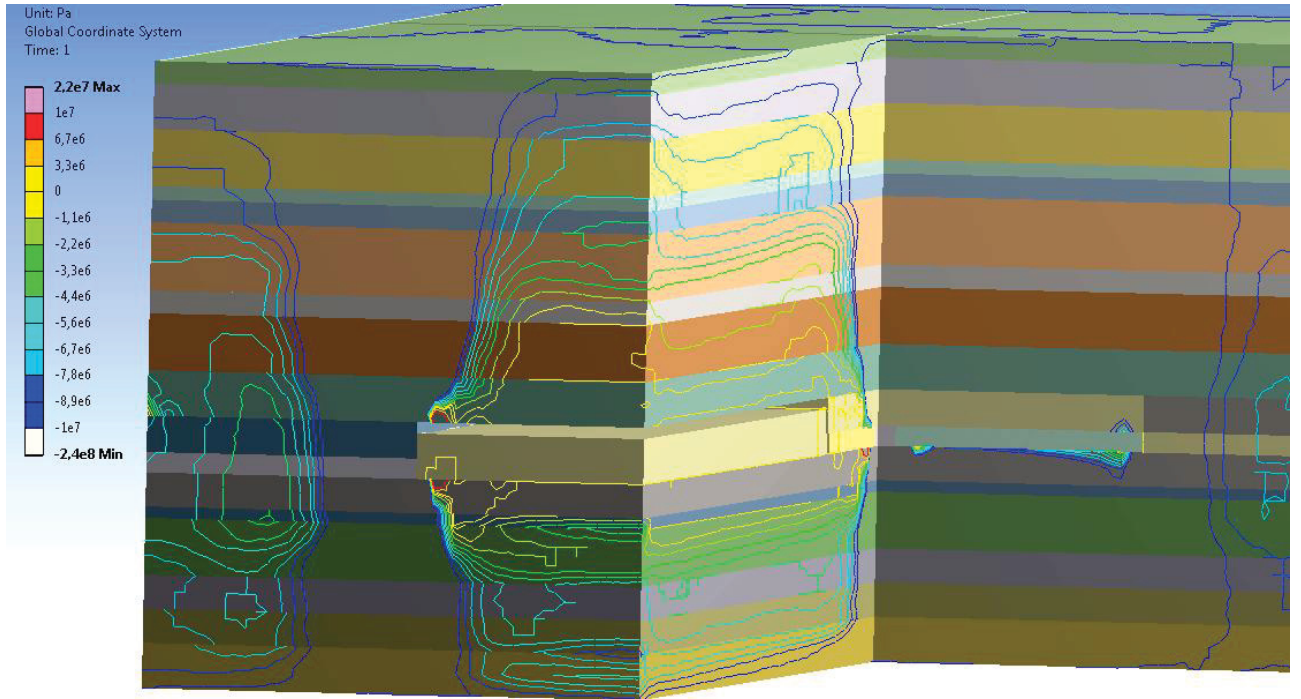
- during the approach to the face bench, where is located the actuating element of the miner, the concentration σ_y increases up to $(5,0 \dots 6,2)\gamma H$ at the distance up to 8,3 m from the coal bench, and at the distance up to 2,5 m the concentration reaches up to $\frac{\sigma_y}{\gamma H} = 8,4 \dots 10$.

In such a way, a significant concentration of vertical stresses σ_y is revealed in the work area of the actuating element of the miner with the web width of 0,7 m, which softens the coal seam. This facilitates the separation of the coal from the massif, but increases the rock pressure in the operation area of the miner.

In front of the miner is considered the cross section in the coal seam distant from the wall face on 0,8 m. Here, the concentration of bearing pressure is reduced: the value $\frac{\sigma_y}{\gamma H} = 3,4 \dots 3,8$ is spread along the longwall up to 7,5 m from the coal bench and on height up to 4,2 m, involving two layers of the main roof.

In general, is revealed a significant bearing pressure in the face vicinity, which in the work area of the actuating element of the miner is increased in several times in relation to the average pressure along the length of the longwall. Consequently, with the actuating element web width of 0,7 m is observed the intense wave of stresses σ_y that must be considered during the operation of the longwall set of equipment. A significant bearing pressure is explained by the increased strength of the coal seam relative to immediate roof rocks and bedrocks, which differs up to 2 times, and with the watering of the mudstone and siltstone differs more intense.

a)



b)

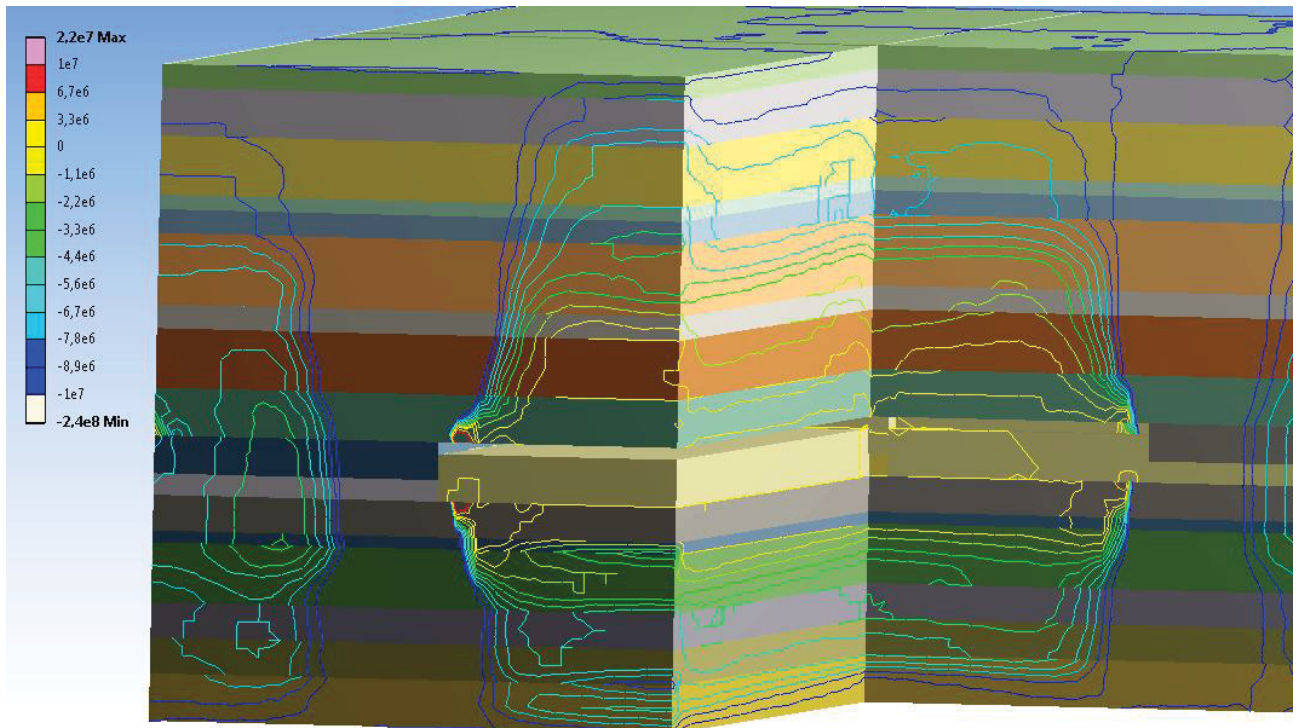
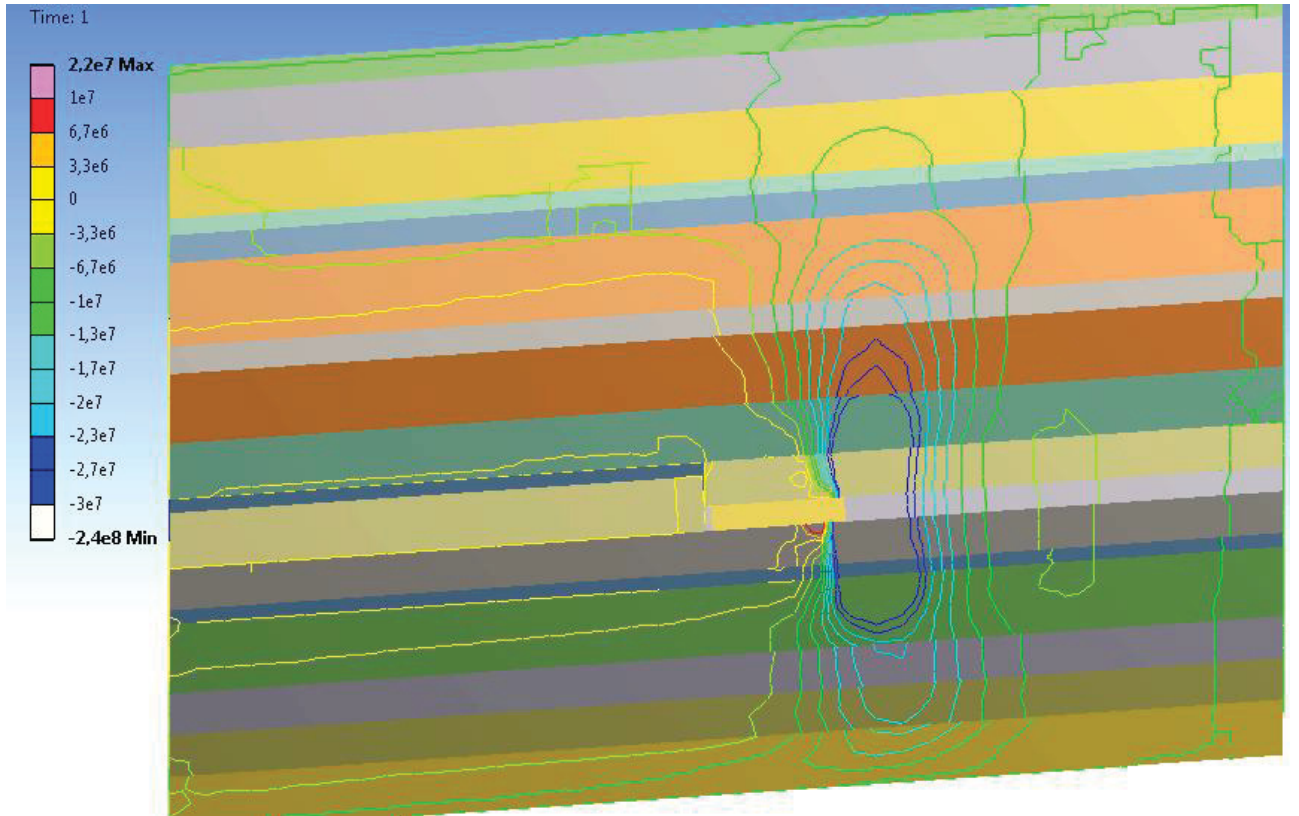


Fig. 8.3. Isolines of vertical stresses σ_y for sections: a) view №1; b) view №2

a)



b)

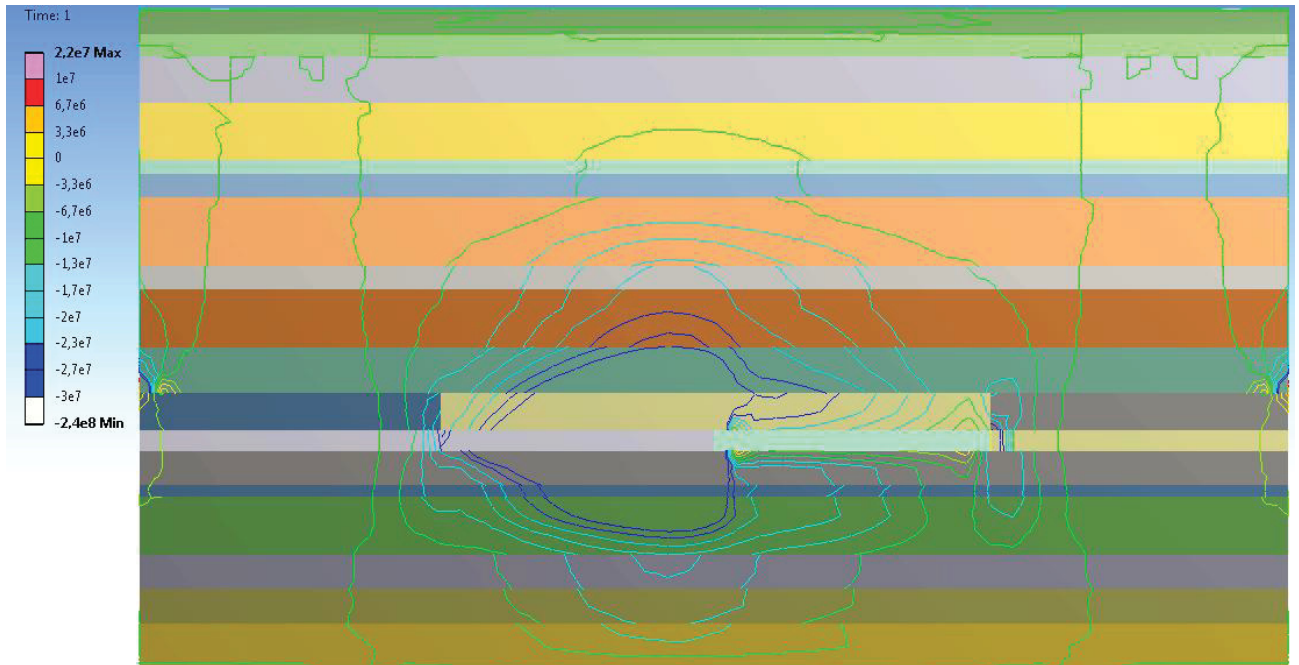
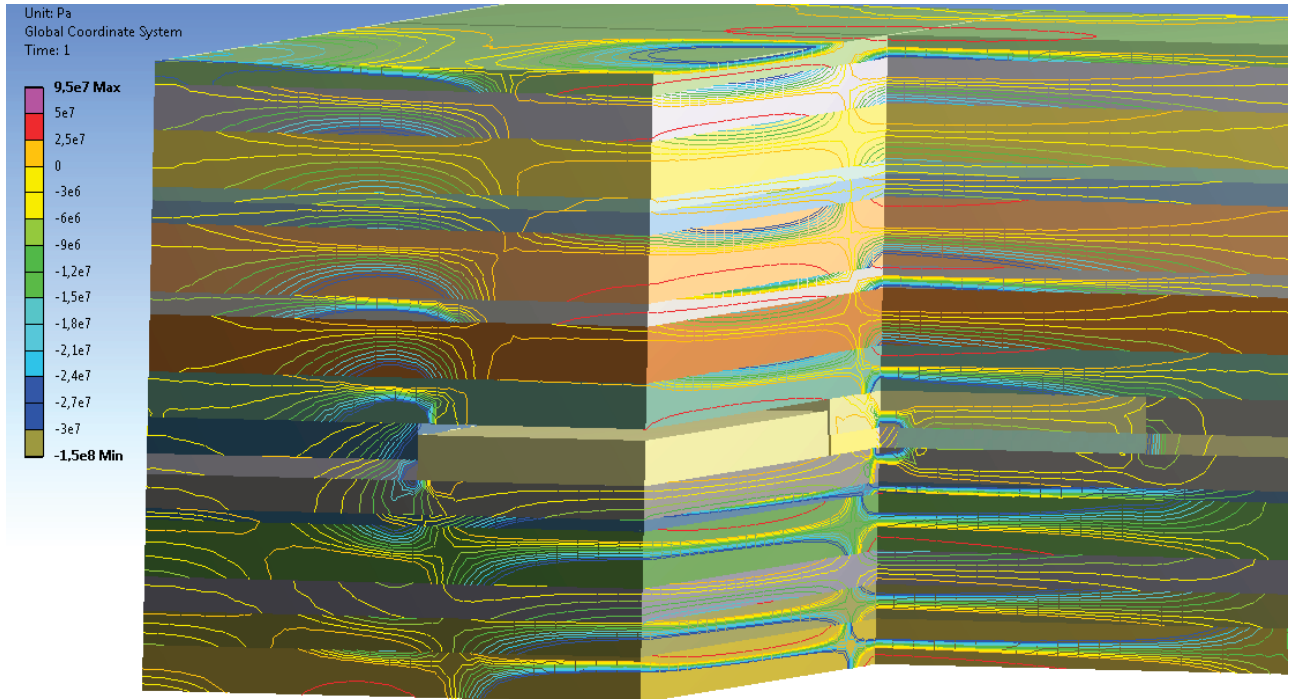


Fig. 8.4. Isolines of vertical stresses σ_y for sections: a) view №3; b) view №4

The stress-strain state of the roof above the powered roof support (Fig. 8.3 and Fig. 8.4, a) and in the goaf is characterized by the following features. Above the edge of the roof support console at the distance of 0,5 m from the wall face occurs the intense drop of the vertical stresses from $\frac{\sigma_y}{\gamma H} = 2,5 \dots 3,4$ to $\frac{\sigma_y}{\gamma H} = 0,4 \dots 1,0$; on the rest of the canopy length $\frac{\sigma_y}{\gamma H} = 0 \dots 0,1$. In the goaf σ_y are falling down to zero along the length of the cavity between the caved rocks of the immediate roof and hovering rocks of the main roof. Then during the formation of contact between them, σ_y is increased up to $(0,1 - 0,4)\gamma H$, i.e., roof rocks in the goaf are unloaded on the modeled area. For the cross section with sections before the coal bench (Fig. 8.3, b), the powered support is located in a more unloaded state without the formation of concentrations of σ_y at the edge of the console, and in the mined-out space the diagram is almost identical to the previous one. The above data of the stress-strain state calculation generally are comply with the developed mechanism of the roof caving and its interaction with the powered support.

Further were considered horizontal stresses σ_x , perpendicular to the working face, isolines of which by views №1 and №2 are shown in Figure 8.5 and by views №3 and №4 - in Figure 8.6. The greatest gradient of σ_x changes by the thickness of each layer (from tensile stresses in the upper part to compressive stresses at the lower part) is observed in the coal bench area, which is manifested even at the upper limit of the model (the height from the top of the layer is 17,2 m) and is distributed up to 7,7 m in the direction of the excavated strip of the coal and up to 6,1 m over the coal bench. With the approaching to the seam C_6 the deflection of layers is increased and the distance of the beginning of essential deflection in front of the longwall is also increased and varied from 3,0-3,5 m in the lower layers, up to 1,5-2,5 m in the upper layers of the main roof. The revealed feature confirms existing concepts of the above-coal strata displacement and the developed mechanism of the behavior process in conditions of Western Donbass.

a)



a)

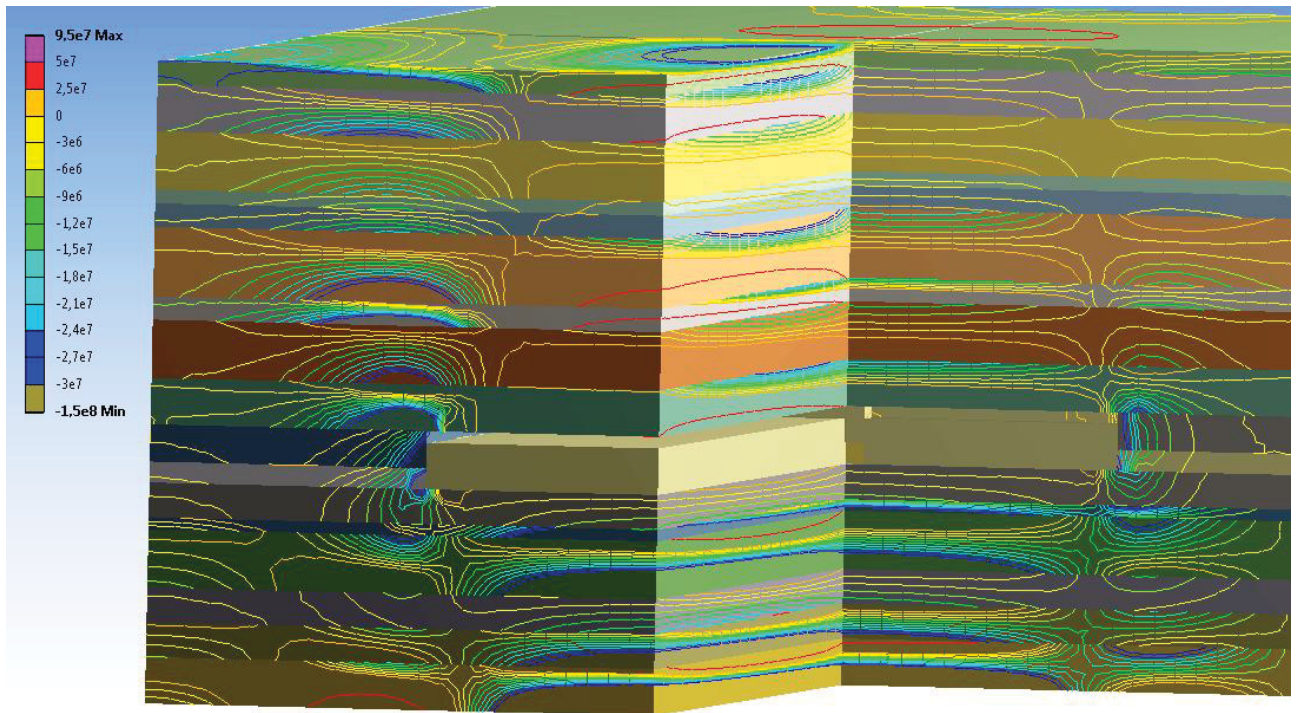
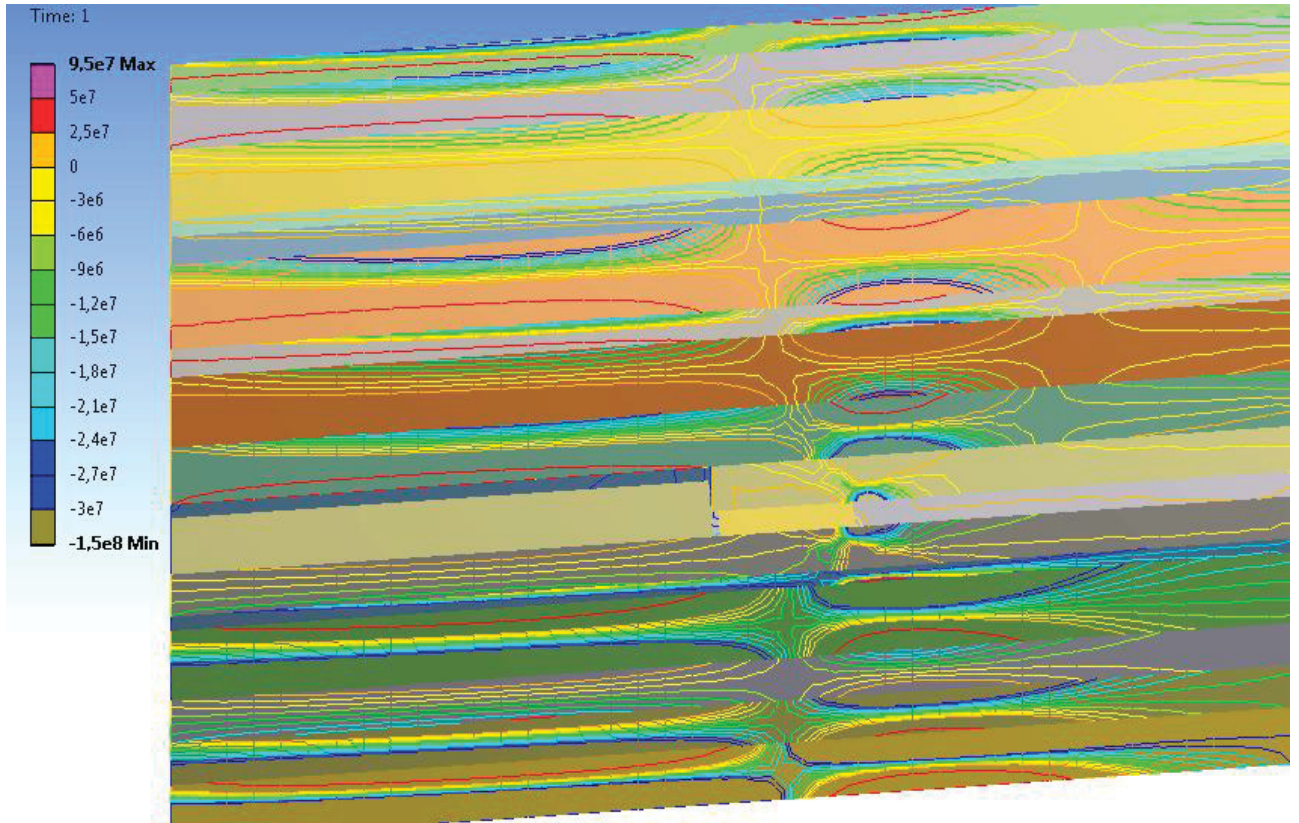


Fig. 8.5. Isolines of horizontal stresses σ_x for sections: a) view №1; b) view №2

a)



b)



Fig. 8.6. Isolines of horizontal stresses σ_x for sections: a) view №3; b) view №4

Above the powered support, in spite of its resistance, there is the change of the sign of the curvature of main roof rock layers. The geometric locus of the points of inflection of layers lies on the line, starting from the wall face (with the displacement of 0,2-0,3 m in the direction of the goaf) and spreading to the main roof at an angle of 75-80°, which is slightly higher than values of this angle in the existing studies. Some discrepancy of obtained values of the internal angle of complete displacements is explained by the increased deformability of rocks in Western Donbass, which are incurvated and caved without the occurrence of extended consoles. This is confirmed by a number of conditions of the developed mechanism of the process of above-coal strata displacement and corresponds to the mine observations about weak rock pressure manifestations during the main roof caving.

Over the powered support is started an intense deflection of layers with the appearance of high tensile stresses σ_x in the lower part of each layer and compression stresses σ_x in the upper part. At the same time, they sink lit-by-lit on gob rocks of the immediate roof, and the intensity of their deflection has the general trend to lower with the distancing from the seam. Here it should be taken into account manifested anomalies of the flexure and isolines σ_x . Layers with the reduced deformation modulus are caved more intensive, but, due to the support of underlying layers of increased stiffness, stresses σ_x in them is much smaller (both tensile and compressive). The same happens if at the top of the layer occurs tougher rock layer. In the tougher layers, vice versa, the lowering is less, and the value σ_x is higher than in more yielding layers, which is fully matched to a number of classic statements of rock mechanics.

Along the longwall more substantial deflection of layers takes place in the area of coal bench, which is defined by the position of the miner. Typically, this deflection is localized on the length of longwall up to 10 m in one and the other directions from the coal bench. By the X coordinate the deflection is manifested in a few meters behind the safe-guard of support sections. Nevertheless, the stresses wave σ_x (as well as σ_y) occurs in the area of the miner operation with the web width of 700 mm. This

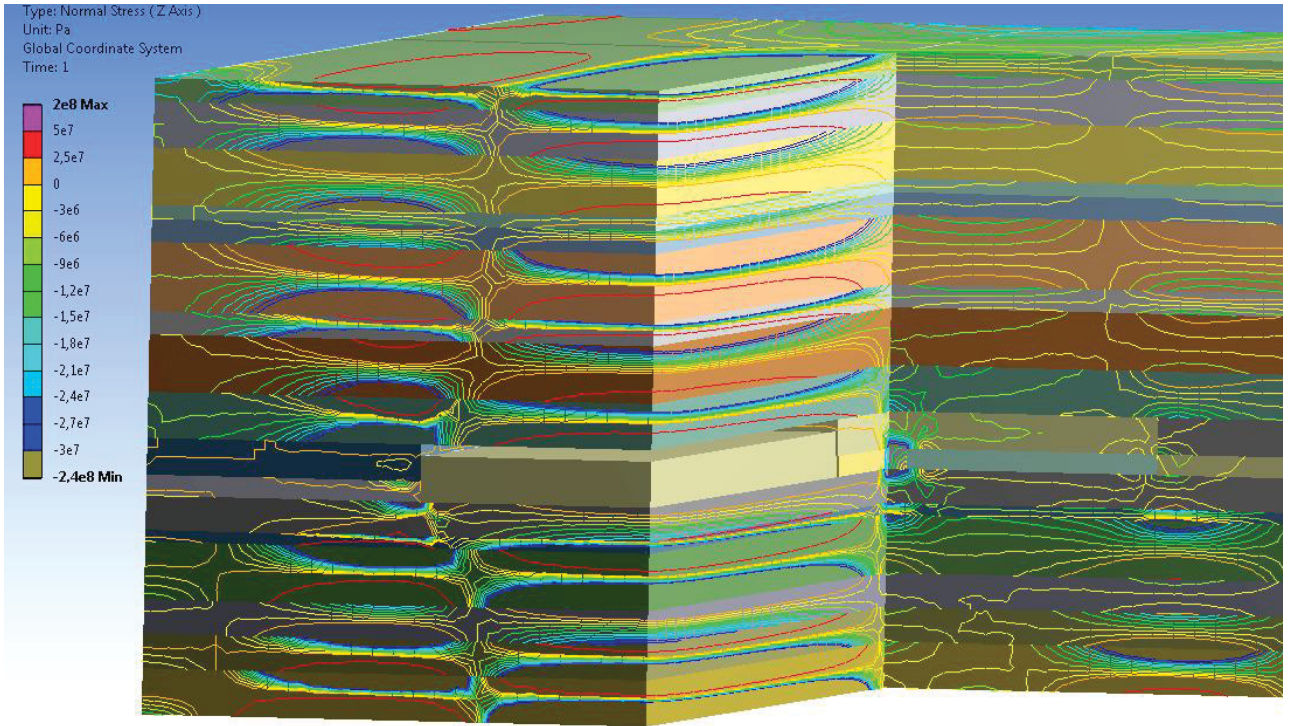
should be considered in assessing the stability of the face and the load development on the support.

The next component for the analysis of the stress-strain state is the horizontal stresses σ_z acting parallel to the working face (Fig. 8.7 for views №1 and №2 and Fig. 8.8 for cross sections №3 and №4). Isolines of the component σ_z the most complete characterize the flexure of rock layers parallel to the wall face, i.e., in the plane YZ .

Almost at the edge of the coal seam (0,1 m from the wall face) along the length of the already excavated strip of the coal up to the bench, the flexure of roof layers in the plane YZ takes place in the direction of the seam. The value of the deflection and the gradient of σ_z changes of the thickness of each layer is not so strongly expressed as for the component σ_x (see Fig. 8.7, a): tensile stresses σ_z are appeared only in the thick (3,0 m) and quite stiff ($E = 1 * 10^4 \text{ MPa}$) siltstone; in the upper part of each layer compressive stresses σ_z only in 1,2-1,6 times higher than the original state of the virgin massif.

In the plane XY , perpendicular to the wall face, the significant flexure of roof layers and the gradient of σ_z changes within the thickness of each layer are observed (in the upper part it is tension, in the lower part - compression). This is explainable by the deflection of layers and by the subsequent caving of the roof in the mined-out space. Since the flexure of layers takes place not only in the plane XY , where the determining component is σ_x , but also in the plane YZ (along the longwall), which is confirmed by lines of the component σ_z . In such a way, due to the flexure of rock layers in the plane YZ , they are divided by tensile stresses σ_z into rock blocks, which corresponds to the established concepts of the presence of the joint-block displacement zone that is enclosed between the zone of the disordered caving of the roof and the zone of the smooth deflection of layers without discontinuity. The latest is located beyond the height of the model, since even the most distant from the seam layer (17,2 m) experiences significant tensile stresses σ_z (up to 10-12 MPa) the upper part of its thickness.

a)



b)

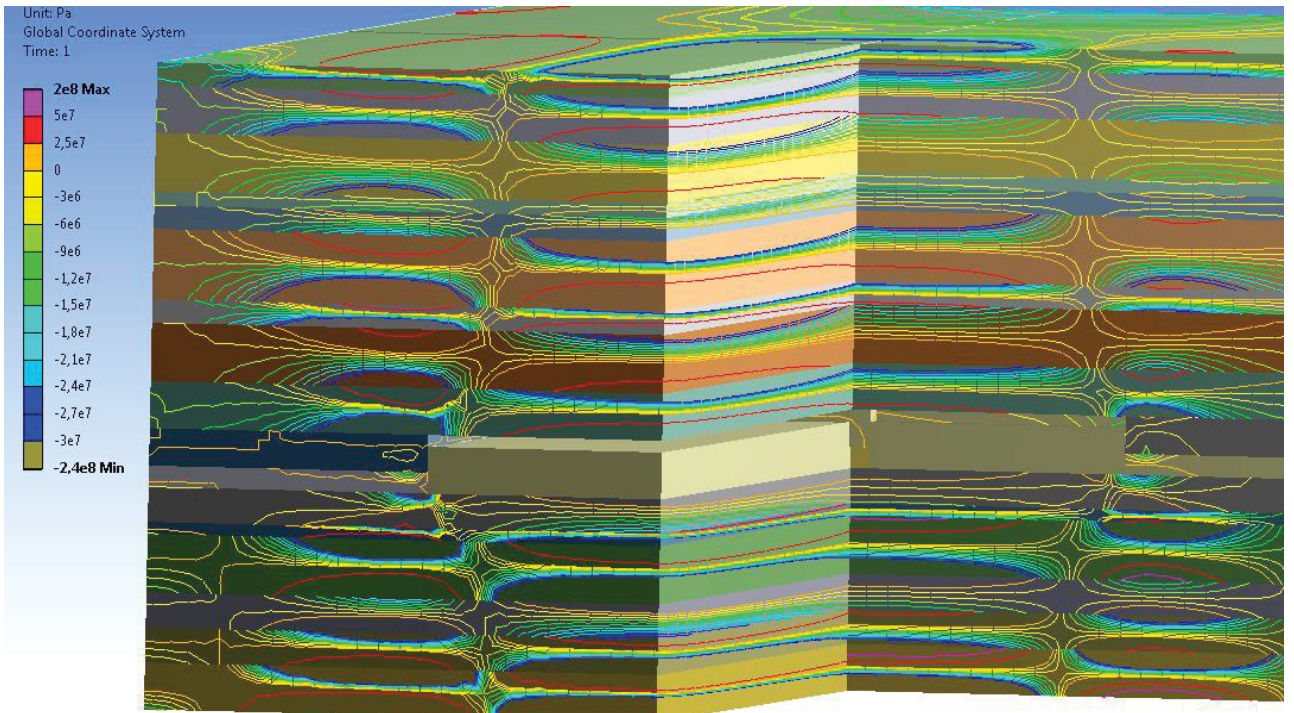
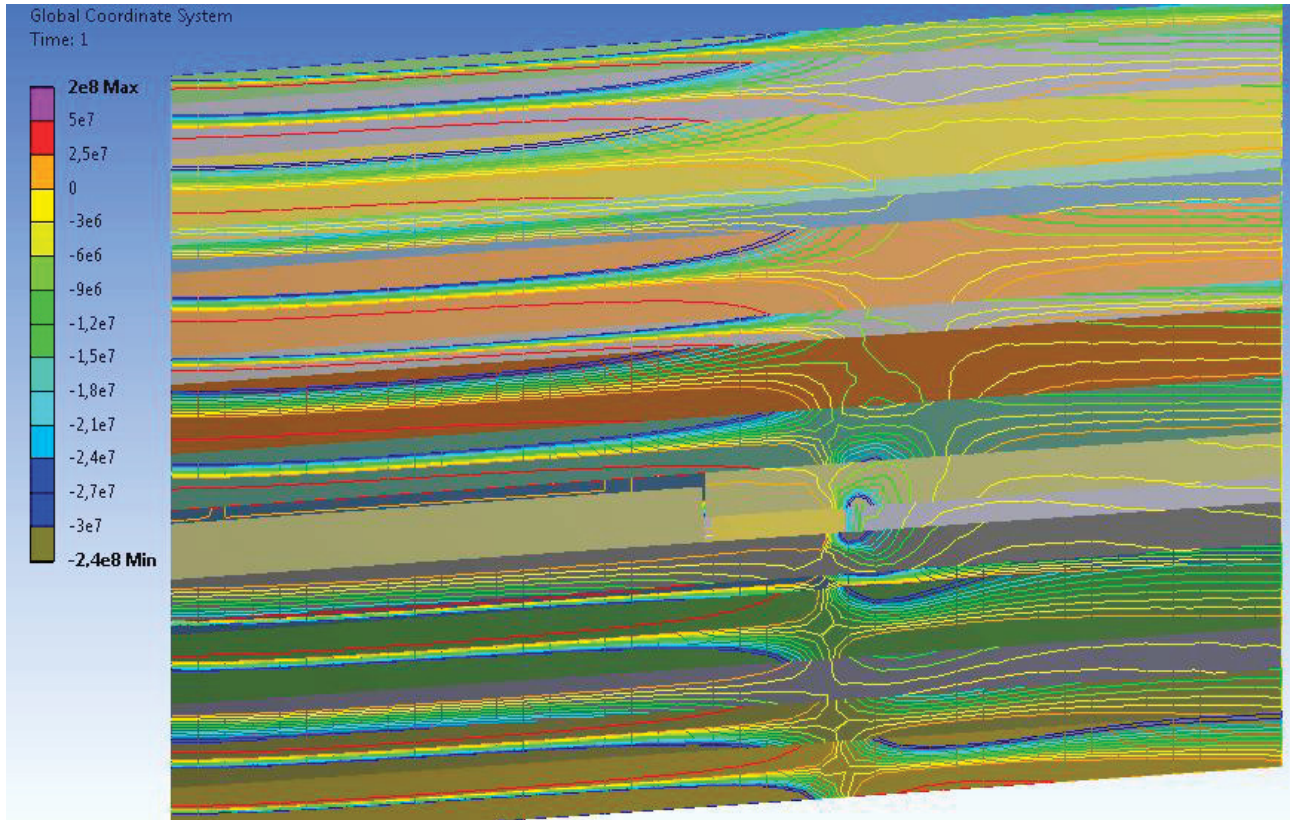


Fig. 8.7. Isolines of horizontal stresses σ_z for sections: a) view №1; b) view №2

a)



b)



Fig. 8.8. Isolines of horizontal stresses σ_z for sections: a) view №3; b) view №4

For the view №2 (Fig. 8.7, b), where all cross sections are located in the goaf, the picture of isolines σ_z similar to the like for the previous view №1, where a part of the cross sections is also located in the goaf.

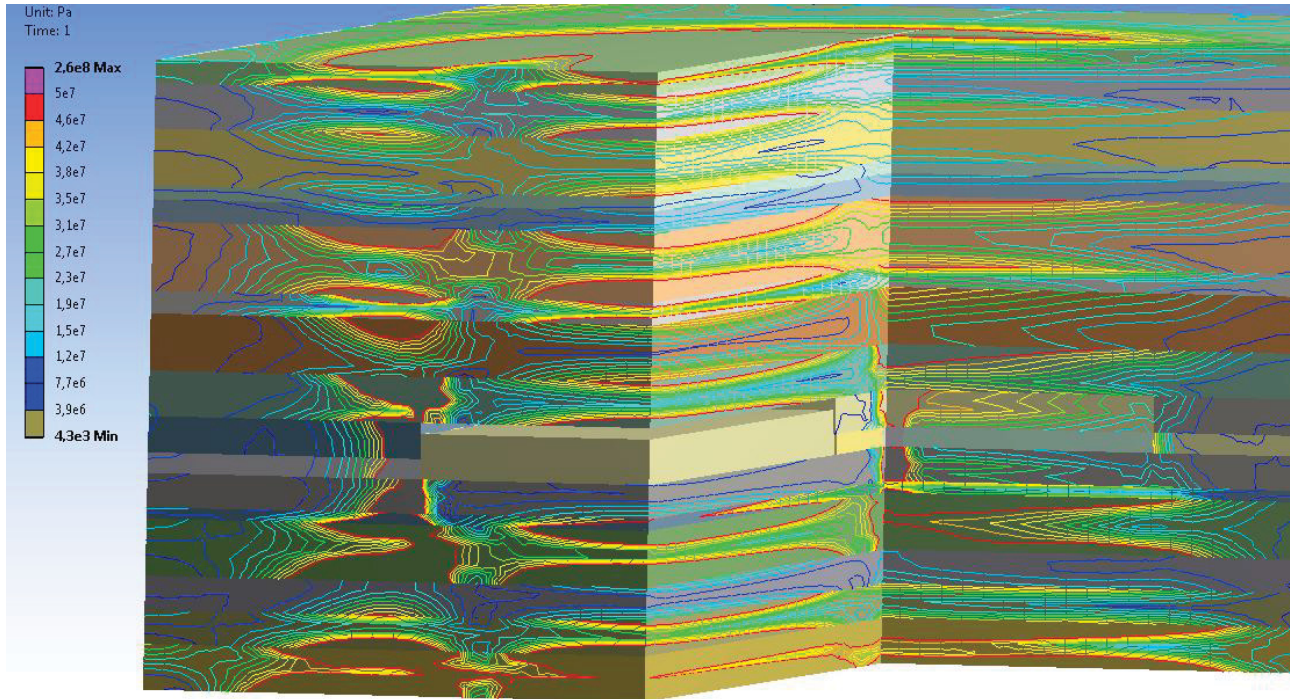
Cross sections in planes YZ (Fig. 8.8 a) and XY (Fig. 8.8, b) confirm previously identified features. Here it should be noted that with the advancement deep into the virgin seam the flexure is essentially manifested only for lower layers of the main roof, and during the approach to the seam C_6^1 flexural stresses σ_z are uniformly distributed over the thickness of layers, which indicates the attenuation of deformations at a distance up to 10-12 m deep into the virgin seam. Along the longwall from the side of the unmined coal strip (the distance from the wall face is 0,8 m into the massif) flexural stresses σ_z are distributed on a smaller distance than from the side of the excavated coal strip. The significant gradient σ_z along the layer thickness is observed only in the rigid and harder sandstone and in the seam C_6^1 , as in elements, which receive the increased load.

In general by results of the analysis of isolines of the component σ_z the following may be noted:

- Firstly, at a distance of 10-12 m deep into the seam flexural stresses σ_z are damped that indicating the limited size of the zone of active deformations of the roof (identical to the bearing pressure zone), as it should be in the hard coal seam and in soft easy-deformed rocks of the immediate roof and bed;
- Secondly, above sections of the powered support and in the goaf appears the intense gradient of σ_z changes by the thickness of each layer of the roof, which indicates on their active flexure in the plane XY and in the plane YZ along the longwall.

The final component of stresses in the analysis of the stress-strain state of the geomechanical system are reduced stresses σ , isolines of which are shown in Figure 8.9 for views №1 and №2 and in Figure 8.10 for views of №3 and №4. As noted earlier, reduced stresses σ characterize the combined action of all components of stresses and are the criterial parameter for the estimation of stability of the rock volume.

a)



b)

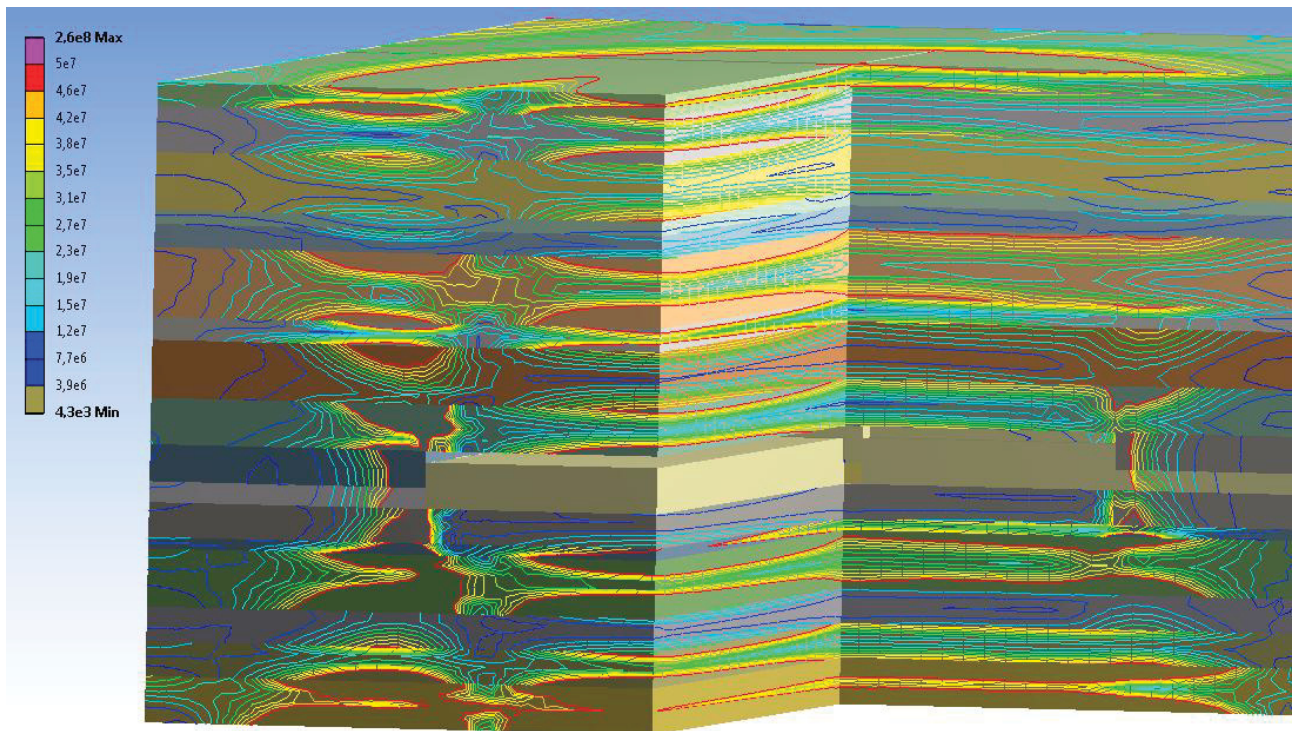
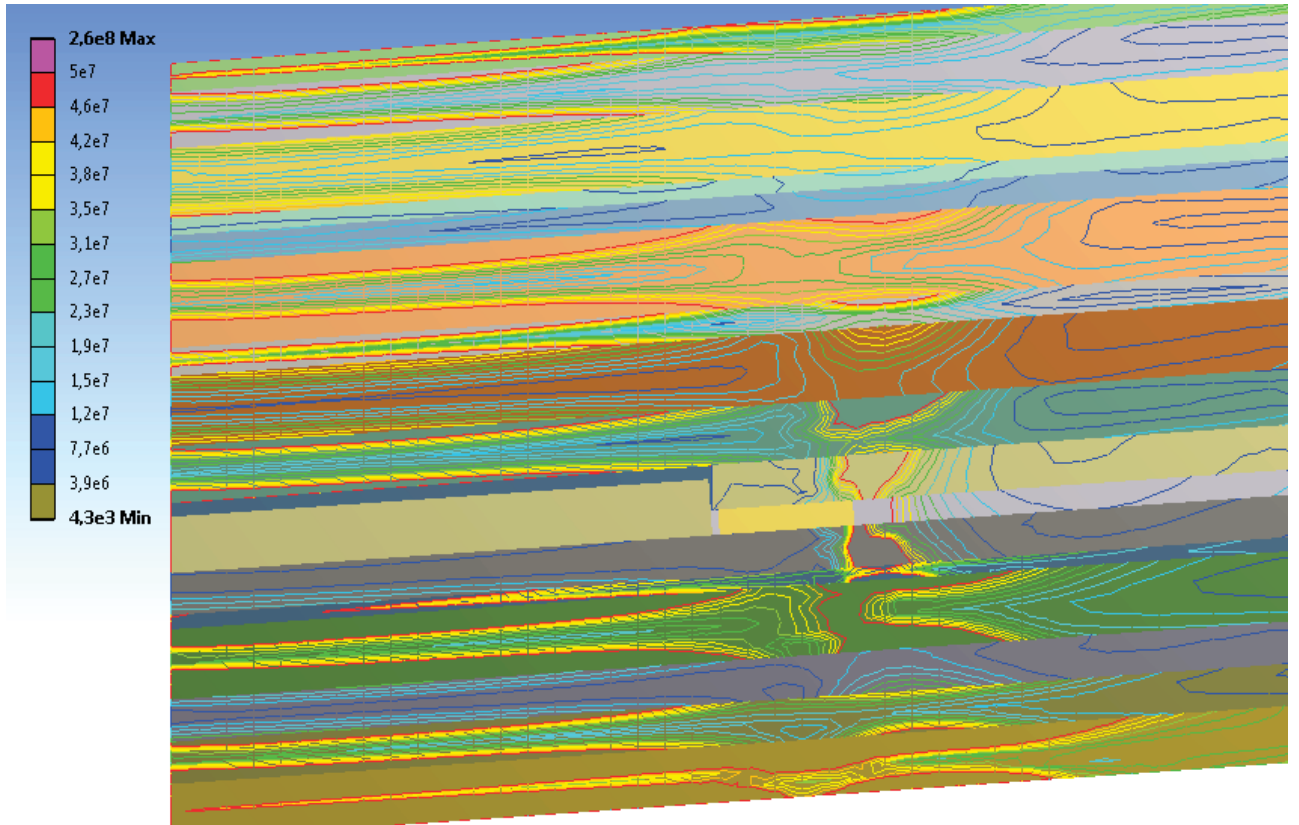


Fig. 8.9. Isolines of reduced stresses σ for sections: a) view №1; b) view №2

a)



b)

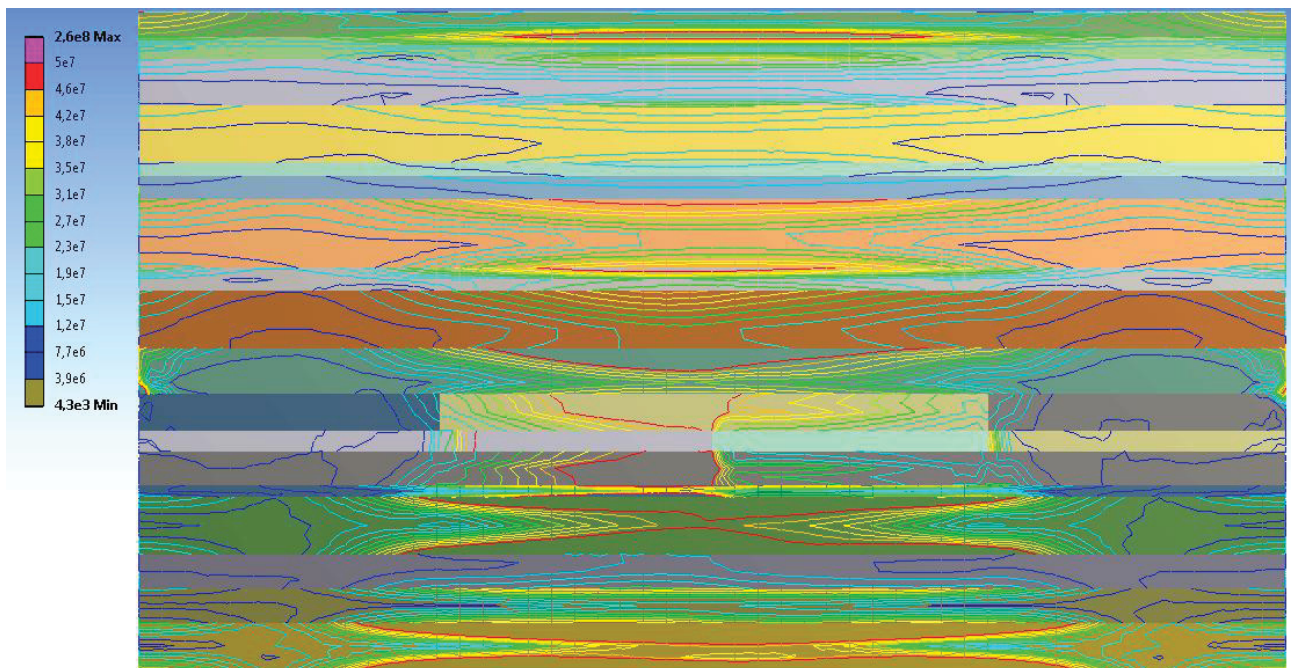


Fig. 8.10. Isolines of reduced stresses σ for sections: a) view No3; b) view No4

In the area of the excavated strip of coal up to the bench (Fig. 8.9, a) the most intense reduced stresses σ are manifested (distance of 0,1 m from the wall face) in more rigid layers of the roof in the upper part of the thickness, where reaches values of $\sigma = 45 \dots 50 \text{ MPa}$. Such value is destructive almost for all lithological differences that is caused by the combined action of compressive stresses σ_y and tensile stresses σ_x and σ_z . This situation is characterized as the most unfavorable combination of stresses in terms of the holistic state of the rock. That is why in the wall face have already been occurring a certain softening of upper parts of most rigid layers, while easily deformable layers are exposed much less stresses σ . Even at the upper boundary of the model the area of softening of the hard sandstone is developed up to 1,2-1,4 m deep into the massif.

The similar situation occurs with the condition of rock layers in the mined-out space (Fig. 8.9, b): in the upper part of each layer the combination of compressive stresses σ_y and tensile stresses σ_x and σ_z leads to the formation of σ up to 45-50 MPa that is usually softens the upper part of layer, and with its inflection in the goaf the lower part of layer is softened too. This leads to the partition of rock layers by cracks into separate blocks and to its lowering on “the bed” from caved rocks, which is consistent with existing concepts and developed regulations of displacement mechanism of the above-coal strata in the goaf.

Areas with the maximum stress σ are located in the wall face area (Fig. 8.10, a) and are distributed in the interior of the seam at a distance of 2,5-3,0 m in more rigid layers; in more yieldable layers such maximum does not occur at all. With further movement deep into the seam the value σ is stabilized at a level, which is lower than the compressive strength of any lithological difference. Along the longwall (Fig. 8.10, b) dangerous concentration of σ is located in the upper part of rigid layer and is distributed to a distance of 5,5-6,5 m from the coal bench.

In such a way, the roof rock softening occurs usually in the upper part of more rigid layers and is localized in the area of the coal bench at a depth of 2,5-3,0 m in the massif.

8.3. Calculation and analysis of the complete displacement of the coal-containing rock mass

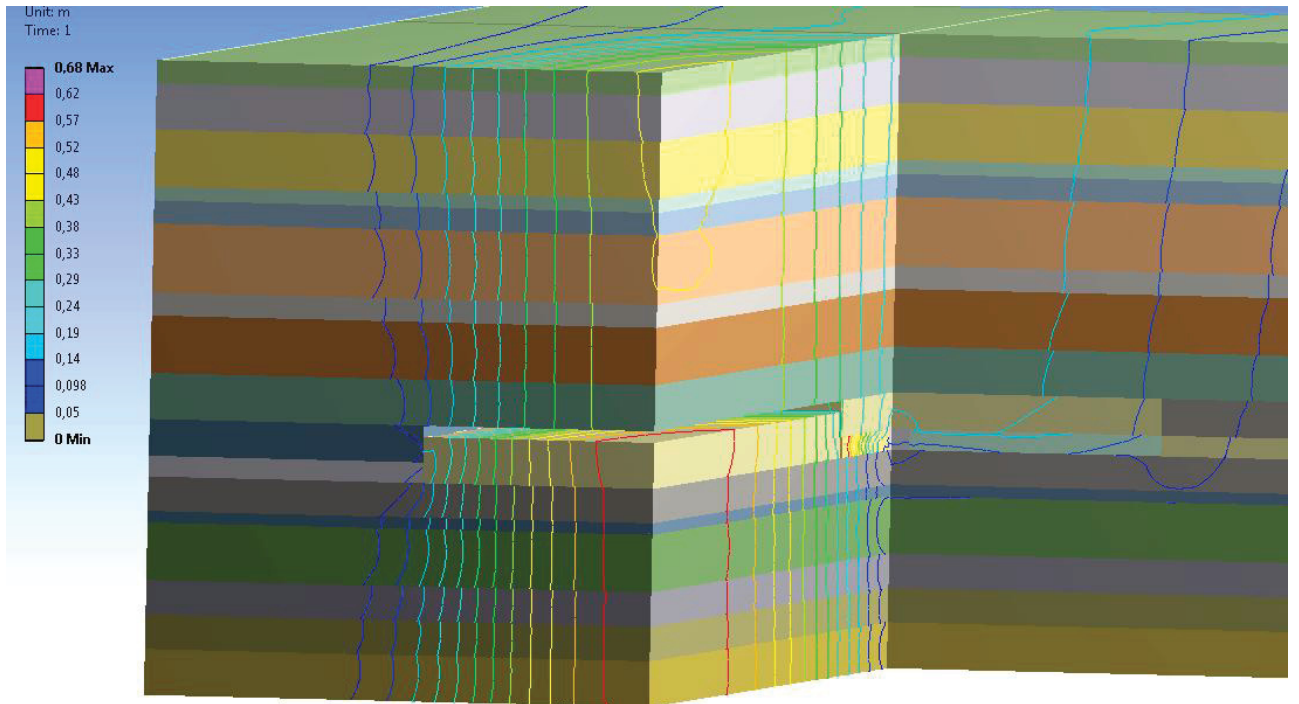
As it follows from the concept - the stress-strain state of rock mass - in addition to stress components must be analyzed deformations of the massif, which in the integral form are implemented in its movements. Diagrams of full displacements U are shown in Figure 8.11 for views №1 and №2 and in Figure 8.12 for views №3 and №4.

In the area of the face (0,1 m deep into the seam, Fig. 8.11, a) from the side of the excavated strip of coal is observed uniform roof lowering of $U = 140 \dots 190 \text{ mm}$ along the length of the longwall. This should be considered when determining the amount of the extracting thickness of the seam in order to remain the reserve of the support extension for the compensation of the further lowering of the roof directly above the support. This value of displacements is reduced up to 50-98 mm only in the area of face ends of the longwall. Over the height of the roof displacements U are relatively constant and most likely will be manifested on the surface, which is observed in Western Donbass.

Above the roof support in the area of the coal bench the roof lowering increases from $U = 98 \dots 140 \text{ mm}$ on the edge of the canopy console up to $U = 330 \dots 380 \text{ mm}$ on the end of the canopy, where is located the safe-guard of sections. In the goaf due to the immediate roof collapse and the appearance of the cavity at the boundary with the main roof occurs the growth of the lowering of the main roof up to 570-620 mm. Further, in the goaf is formed some stabilization of the main roof lowering because of the appearance of its contact with caved rocks. Here, along the Z coordinate of the longwall length (Fig. 8.11, b) is clearly manifested gradient of the roof lowering: on the face ends is a minimum; in the middle of the longwall length in the goaf is a maximum. This fact is consistent with existing concepts about the formation of the dome in the lowering massif over the goaf, which is similar to the dome of natural equilibrium and assumes the movement of rocks in the central part (along the Z coordinate) of the extraction area. Indicated regularities

are confirmed by isolines of full displacements of the roof in Figure 8.12, a in the XY plane, and Figure 8.12, b in the YZ plane.

a)



b)

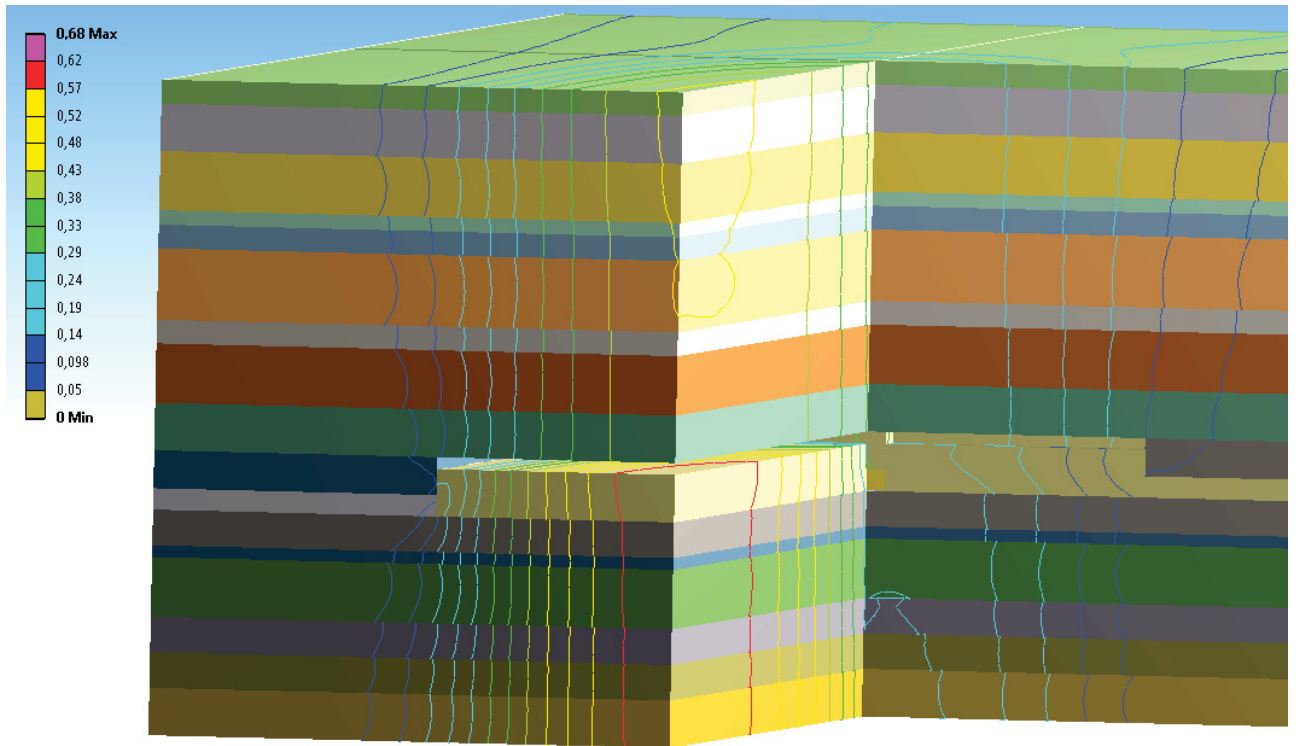
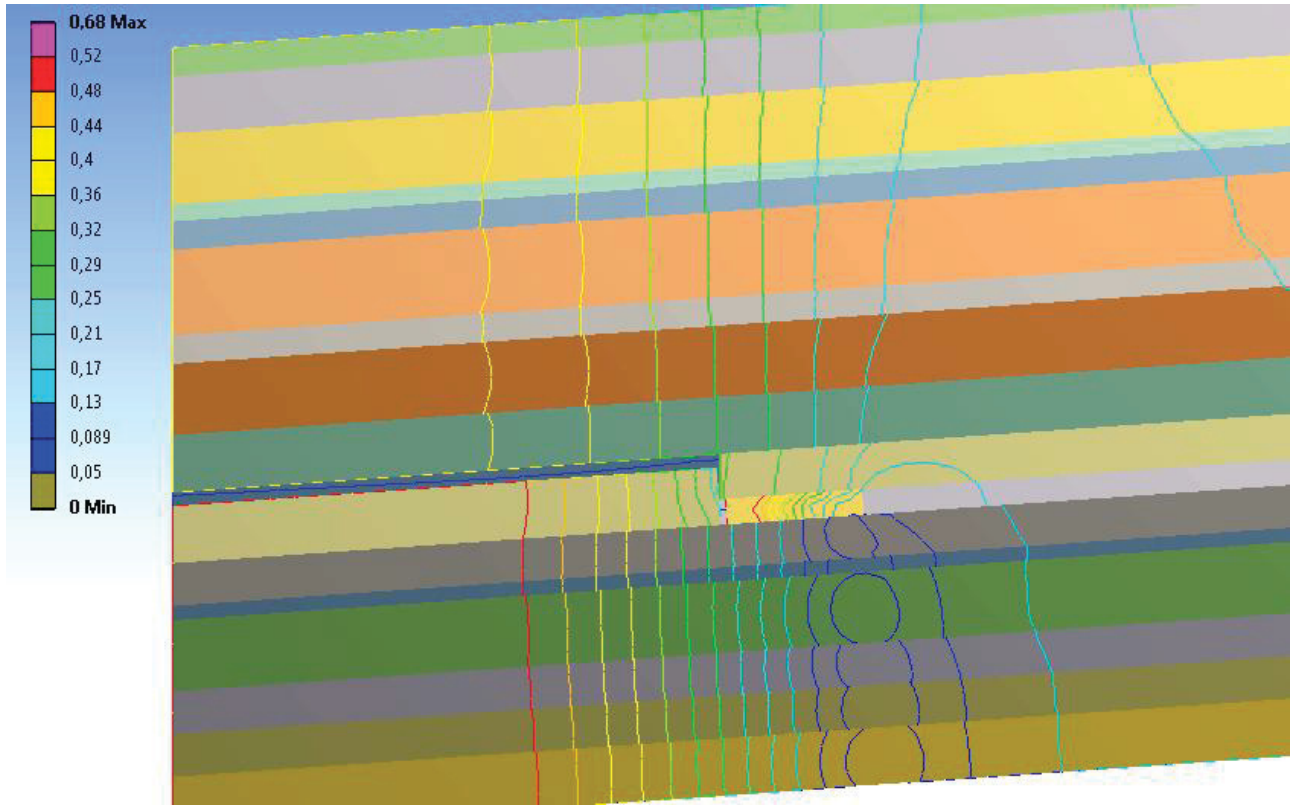


Fig. 8.11. Isolines of full displacements U for sections: a) view №1; b) view №2

a)



b)

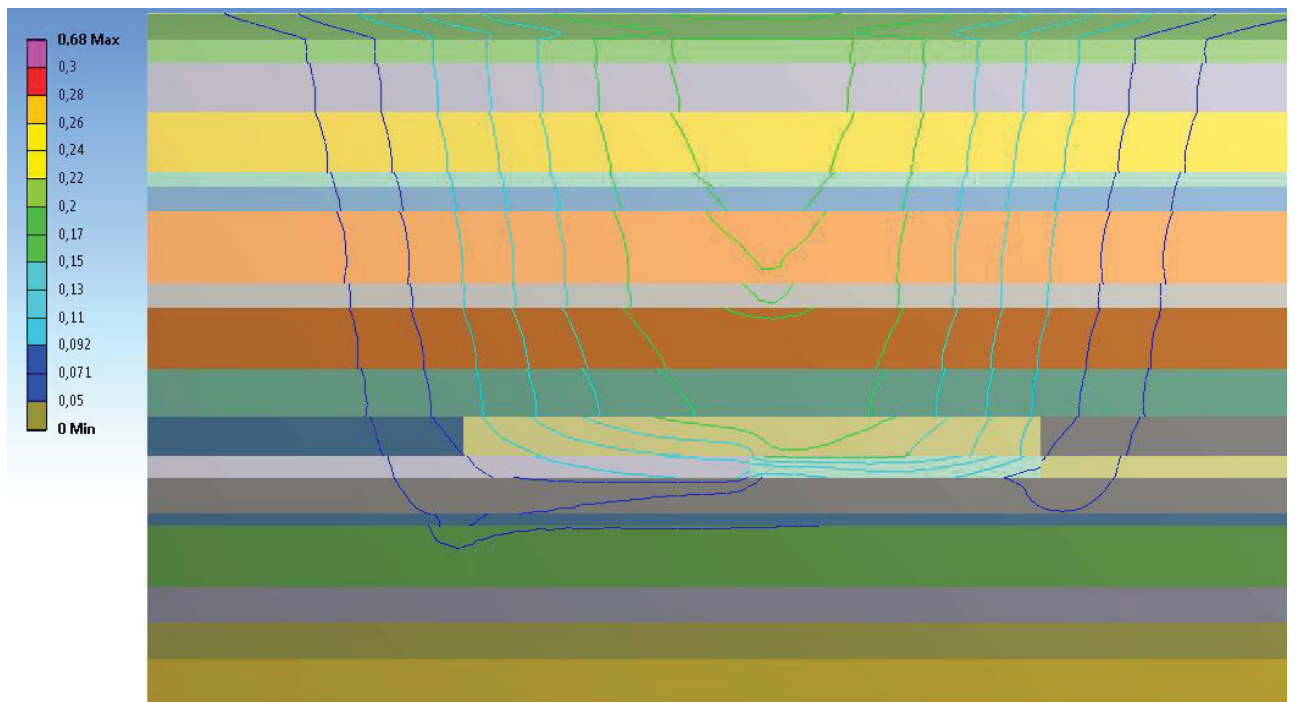


Fig. 8.12. Isolines of full displacements U for sections: a) view №3; b) view №4

In such a way, modeling of the above-coal strata displacement has revealed the following features of this process:

- in the area of the wall face the roof lowering are quite substantial $U = 140 \dots 190 \text{ mm}$ and they should be taken into account in order to ensure the stable operation of the longwall set of equipment and prevent it from landing on the “rigid base”;
- along the length of the longwall (the Z coordinate) in the area of the face the roof lowering are constant with their gradual lowering only on face ends;
- along the length of the section (the X coordinate) the roof lowering is increased up to 330-380 mm in the area of the safe-guard, but this is compensated by the geometry of the section, which has the height at the trailing edge of the canopy by 200 mm less than at the leading edge of the console;
- in the goaf behind the powered support the main roof lowering are increased up to 570-620 mm and stabilized with the appearance of contact between caved rocks and the main roof, which is broken into blocks;
- along the vertical coordinate Y displacements of the main roof are sufficiently constant and with the extrapolation beyond the height of the model is predicted the formation of the subsidence trough of the surface.

In total, modeling results do not contradict to existing concepts about processes of the above-coal strata displacement, mine observations and confirm fundamental principles of the developed mechanism of the roof interaction with the powered support for conditions of Western Donbass mines.

8.4. Conclusions

1) The methodology for monitoring of high loaded longwalls was developed, which has allowed to produce mine studies, considering the separation of tendencies influence of the geological environment, advance rates of the stopping machine and web width of its actuating element.

2) Analysis of results of pressure measuring in hydraulic props of the roof support in the process of assessment of the geological environment influence has established the following:

- ambiguity of the influence of the main roof structure is imposed on ambiguity of the influence of the depth of the longwall placement, which can be assessed as insignificant in terms of the formation of the load on the support (growth of average pressure in hydraulic props up to 4.5-6% with an increase in the depth of development on 22-27%); this is because the load on the support is determined by the volume of rocks inside the contour of the natural self-supporting arch;

- in areas of the extraction panel, where in the middle or in the upper part of the main roof are placed heavy layers (3,5-4,0 m or more), are observed perturbations of the pressure in hydraulic props that is caused by the congestion of rock consoles of a certain length; after their caving the load on the roof support falls below the average value; these areas require special attention in terms of stably high advance rates of the stope and prevention of its prolonged stoppages;

- in areas of the medium- and thin-layered structure of the main roof manifestations of its caving haven't been recorded, and the value of average pressure is slightly higher than for areas with heavy layers except the moment before the main roof caving.

3) At high feed rates and small web width of the plough actuating element stresses wave in the zone of its operation insignificantly affect pressure changes in hydraulic props and increase of the load on the roof support; this fact is stably manifested in a wide range of variation of geomechanical and technological factors; the exception is provided by non-dominant conditions of the caving period of the main roof, which is represented by partly heavy layers; here are recorded overpressure up to 10-12% in the short segment of length up to 12 m (up to 8 sections) and only at unremoved to the wall face sections.

4) For the reasonable and authentic division of the influence of individual technological parameters on the development of the load on the powered support is most advisable to use numerical methods for computer modeling, allowing to vary

these parameters over a wide range of changes. Therefore, was developed and justified the computer model for the calculation of stress-strain state of the geomechanical system in the area of the longwall. As an example, is presented the calculation and analysis of vertical stresses, which showed that near the face is formed a significant bearing pressure, which is increased in the zone of operation of the actuating element of the miner in a few times (in contrast to the plough) relative to the average pressure along the length of the longwall. Consequently, with the web width of 0,7 m is observed intense wave of stresses that must be considered during the work of the longwall set of equipment.

FINAL CONCLUSION

In the process of work monitoring of the plough were analyzed pressure P readings in hydraulic props of support sections of the complex with different combinations of geomechanical and technological factors, as the result of which was established a number of regularities.

1) Sufficiently stable regularities of pressure P changes along the length of the longwall are not revealed. Pressure P fluctuation upward or downward from the average value occurred in different areas of the longwall and most likely caused by the variation of the structure of above-coal strata (along the length of the longwall), the degree of water content, the intensity of fracturing and other geomechanical factors affecting the stability of the roof rocks. Thus, in the studied mining and geological conditions of the work of plough longwall of the mine “Samarska” is not confirmed the classic position about the prevailing load on the support in its central part relative to face ends of the longwall. This is facilitated the curved (convex) shape of the face with its advance in “massif” of the central part relative to face ends of the longwall length, which increases the stability of the rock outcrops in comparison with a rectilinear face shape.

2) The stable regularity of the formation of reduced load on roof support sections, which are pushed to the wall face in comparison with unremoved sections, was identified. Was established that sections located at a distance from the wall face perceive loading in 65-80% from the maximal reaction of resistance and almost universally work in the rigid mode of resistance without switching in to the yielding mode with concomitant decrease in the height of hydraulic props expansion.

3) Was established the tendency of pressure P reduction in hydraulic props of sections with the increase of average daily advance rates V_a of the stope. The regularity of connection P and V_a is such that the main part of the pressure drop occurs in the range of growth of $V_a = 2,5 \dots 8,0 \text{ m/day}$; further the dependence $P(V_a)$ is flatten out and influence of V_a is only 0,9-7,4%.

4) At the coarse- and medium-layered structure of roof rocks during the period unrelated to its caving, the load on sections on average by 10-15% lower than in extraction panel areas with the thin- and medium-layered structure of the roof.

5) In the period preceding the caving of the main roof with the coarse- and medium-layered structure, the load on sections increases up to 20% and on 5-10% exceeds the load in the thin- and medium-layered structure of the roof, which is characterized by the absence or very weak manifestation of the effect of the main roof caving.

6) Was established that in the investigated mining and geological conditions depth change of the longwall placement (development is carried by the long-pillar up-dip of the seam) from $H = 440\text{ m}$ to $H = 340\text{ m}$ unessential affects the rock pressure manifestation.

7) At the stoppage of the extraction face is manifested regularity of the load increment during the down-time t . Was established that, regardless of geomechanical and technological parameters the most intense growth of the pressure P occurs in the first 6 hours of the face stoppage; during this period the pressure increase is 70-91% from the growth value P over 11,5-13,5 hours of the down-time, when the regularity $P(t)$ is determined.

According to the results of monitoring of the operation of the plough longwall is substantiated a number of practical recommendations concerning the reduction of the load on the powered support and the probability of the roof support landing on the “rigid base” for conditions, which are similar to the investigated:

- the most intense reduction of the load on the support takes place during an average daily advance rates of the stope not less than 7,5-8,0 m/day, which is recommended to maintain for the creation of the reserve of resistance reaction of the roof support about 20-30%;
- the increased load on sections is formed before the main roof caving of the coarse- and medium-layered structure, that is way on the marked areas along the length of the extraction panel is recommended to maintain the above-mentioned average daily advance rates of the stope;

- the most dangerous situation of active load development on the roof support occurs during the stoppage of the face on areas with the coarse- and medium-layered structure in the period preceding to its caving; here is permitted the stoppage of the face lasting up to 6 hours at the previous average daily advance rates $V_a \geq 7,5 - 8,0 \text{ m/day}$ and up to 12 hours at $V_a \geq 9,0 - 10 \text{ m/day}$;
- if there is nonfulfillment of above-mentioned conditions in the case of stoppage of the face lasting more than 6 hours is recommended to push sections to the wall face; this operation provides a reserve of resistance reaction of the roof support up to 35% and save for some time the required height of hydraulic props extension;
- with the thin- and medium-layered structure of the main roof a safe (from the point of view of the roof support caving on the “rigid base”) time of stoppage is up to 12 hours during the previous advance rates not less than 5 m/day; in the case of $V_a < 5 \text{ m/day}$ and the duration of the stoppage more than 6 hours is recommended to perform the operation of all sections pushing to the wall face.

Another position of summarizing of obtained results is substantiation of directions for further research with the final objective to expand recommendations on the selection of rational technological parameters of coal mining.

1) Requires constant monitoring of the operation of various stoping equipment realizing coal mining in different mining and geological conditions for the expansion of the base of the experimental data on the influence of geomechanical and technological factors on the load development on the powered support. Here already proven methodology of initial data accumulation should be supplemented with the fixation of pressure relief in overloaded hydraulic props; should also be explore the process of redistribution of the load on adjacent sections during their advancing and shifting in the yieldable operation mode. The relevance of such research is the timely detection of a possible outbreak of occurrence of the process of landing on the “rigid base” and the development of measures for its prevention.

2) Complex of researches on the basis of multifactorial modeling on the identification of regularities of the connection of load on the support with parameters of the structure and mechanical properties of lithological differences of the roof and

bed in the whole range of changes in the complex mining and geological conditions of flat coal seams is actual. Logical result is the creation of set of recommendations on the selection of rational parameters of extraction works and related equipment depending from geomechanical conditions of the coal seams mining.

3) On the basis of planned researches reasonable to develop a strategy for the development of the coal mining in for the nearest and medium term perspective in terms of the improvement of technological schemes of the coal-face operation and the creation of package of technical requirements for newly constructed stoping equipment relating to conditions of Western Donbass for example.

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ВІДГУК

На дипломну роботу магістра студента групи _____ ГРг-14м
Державного ВНЗ «Національний гірничий університет» за спеціальністю
8.05030101 Розробка родовищ та видобування корисних копалин (підземним
способом)

Лисенка Романа Сергійовича

(прізвище, ім'я та по батькові)

виконану за темою: Дослідити механізм навантаження кріплення
очисних та підготовчих виробок на шахті “Самарська” ШУ “Тернівське”

Обсяг пояснювальної записки:

записка 203 с.

таблиць 1

рисуноків 49

Загальна характеристика виконання роботи

Обрана тема дипломної роботи актуальна у зв'язку з тим, що пропонувані рекомендації по параметрах ведення очисних робіт в складних гірничо-геологічних умовах Західного Донбасу дозволяють не тільки забезпечити стабільну добичу вугілля та економію, а і підвищити рівень безпеки та безаварійність робіт. Дослідження процесу формування навантаження в період зупинки забою вирішують питання щодо запобігання посадки кріплення комплексу на “жорстку базу”. Експериментально встановлені закономірності прояву гірського тиску в період проведенні очисних та підготовчих робіт практично повністю співпадають з результатами шахтних досліджень. Також отримані результати є обґрунтуванням для напрямків подальших досліджень з кінцевою метою розробки та розширення рекомендацій щодо вибору раціональних технологічних параметрів виїмки вугілля. Запропоновані

рекомендації доцільно впровадити в практику діяльності досліджуваного підприємства.

Зміст роботи повністю відповідає меті роботи і поставленим завданням.

Оформлення пояснювальної записки дипломної роботи виконано у відповідності із стандартами ЄСКД.

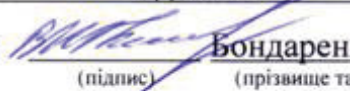
Ступінь самостійності виконання дипломної роботи високий.

Проаналізовано велику кількість джерел та літератури.

Студент, в процесі написання дипломної роботи проявив такі якості, як організаторські здібності, вміння працювати в команді, ініціативність, націленість на результат, відповідальність, вміння працювати з декількома завданнями одночасно.

Дипломна робота магістра відповідає вимогам, що пред'являються до дипломних робіт, і може бути рекомендована до захисту на засіданні ДЕК.

Оцінка дипломної роботи магістра _____ "відмінно"

Науковий керівник проф., д-р техн. наук  Бондаренко В. І.
(підпис) (прізвище та ініціали)

« 19 » 06 2015 р.

РЕЦЕНЗІЯ

на дипломну роботу магістра студента групи ГРг-14м

Державного ВНЗ «Національний гірничий університет» за спеціальністю 8.05030101 Розробка родовищ та видобування корисних копалин (підземним способом)

Лисенка Романа Сергійовича

(прізвище, ім'я та по батькові)

Тема роботи: Дослідити механізм навантаження кріплення очисних та підготовчих виробок на шахті “Самарська” ШУ “Тернівське”, що виконана за матеріалами ДТЕК ШУ “Тернівське” шахта “Самарська”
назва організації (підприємства)

Дипломна робота студента Лисенко Р. С. виконана на актуальну на сьогоднішній день тему, оскільки успішний розвиток вугільної промисловості України, з точки зору технічної та технологічної політики, можливий тільки за умови концентрації гірничих робіт та інтенсифікації підземного виробництва на базі ефективного використання сучасних техніки і технологій. У роботі сформовані закономірності навантаження кріплення та рекомендації по параметрах ведення очисних робіт для гірничо-геологічних і гірничотехнічних умов Західного Донбасу на базі моніторингу роботи стругової установки. Також автор проводить розширення галузі використання рекомендацій на базі моделювання поведінки вуглевміщуючої товщі в околиці очисних робіт.

У вступі обґрунтовується актуальність обраної теми дипломної роботи, відбивається ступінь вивченості проблеми, визначаються мета і задачі дослідження, формується практична значимість рекомендацій і пропозицій.

Перший та другий розділи роботи присвячений загальній відомості про математичне моделювання геомеханічних систем в гірничій справі, тенденції застосування методу скінченних елементів, технології обчислювального експерименту, проведення досліджень напружено-деформованого стану та обчислювального експерименту методом кінцевих елементів.

В третьому розділі роботи визначено сутність та особливості зсування порід надвугільної товщі на кінцевих ділянках лави і механізм навантаження кріплення виїмкових штреків. Також досліджені гірничотехнічні і геомеханічні аспекти повторного використання виїмальних штреків та проаналізовані шахтні дослідження розвитку переміщень породного контуру цих штреків в процесі посування очисного забою.

У четвертому розділі дипломної роботи автором проведений досить докладний і кваліфікований аналіз механізму деформування надвугільної товщі порід при очисній виїмці пологих вугільних пластів Західного Донбасу та досліджено формування зони шарнірно-блокового зсування порід.

У п'ятому розділі роботи розроблено та проаналізовано механізм навантаження кріплення очисного комплексу при зрушенні надвугільної товщі слабких порід. Детально опрацьовані такі механізми впливу, як механізм впливу структури надвугільної товщі, механізм впливу деформаційних властивостей породи, механізм впливу міцності і реологічних характеристик та механізм впливу кроку посадки основної покрівлі та реакції опору кріплення.

У шостому розділі представлений глибокий аналіз і дослідження впливу технологічних параметрів на навантаження механізованого кріплення. Автором розроблені закономірності зміни тиску в гідростійках кріплення при різних швидкостях посування очисного вибою. Також проведено моніторинг взаємодії секцій кріплення з породами кривлі по довжині лави.

Сьомий розділ присвячений впливу на навантаження механізованого кріплення таких геомеханічних факторів, як структура вуглевміщуючої товщі і глибина розташування очисного вибою. Виведено закономірності впливу кроку посадки основної кривлі та тривалості зупинки очисного вибою.

Восьмий розділ присвячений комп'ютерному моделюванню зсування вуглевміщуючого масиву слабких порід в околиці очисного забою. Автор розробив комп'ютерної моделі і обґрунтував її параметри. Також виконав розрахунок і аналіз напружено-деформованого стану порід та повних переміщень вуглевміщуючого породного масиву.

Наведені висновки і пропозиції мають конкретний характер, повністю впливають з аналізу, проведеного в роботі. Виконані розрахунки та дослідження можуть мати практичне застосування в подальшій діяльності підприємства.

Автор дипломної роботи показав відмінну здатність формулювати власну точку зору з даного питання, обробив велику кількість наукового матеріалу, на високому теоретичному та методологічному рівні. Увесь матеріал в роботі логічно структурований, написаний науковим стилем викладу.

Дипломна робота грамотно оформлена. Вона містить велику кількість табличного та ілюстративного матеріалу, що дозволяє більш наочно розкрити її основні результати. Істотних недоліків у дипломній роботі не виявлено.

Дипломна робота магістра відповідає пред'явленим вимогам і рекомендована до захисту на засіданні ДЕК та заслуговує оцінку "відмінно".

Рецензент

Професор кафедри підземної розробки родовищ,

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