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Studiengang: Advanced Mineral Resources Development

Master Thesis

"**Researching of modern technologies of the mechanical properties determination by simulation procedures for the purposes of controlling of the slightly metamorphosed massif stability"**

"Erforschung moderner Technologien zur Bestimmung der mechanischen Eigenschaften durch Simulationsverfahren zur Kontrolle der leicht metamorphosierten Massivstabilität"

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РЕФЕРАТ

Пояснювальна записка: 92 сторінки, 5 таблиць, 24 малюнка, 77 джерел.

Об'єкт дослідження: процес зміцнення слабометаморфічного масиву системою кріплення.

Мета дослідження полягає в обґрунтуванні параметрів системи кріплення з урахуванням закономірності зміни НДС оточуючих порід для забезпечення стійкості гірського масиву.

У першому розділі виконано аналіз стану проблеми і визначені шляхи її науково-практичного рішення. Розглянуті шляхи вивчення механічних властивостей корисних копалин та гірського масиву методами моделювання. Розглянуті шахтні та лабораторні методи визначення механічних властивостей порід та чисельні методи моделювання.

У другому розділі зроблений вибір методу дослідження слабометаморфічних порід і побудова методу дослідження. Розглянуті основні особливості методу кінцевих елементів (МКЕ) та обґрунтовано вибір програми для дослідження. Обрані початкові та граничні умови відповідно до гірничо-геологічних умов ПрАТ «ДТЕК Павлоградвугілля» ВСП «Шахта "Дніпровська"». Надані загальні відомості про підприємство, за умовами якого проводилась дослідницька робота. За допомогою програми 3D-моделювання ”SolidWorks”, побудовані декілька варіантів напружено-деформованого стану масиву з різними типами кріплення виробки. Після модулювання процесів напруження виробки, зроблений аналіз та висновки по НДС масиву.

У третьому розділі розглянуті рекомендації по забезпеченню стійкості виробок. Обґрунтовано вибір системи кріплення виробки для комп'ютерного моделювання. Проведено аналіз застосованих конструкцій кріплення для гірничо-геологічних умов Західного Донбасу. Зроблені рекомендації параметрів кріплення для умов ПрАТ «ДТЕК Павлоградвугілля» ВСП «Шахта "Дніпровська"» та обґрунтовано можливе ресурсозбереження при кріпленні допоміжної виробки. У розділі «Охорона праці» визначенні заходи по запобіганню пожеж та заходи безпечного встановлення анкерного кріплення, прорахована очистка пилу з вихідного струменю повітря та заходи безпеки. У економічному розділі розглянуті показники, що свідчать про економічну привабливість прийнятого рішення.

STRUCTURAL ABSTRACT

Report: 92 pages, 5 tables, 24 figures, 77 references.

Object of study: the process of stabilization of a slightly metamorphosed rock massif by the support system.

Purpose of the study is in reasoning of the parameters of support system, taking into account the regularity of changes in the stress-strain state of the surrounding rocks to ensure the stability of the rock massif.

The first section analyzes the state of the problem and identifies ways of its scientific and practical solution. It is researched the ways of studying the mechanical properties of minerals and rocks by simulation methods. Field and laboratory methods for determining mechanical properties of rocks and numerical simulation methods are analyzed.

In the second section it is made a choice of slightly metamorphic rocks investigation method and the construction of the research method. The main features of the finite element method (FEM) are viewed and the choice of the program for investigation is substantiated. Selected initial and boundary conditions in accordance with the mining and geological conditions of PJSC "DTEK Pavlogradvugillya" PSD "Dniprovska" mine. General information about the company under the terms of which the research was conducted is provided. By the means of 3D-simulation program “SolidWorks”, several variants of stress-strain state of the massif with different types of the working support are constructed. After simulation of the processes of working loads, the analysis and conclusions of the rock massif stress-strain state are made.

In the third section it is viewed the recommendations for providing of the working support stability. The choice of the working support system for computer simulation is based. An analysis of the support structures used for the mining and geological conditions of Western Donbass is carried out. Recommendations of support parameters for the conditions of PJSC "DTEK Pavlogradvugillya" PSD "Dniprovska" mine and substantiation of the resource saving at the supporting of development drift were made. In the section "Labor safety" defines the measures for fire prevention and measures for the safe installation of rock bolts, the calculation of dust from the return ventilation air and protection activities. In the economic section, the indices that indicate the economic attractiveness of the decision, are viewed.

ЗМІСТ

CONTENT

ВСТУП

Актуальність проблеми. Розробка родовищ корисних копалин залишається актуальним завданням для становлення енергобезпеки. Значний відсоток собівартості добутку вугілля випадає на кріплення та підтримання гірничих виробок. Задача оптимізації використання ресурсів ґрунтується на покращенні найбільш економічних технологій для кріплення виробок, заснованих на управлінні НДС гірського масиву з керуванням проявами гірського тиску. Ця проблема актуальна для шахт Західного Донбасу. В гірничо-геологічних умовах цих шахт, слабометаморфічні породи викликають розвиток високого гірського тиску на кріплення та контур виробок. Також мають вплив ряд послаблюючих факторів, таких як тріщинуватість, насиченість водою та реологія. Властивості порід мають суттєве значення при виборі технології ведення гірничих робіт. Тому важливими залишаються дослідження, які направлені на вивчення властивостей корисних копалин різними методами. Серед методів необхідно виділити з одного боку найбільш сучасні, а з іншого найбільш ефективні для умов розробки родовищ слабометаморфічних порід Західного Донбасу. Механічні напруги та тиск впливають на кожен пласт та виробки. Тому дослідження спрямовані на вивчення сучасних технологій механічних властивостей порід методами моделювання для рішення завдань керування стійкістю масиву гірських порід. За рахунок залучення процесів моделювання, стає можливим прорахувати найбільш економічно вигідні варіанти системи кріплення та підтримання стійкості виробки.

Ідея досліджень полягає у використанні закономірності зміни напружено-деформованого стану (НДС) системи «масив-кріплення» для вирішення завдань управління стійкістю слабометаморфічного масиву.

Мета дослідження полягає в обґрунтуванні параметрів системи кріплення з урахуванням закономірності зміни НДС оточуючих порід для забезпечення стійкості гірського масиву.

Завдання дослідження:

1. Виконати аналіз методів моделювання при вивченні механічних властивостей гірського масиву;
2. Побудувати і обґрунтувати геомеханічні моделі обчислювального експерименту на базі обраного методу моделювання;
3. Провести аналіз отриманого НДС для умов ПрАТ «ДТЕК Павлоградвугілля» ВСП «Шахта "Дніпровська"» та обґрунтувати раціональні параметри системи кріплення.

Об’єкт дослідження: процес зміцнення слабометаморфічного масиву системою кріплення.

Предмет дослідження: закономірності розвитку зон НДС порід та їх зв’язок з досліджуваними параметрами.

Методи досліджень: для вирішення поставлених завдань використаний комплексний підхід, що включає аналіз методів моделювання, обчислюваний експеримент та аналіз НДС «масив-кріплення».

Практичне цінність роботи полягає в обґрунтуванні геомеханічної моделі слабометаморфічного вуглевмісного масиву порід в околиці підготовчої виробки, яка реалізована за допомогою пакета сучасних комп’ютерних програм і їх додатків; обґрунтуванні параметрів прийняття технологічних рішень при виборі систем кріплення виробок.

Наукова новизна роботи полягає у дослідженні НДС масиву при обґрунтуванні комп’ютерної геомеханічної моделі підготовчої виробки з урахуванням слабометаморфічного масиву порід та характеристик елементів рамно-анкерної системи кріплення.

INTRODUCTION

Significance of the problem: mining of mineral deposits remains a topical issue for the development of energy industry. A significant percentage of coal mining costs is accounted for the costs on mining workings development and their support. The problem of cost-effective usage of resources is come out as the improvement of the most economically viable technologies for support of workings based on managing the stress-strain state of the rock massif with the manifestations of rock pressure. This issue is especially of current interest for Western Donbass mines. In the conditions of these mines, slightly metamorphosed rocks generate the development of high rock pressure on the mine support and the working outline. In addition to this, there is an influence of the weakening factors such as rock fracturing, water-saturation and rheology. Rock properties have an important role at choosing of mining technology. Consequently, investigations that focus on the study of mineral properties by various methods remain important. It should be highlighted both the most modern and effective methods for the development of the slightly metamorphosed rocks of the Western Donbass. Mechanical stresses and pressures act on every layer and working. For this purpose, the studies are aimed to investigate the modern technologies of mechanical properties researches by simulation methods for solving problems of stability control of the rock massif. By engaging in simulation processes, it becomes possible to calculate the most cost-effective options for support systems and maintaining production stability.

Investigation concept is in use of regularity of changes of the stress-strain state of system “rock massif-support system” to solve the problems of stability control of a slightly metamorphosed rock massif.

Purpose of the study is in substantiation of support system parameters, taking into account the regularity of changes in the stress-strain state of the surrounding rocks to ensure the stability of the rock massif.

Targets of the study:

1. To carry out analysis of simulation methods in the study of mechanical properties of the rock massif;

2. To design and give an argumentation of the geomechanical models of simulation experiment on the basis of the chosen simulation method;

3. To make an analysis of the SSS received for the conditions of PJSC “DTEK Pavlogradvugillya” PSD “Dniprovska” mine and substantiate the rational parameters of the support system.

Object of study: the process of stabilization of a slightly metamorphic rock massif by the support system.

Subject of study: regularity of SSS zones development of rocks and their relationship with the studied parameters.

Methods: an integrated approach was used to solve the set tasks, including the analysis of simulation methods, the simulation experiment, and the analysis of the SSS of "rock massif-support" system.

The practical value of work is to substantiate the geomechanical model of a slightly metamorphic carbonaceous massif of rocks in the vicinity of development drift, which is realized with the help of a package of modern computer programs and their applications; substantiation of the parameters of technological decisions in the choice of the working support system.

The scientific novelty of the work is in the study of the SSS of rock massif when reasoned a computer geomechanical model of development drift taking into account a slightly metamorphic rock massif and characteristics of the elements of the “frame-rock bolts” support system.

PART 1

INVESTIGATION OF THE ROCK MASSIF MECHANICAL PROPERTIES BY SIMULATION PROCEDURES

1.1 Current state of Ukraine's coal industry

Coal industry is one of the main parts of the world's fuel industry. Coal mining enterprises located in 7 regions of Ukraine and it is the main raw material of the country. There are 3 main coal basins such as Donbas, Lviv-Volhynian and Dnipro brown coal mining basins. This fossil fuel can ensure energy independence and contribute development of different industries, especially metallurgical and chemical. Reserves of this fuel in Ukraine are estimated at 60 billion tons, among them approximately 10 billion tons are economically extractable [1].

The largest privately owned energy company in Ukraine is DTEK. DTEK Energo produces coal and coking coal, which is enriched in Ukrainian factories. According to the official statistics of the company, in 2018 the total coal production was 24.1 million tons. The largest coal mining enterprise in Ukraine is PJSC "DTEK Pavlogradvugillya", that is situated in the Western Donbass region. There are 10 mines in this company and the enterprise employs more than 26 thousand people.

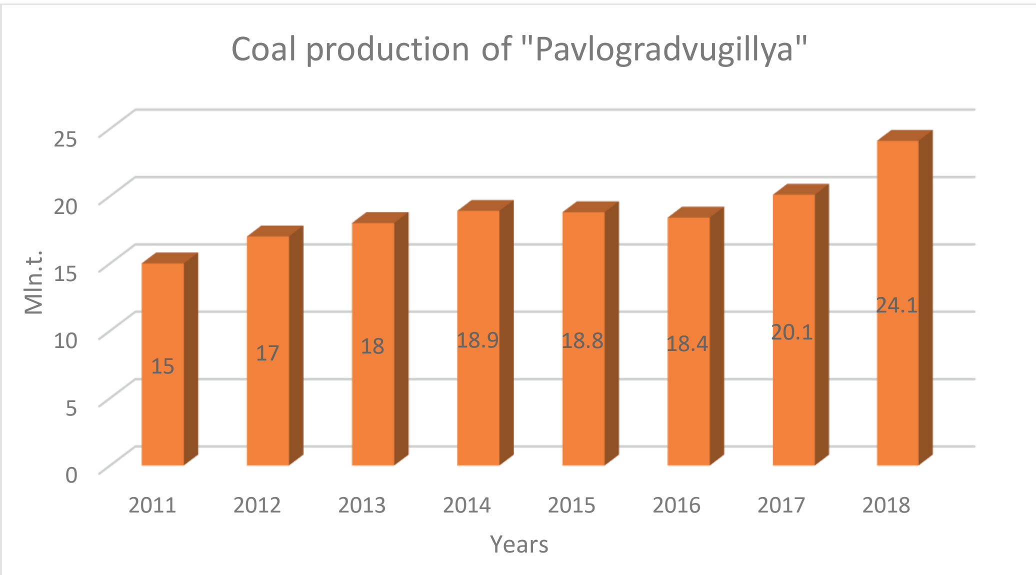

Figure 1. Coal production of "Pavlogradvugillya"

DTEK Energo produces steam and coking coal, which is enriched on Ukrainian factories. Gas coal is extracted by the miners of DTEK Pavlogradvugillya, DTEK Dobropillyavugol and the Belozerska mine [2].

The mines of the Western Donbass are characterized by rough ground conditions due to mining operations in stratified massif of soft rocks, some rocks are water saturated and smashed by intensive laminationт[3].

1.2 Mechanical characteristics of rocks

Mechanical properties, strains and pressures have an influence on each productive layer and mining workings. These specifications are very significant for mining engineers and geologists. In coal industry, a number of emergency conditions can be obtained due to the wrong interpretation of mechanical properties of rocks. But nowadays in virtue of monitoring of the geomechanical processes as well as geotechnical measurements and simulations, engineers have an opportunity to control the massif stability and prevent accidents.

Mechanical properties of rocks play an important role in the field development and design. These properties characterize the behavior of rocks under the application of various forces. There are different reactions on external mechanical action depending on the structural rock features, type, direction, value and time of the load application. These properties are divided into a number of groups: **strength characteristics**, characterizing the ability of rocks to collapse irreversibly with discontinuity; **strain characteristics**, characterizing the ability of the rock to deform elastically under load; **rheological**, characterizing the change in mechanical characteristics under the long-term impact of loads**; plastic**, characterizing the ability of rocks to deform irreversibly without breaking the continuity [4].

Special attention is paid to the such rock characteristics as heterogeneity, jointing, porosity, strength, lithology, structure, texture, mode of occurrence, depth of occurrence, faulting, density and seam thickness. These parameters are the basis for the follow-on development of the physics and numerical schemes [5].

It is existed a number of methods for mechanical rock characteristics determination. Among them there are such indirect methods with correlations with classification indices as the Rock quality designation [6], the Rock mass rating [6], the Q-system [7] and the Geological strength index [8].

At the present time, there are many techniques for researching the mechanical rock features, which are based on usage of core analyzes and geomechanical modeling of deformations, stresses and rock strengths to achieve better design and maintenance workings. Thus, in this paper we consider the basic modeling methods for determining the mechanical properties of rocks.

Strain characteristics of rocks

The ability of rocks to deform elastically is characterized by values such as modulus of elasticity - E (Young's modulus), Poison's ratio (ν), shear modulus (G) and bulk modulus (K).

Modulus of elasticity (E) defines tensile elasticity. It is a ratio of normal stress to axial strain. It is a trend of the subject to deform along axis when resistance forces are attached along that axis [9], [4]:

$$E = \frac{\sigma}{\varepsilon}$$

where σ - is a normal stress (compressive or tensile) and ε - is axial strain (deformation).

Poisson's ratio (ν) is the ratio of transverse compression to the axial tension.

The shear modulus (G) is coefficient of proportionality of shear strength to corresponding shearing strain, that characterize the deformation of the material:

$$G = \frac{\tau}{\gamma}$$

where τ - is shear strength and γ- is shearing strain.

The bulk modulus (K) characterizes the reaction of the material to uniform pressure. These moduli are connected between each other for isotropic materials by the equation [9], [4]:

$$2G(1 + \nu) = E = 3K(1 - 2\nu)$$

Due to the fact that the real relationship between stresses and strains for rocks is curvilinear and plastic deformations often appear immediately after loading, it is needed to introduce the concept of the **modulus of deformation (E_m)**. According to the International Society for Rock Mechanics (ISRM), this modulus is the relation of stress to appropriate strain within load of rock mass, including elastic and inelastic behavior [10].

Strength characteristics of rocks

Yield value - is the value of the stress when the sample starts to deform plastically. Yield point - it is a point where starts elastic and plastic deformation [11].

Compressive strength [σ_c] - is a maximum compressive stress (loads) that object or material is able to withstand tending to reduce size, it is determined relatively to the original cross-sectional area [4].

$$R_c = \frac{F}{S}$$

where F - is a compressive strength, S - initial cross-sectional area.

Tensile strength [σ_t] is a maximum uniaxial tensile strength, that material or object is able to withstand tending to elongate. It means, tensile strength resists tension, whereas compressive strength resists compression [4].

$$R_t = \frac{F}{S}$$

where F - is a uniaxial tensile strength, S - initial cross-sectional area.

Shear strength [σ_s] is uniaxial shear strength in the absence of normal stresses, the ratio of the critical force F to the shear surface S.

In addition to this, shear strength for coherent rocks is determined by **adhesion** and **angle of shear resistance**. Shear strength of the dense rock in a certain range of stress is defined by linear function of Coulomb [4]:

$$\tau = \sigma_n tg\phi + c$$

where τ - ultimate shear strength, Pa, σ_n - normal stress, Pa, tgϕ –internal friction coefficient, c - adhesion, Pa, ϕ - angle of shear resistance, ◦.

Bending strength - maximum value of stress at which the rock sample is breaking under the bending load [4].

Rheological characteristics

It is a set of properties that determine the ability of rocks to change in time the stress-strain state in the field of action of mechanical forces. The main rheological properties include: elasticity, plasticity, strength, viscosity, creep, stress relaxation. Rheological properties characterize the change (increasing) in time of deformations of rocks at constant stress (**creep phenomenon**) or the change (drop) in stresses at constant strain (**relaxation phenomenon**). Creep and stress relaxation are associated with the transition of elastic deformations into plastic [4], [12].

Plastic properties

Plastic deformation is different from destructive since it occurs without a clear rock discontinuity. A greater amount of energy is spent on additional deformation of plastic rock in order to destroy it than on the destruction of elastic (brittle) rock with the same tensile strength. The ratio of work spent on the destruction of a real sample to the work spent on the destruction of an ideally brittle rock with the same σ_c is **the plasticity coefficient** [4].

1.3 Mining and in-situ experiments

During mining workings there are a lot of deformation processes in the rock massif. Accordingly, it can be not enough only theoretical calculations for evaluation of rock deformation, stability and strength of the mine workings. In-situ tests give an opportunity to receive the most accurate information about the rock stress-strain behavior as well as the possibility of studying relatively large volumes of rocks and studying such types of soils, samples of which hard to select with intact structure for laboratory testing. But there is a number of disadvantages of this method such as a large measurement base, complexity, required accuracy, theoretical justification and large financial expenses. This method of simulation procedures is the most acceptable with the availability of underground workings, wells, complex measurement equipment and proven technology for their implementation [13].

As It has been already expounded by a variety of scientists [14,15], the diversity of block size, extent of fracturing, sheared zones, faults, blast damage, research operation and method of investigation have a direct influence on the test results. It should be conducted several tests to check the accuracy of the results.

According to the list of different structures that have been built on, in and of rock [17,18] to the underground mining structures it was referred shaft, pillar, draft and stope design; drilling and blasting; fragmentation, cavability of rock and ore, repairing of rock burst, mechanized excavation and in-situ recovery.

It can be highlighted such main types of in-situ tests that determine deformation characteristics (deformation modulus) of rocks such as plate bearing tests, pressuremeter, Cone penetration test (CPT) and Standard penetration test (SPT), geophysics, Dilatometer test (DMT) [16].

1.3.1 Plate bearing test and pressuremeter

In regard to the loading of certain part of rock mass, there are such widespread tests for the definition of deformation modulus and ultimate bearing capacity such as plate bearing test (PBT). There are 2 common types: plate loading and jacking test. During PJT it is applied extensometer, - is a device for measuring changes in the length of an object. PBT is usually conducted in accordance to the ASTM D1194 [19]. Within **PLD**, it is excavated a prospecting borehole with a needed deepness. The dimension of a well should be leastwise 5 times of the test plate. In the center of the well, it is creating a hole with the same size as plate. A square or circular formed plate is used as load bearing plate. A load is transmitting to the plate via the column that is situated in the center. It is used 4 dial gauges on the plate, that register every increase of the load. The main advantage of this type of test is a possibility to determine rock properties that can be affected by low scale effects (inhomogeneity, jointing). The main disadvantage of this method is the reliance of the test for homogeneous deposits [19].

It was established by scientists from the USA [20], that PBT is a reliable method for determining bearing capacity of roof and floor of seams in coal mines. For minimizing an extrapolation, it is better to use plates of large areas.

It is also used **Goodman jack test,** in 76 mm boreholes, with 2 possible models: a 12 piston model for usage in hard rocks, and a 3 piston model in soft rock and stiff clays to figure out consolidation properties. Although, it was proposed several corrections to the measurement of this test, the precision is still not satisfied. In the research literature it was noted 2 factors that decrease values of the test. The first one is not a complete connection between rock and plate if the borehole is undersized or oversized and the second one is the upper limitation on the used hydraulic pressure. Regarding the first factor, it was proposed a diameter-related correction on the grounds of numerical simulation's results. For the second factor, it was calculated the upper limitation for the hold pressure in virtue of the Mohr-Coulomb and tensile failure criteria [15,17,21].

In addition to these types, there are also such tests as large flat jack and radial jacking test. During **radial jacking**, it is used a test chamber with circular cross profile with uniformly distributed radial loading. Not only anisotropic deformability of rocks can be measured, but also time-dependence deformation, as applied a constant pressure and deformation is recording during the time. This method is costlier and more precise in comparison to jacking tests or others and should be used when it is needed to obtain values which show real rock mass properties not considering the influence of discontinuities [22,23]. During **large flat jack** rock volume that is need to be loaded is smaller than for radial jacking, but larger than for plate loading test. The jacks consist of 2 steel parts with thickness less than 1 mm, welded around the edges, filled with hydraulic liquids for applying a uniform pressure. It is used leastwise three circles of loading before the difference in minimum and maximum loads in last 2 loadings does not surpass 5% of total deformation [24,23].

According to the investigations [25], where it was used 3 types of tests: PJT and PLT, and Goodman jack test, it was obtained that the PJT gives the most accurate result, during which the deformations were tested by extensometers in boreholes.

During PLT, the deformations have been fixed at the rock area of loading application. And within radial jacking test, it is used 2 crimple coarse bearing plate with angular divergence 90˚ that can be unclench inside the borehole by a range of pistons. In the last test the deformations are measured by 2 detectors fixed in both ends of the plates. PLT gives wrong results of the measures because experiments were made in the faulty area from blasting.

According to ASTM D4719 [26], **pressuremeter testing** is applied for deformation modulus specification of hard rock, semi-rocky and organic soil. It is used a probe that interposed to the hole at the needed depth. The probe is an elastic capsule, that is extended under the pressure and exerts uniform pressure on the walls of the well. The walls start to deform under the constant pressure and after some time it is noted a volume rising that is needed to control pressure. There are 2 types of pressuremeter test: the first one with identical pressure increasing (stress controlled test) and the second is increasing of volume with equal adding (strain controlled test). Pressuremeter is also used for measurement of coefficient of lateral earth pressure, yield and limit pressures among others. Advantages of this testing are application almost for any type of rock, a quick way for soil investigation, the borehole diameter can be not large, more economical than plate load testing, compactness of the equipment set for testing. The only one disadvantage is the possible difficulties if walls of engineering geological borehole liable to move [26,27].

1.3.2 Cone penetration test (CPT) and Standard penetration test (SPT)

One of the popular dynamic in-situ method is **Standard penetration test (SPT)** that is used for determining the resistance of soil by a dynamic implantation of a detachable sampler (sample tube) into the soil with sampling of broken rocks. The test operation is represented in the number of standards [28-30]. It is conducted by injection of sample tube into the rock under the weight 63.5 kg of hammer falling from the height of 760 mm. It records the amount of percussions the tube required to enter every 150 mm of depth till 300 (or 450 mm). The immersion is conducted in 3 steps. During 1-st step the tube immerses without counting the number of blows. During 2 and 3 steps, the total number of blows needed for tube immersion to the requires depth is counted.

This number is called "standard penetration resistance" and indicated like "N-value". This test is simple and inexpensive. The main aim of this method is to give a relative density of granular deposits as well as internal friction angle and immersion in loose soil. However, the exact values should be found using correlations between field and laboratory tests. The rock strength parameters are approximate, but it can give a beneficial information about soil conditions, where it is difficult to have borehole samples of normal quality for sand, clay, soft rocks. The test is simple, but its main disadvantaged are time-consuming, the samples for testing are broken, the usage of this method in cohesive rocks is limited [31].

In recent years, it is becoming popular **Cone penetration tests,** since they are more productive and give additional information about rock properties due to measuring of pore pressure and shear wave velocity. The standard cone has a diameter of cylinder 35.7 mm, the angle at the apex of the axial section is 60^o, the projection (base) area of the cone is 10 cm^2, the area of the friction clutch (side surface) is 150 cm^2. There is a number of cone types. The strain gage (CPT) is able to measure front resistance and friction along the lateral surface [31].

Last years, it is produced seismic probes (SCPTU) which additionally allow to measure the velocity of a shear wave. In the work [32], It was studied the areas of potential zones of discontinuity of the carbonaceous massif of the Western Donbass and used the approach of calculating seismic characteristics such as coherence and curvature of the wave field, using seismic cone penetration test.

Modulus of deformation for sand and clay soils can be defined with cone penetrometer value (q_c):

$$E_d=\alpha_c q_c,$$

where α_c - correlation coefficient depending on soil type and cone penetration resistance, q_c - cone penetration resistance [33].

q_c, MPa		α_c	Soil
<0.7		3-8	Low plasticity clay
0.7-2.0		2-5	
>2.0		1-2.5	
>2.0		3-6	Low plasticity silt
<2.0		1-3	
<2.0		2-6	Highly plastic silt and clay
<1.2		2-8	Organic silt
<0.7 at natural moisture, W %	50-100	1.5-4.0	Peat and Organic Clay
	100-200	1.0-1.5	
	>200	0.4-1.0	

Table 1. Correlations coefficient and Cone penetration resistance

The main elastic parameters are the deformation modulus and Poisson's ratio. Using solutions of the theory of elasticity, the remaining modules can be determined using the expressions (Table 2) [33]:

	Shear modulus (G)	**Modulus of elasticity (E)**	**Bulk modulus (K)**	**Poisson's ratio (ν)**
G,E	G	E	$\frac{GE}{9G-3E}$	$\frac{E-2G}{2G}$
G,K	G	$\frac{9GK}{3K+G}$	K	$\frac{3K-2G}{2(3K+G)}$
G, ν	G	2G(1+ν)	$\frac{2G(1+\nu)}{3(1-2\nu)}$	ν
G, ν	$\frac{E}{2(1+\nu)}$	E	$\frac{E}{3(1-2\nu)}$	ν

Table 2. Expressions for modulus determination

Type of soil	ν
Coal	0.3-0.35
Coarse rock	0.27
Sand and clay sand	0.30
Loam	0.35
Clay	0.42

Table 3. Poisson's ratio for different types of soils

1.3.3 Geophysics

There are two the most widespread methods for studying of coal well log such as **scattered gamma radiation and natural gamma radiation**. By gamma-method, well logs are studied by researching natural gamma radiation of rocks, that compose well log. By scattered gamma radiation method, it is studying gamma ray scattering effect upon irradiation of rocks with a gamma ray source. The intensity of this radiation is related to the density of the rocks. **Radioactive tracer method** studies the intensity of gamma radiation of isotopes injected into the rock during drilling or during subsequent well cleanout by these solutions. The method makes it possible to identify porous beds in well sections. There is also such method as induced activity with the help of which it is possible to determine the content of minerals in the rocks [34].

Also there are 2 geophysical testing (**down-hole and cross-hole geophysics**). In both cross-hole and down-hole geophysics tests, it is determined primary compressional and secondary shear wave velocities as a function of depth, which further allow getting Young's and shear modulus as well as Poisson's ratio. During down-hole geophysics it is determined VP and VS velocities in rocks crossed by a single well. It is conducted a measurement of a transit time of a wave created by a seismic source at the surface and 2 three-component geophones with interval of 1 meter between them. Registration of test results were performed each 1 meter and the maximum depth was 91 meters. During cross-hole geophysics tests, it is also performed a measurement of VP and VS velocities. According to the ASTM D4428 [47], the depth of the well was 100 meters. It is needed to conduct a number of measurements at various depth to fix spreading time of waves [35].

1.3.4 Dilatometer test

Dilatometer is a device for measuring the changes in body size caused by external influence of heat (through heat transfer), pressure, electric and magnetic fields, ionizing radiation or other factors. The most significant characteristic of a dilatometer is its sensitivity to absolute change in body size. This device has a system for temperature measurement.

Audibert-Arnu dilatometer is used for the coal testing. The test is based on the heating of the briquette rod pressed from coal and measuring the changes of its length depending on the temperature changing. This method allows measuring not only coal expansion, but also certain temperature points. The program calculates softening point, temperatures of the maximum compression and expansion. There is also Sheffield dilatometer, that is differed from previous one by not briquetting coal sample. It is just tamped down to the tube bottom, which reduces initial shrinkage and since the diameter of the piston is less than the diameter of the tube, it is achieved for flow coals in the plastic state that the piston acts as a penetrometer. There are also such types as Shevenar dilatometer and Hoffman dilatometer [36].

Summary

There is a number of different methods that can determine physical and mechanical characteristics of minerals and soils. Variable methods are suitable for different types characteristics and depending on the type of mineral and searching properties, it is better to use this or other method. A complex studying of mechanical soil characteristics should include not only in-situ tests but also laboratory investigations for the most accurate and appropriate meanings.

Plate bearing test and pressuremeter tests give the most reliable meanings of deformation characteristics in comparison to Cone and Standard penetration tests, as it is used elasticity and plasticity theories and for the next tests correlation dependencies are used. Also, geophysics methods are highly used for determination of coal properties as well as other minerals.

A correct assessment of the soil mechanical properties allows using correctly the bearing capacity of soils and accurately determine their deformations, which is the key to safe and economical solutions in foundation engineering.

1.4 Laboratory based methods

During borehole drilling some samples of the rocks are taken for laboratory investigations. In laboratory there are different tests of soils for determination their characteristics under static load or dynamic effects. During these tests, soil is loaded and unloaded, shifted as well as deformed in different ways. Laboratories studies and tests are accurate enough. In comparison to field methods, the laboratory tests reduce labor and explicit costs, that allows increasing the number of experiments and their accuracy by statistic processing of special values of the indicator. Both in-situ and laboratory methods have some advantages and disadvantages. As the result, the best option is to use both of these methods for obtaining the most precise values of characteristics.

Soils show linearly elastic behavior to relatively small loads. However, during unloading of samples, residual deformation occurs in soils. In that reason, it is believed that during loading to the limit of proportionality, Hooke's linear relations are valid:

$$E=\frac{\sigma}{\varepsilon},\ \ G=\frac{\tau}{\gamma}$$

where γ is tangential deformation and τ is internal shear and each equal increment of uniaxial stress σ corresponds to a proportional increase in the strain ε.

However, deformations in soils nonlinearly depend on stresses during high loadings. There are such research methods for determining mechanical properties of soils in laboratories as:

- compression tests;
- soil strength tests (uniaxial compression, tension tests, simple shear, triaxial shear);
- determination of angle of friction.

1.4.1 Compression tests

Soil tests under compression are the most widespread due to their simplicity and reliability for samples testing both undisturbed and disturbed structure. There are different types of loading during this test: incremental or kinematic loading and relaxation of stresses. Compression is a process of sample compaction without its destruction (without lateral expansion). It is obtained such deformation characteristics as compressibility modulus, modulus of deformation, compression structural strength, consolidation coefficient (for sands, clays and organic soils). These moduli are obtained after tests of soil samples in compression devices (odometers) or compression-filtration devices. The test results are usually executed as graphs of the dependences of the sample deformation under the load and sample changing over time. For testing, it is used undisturbed samples with natural moisture or water-saturated, or disturbed samples with given moisture and density values [39]. Since soil compaction occurs due to changes in porosity, the results of compression tests are presented as dependency graph of the porosity coefficient and compressive stress.

It is also possible to obtain coefficient of lateral strain by compression tests. This coefficient is very useful for determining active and passive pressures during design of any enclosing constructions in the soil [37].

1.4.2 Soil strength tests

Uniaxial compression test is used for determination of deformation modulus of hard rock and semi rocky-soil. This test can be performed both on samples of circular and square cross-section. But the height of the sample should be 2.5 - 3 times higher than the diameter or side of the sample.

During preparation to the test, special attention should be paid to the quality of grinding of the working surface and their parallelism, and during the test, the accuracy of measuring of the deformation characteristics by measuring instruments and the correct alignment of the sample.

The tensile strength during uniaxial compression is defined for soils with natural moisture, air-dried basis and water-saturated basis.

In a compressive rock sample, it usually occurs heterogeneous stress state due to the friction on the plane between sample and press plates, that limits the lateral deformation of the sample near these planes. At the certain load value, the compressive sample is destroyed because of crossed tension as well as shear stresses, that appear on the inclined planes.

The average breaking load of sample is determined by value of the critical load, that was obtained during the test. After that, the ultimate compressive strength defines as ratio of average breaking load to the average cross-section [37,4].

It is quite difficult to conduct **direct uniaxial tension test**. So it is performed such indirect methods of tension test as beam bending, splitting of core samples, point load test, the Brazilian test, bending cylindrical rock beams.

For the same soil, the tensile strength is always less than compressive strength, since breaking of structural bonds can occur during tension and it is developed large irreversible deformations, and during compression, structural bonds can be only deformed and not destroyed [37,4].

The most effective way to determine uniaxial tension is the **Brazilian test**. During this test, a rock sample is loading by a compressive load that is uniformly distributed along element of cylinder or longitudinal rib of rectangular parallelepiped. Respectively the rock sample can be in form of cylinder or parallelepiped. During transverse compression it is recommended to use a cylinder rock sample [37,12].

During **direct uniaxial tension test**, it is measured a breaking force during longitudinal tension of cylindrical or prismatic soil samples through steel casing of the loading device.

During **splitting of core samples**, the sample is placed between wedges, combining the wedge blades with the line applied to the mesh sample. Sample is loaded uniformly under the press until it breaks into blocks, and then into cubes, that have such forms: 2 parallel polished surfaces and 4 surfaces created during splitting. It is recording destructive force and the average length of splitting during each cracking of the sample.

During the simple **shear strength tests**, it is measured a force that is necessary for shearing a sample at a constant normal load. If the normal load will increase, ultimate shearing strength will also increase.

As it has been already mentioned earlier, ultimate shearing strength is defined by adhesion and angle of internal friction, shear strength of the dense rock in a certain range of stress is defined by linear function of Coulomb:

$$\tau=\sigma_n \, tg\phi + c,$$

where τ - ultimate shear strength, Pa, σ_n - normal stress, Pa, $tg\phi$ - internal friction coefficient, c - adhesion, Pa, ϕ - angle of shear resistance, ◦.

If rocks are cohesive, the angle of internal friction is obtained from the slope of shear curve or from the envelope of the Mohr circle of stresses in the rectilinear coordinate system. The stresses can be obtained from the tests of triaxial compression.

Ultimate bending strength - is a critical value of loading, when sample is destroyed under bending load. The sample is in the form of beam with rectangular cross-section is supported on 2 supports and load is applied in the center [37,4].

1.4.3 Determination of angle of friction

Strength soil characteristics are very important parameters for the calculation of bearing capacity, soil stability as well as soil pressure and others. **In-plane shear** test is conducted for determination of angle of internal friction and specific adhesion for sands, clays and organic-mineral soils. They are determined by tests of soil samples in the in-plane shear devices with a fixed shear plane by shifting one part of the sample relative to the another part by a horizontal load with a preloaded of the sample with a normal to the shear plane load.

Soil shear strength is defined as the limiting average shearing stress at which the soil sample is cut along a fixed plane at a given normal stress.

To determine the particular values, it is necessary to conduct at least three tests of identical samples at different values of the normal stress.

There are 2 main types of such test:

1. During effective shear test, the normal pressure is maintained until the deformation stabilizing and then the soil is sheared at the same pressure.
2. During neutral shear test, density and humidity of the soil do not change, in other words there is no soil consolidation.

The first test is also called consolidated-drained shear test (ASTM D2850 [49]) and it is used for sand, clay and organic-mineral soils independently of their water-saturation coefficient for determination of effective adhesion and angle of shear resistance. The second one is called unconsolidated quick shear test (ASTM D4767 [50]) and it is used for water-saturated clays and water-saturated soils with flow index more than 0.5 and collapsible soil in water-saturated state for determination of adhesion and angle of shear resistance in the nonstable state.

During **triaxial compression** of the soil, it is used special device that is called stabilometer (or triaxial compression machine). With this test, it can be determined the behavior of the soil in natural conditions and it is possible to obtain accurate mechanical properties such as strength and soil deformation. It can be obtained such characteristics as adhesion, angle of internal friction, undrained shear strength, coefficient of filtration consolidation. It is also can be obtained deformation modulus for water-saturated soils, clays and organic-mineral soils in natural conditions and Poisson’s ratio for any dispersive soil.

For the determination of the most accurate values and determination of particular values, it is carried out at least 3 tests on identical samples with different values of the comprehensive pressure.

During the test, soil sample is put into cylinder shell and placed to the chamber of stabilometer. The lower part of the sample is situated on a porous support. The pressure is transmitted from top to bottom. For providing the pressure to the sample from all sides, the space between sample and chamber is under comprehensive compression by air pressure or liquid (water). By means of this comprehensive compression, it is obtained all necessary characteristics. The device is also able to measure a pore pressure in the upper and lower parts of the sample, volumetric strain of the sample, filtering fluid from sample [37].

Detailed description of the triaxial compression tests and stabilometers can be found in writings of A.W. Bishop and D.J. Henkel (1962) [51] and list of advantages and disadvantages of in-plane shear and triaxial compression tests in work of Brenner et al., 1997 [52].

1.4.4 Investigation of deformation characteristics of coal

Such mechanical characteristics as deformation modulus, Poisson's ratio, ultimate tensile and compression strength are also determined for coal by these standard methods. Also, for coal there are such important characteristics as mechanical strength, brittleness, crushability and others. Grain size distribution, sludge formation and abrasive effect on working surface have a connection with all these characteristics.

The mechanical coal strength is evaluated by crushability, brittleness, hardness, temporary compressive strength and thermal stability.

Crushability of coal is determined by Hardgrove grindability index. According to the ASTM D 409-2016 [48], during this test, air-dry samples of coal with a certain size of particles are ground in the Hardgrove graduated device and after that it is carried out sieve analysis of the obtained material [38]. Grindability index provides guidance on such coal properties as hardness, strength, elasticity, jointing, and depends on the metamorphism stage and the petrological composition of coal.

Coal hardness shows the ability of coal to resist penetration of another more solid object. Coal hardness is usually determined by Rockwell hardness tester, Brinell and Vickers hardness testers. These devices measure coal resistance to crushing under static loads.

Rockwell strength is related to other strength characteristics of substances. This connection was investigated by such scientists as N.N. Davidenkov, M.P. Markovets and others. For example, by the results of an indentation hardness test, the yield strength of a substance can be determined.

The elastic properties of coals are characterized by a temporary modulus of elasticity (Young's modulus), which can be determined by the static method (resistance to compression or bending), as well as the dynamic method by applying mechanical vibrations [39].

Summary

Laboratory investigations are performed to determine the physical and mechanical properties of soils. All the tests are carried out on the undisturbed soil samples with natural moisture state or specially prepared samples with specified density and humidity. It can be concluded by a number of investigations, that moisture content in the rocks have an influence on their compressive and tensile strengths, as well as on the destruction process of rocks under the influence of the load. The shape and dimension of laboratory samples are determined depending on the test method as well as on the soil properties.

1.5 Numerical simulation methods

From the above mentioned, it can be concluded that we estimate rock properties and parameters for individual rock layers according to the results of samples in laboratory conditions, performed according to standard methods as well as based on in-situ researches.

However, the samples that were taken in a certain section of the rock massif have a limited size and cannot display the complexity of the texture and structure of the rock massif. In view of this, for solving such problems of rock mechanics as assessment of stress-strain behavior of the rock massif in the vicinity of mine working and pressure determination on the roadway support, it is required a valid transition from sample properties to the mechanical properties of rock massif.

There are 3 main methods for researching processes in the rock massif as a result of mining such as mathematic simulation, physical simulation (studying of geomechanical processes in laboratories) and field measurements. Mathematic simulation may be carried out based on analytical and numerical methods [4].

Analytical methods are the most favorable for solving the problem of stress distribution around single workings located in homogeneous isotropic rock massif with circular cross-sectional shape. Also, it is simulated the plastic range of stress around working (based on the elastic-plastic deformation model), formation of support pressure load, rock heaving of the seam floor. Such solutions can be obtained in the form of finite dependences of the sought quantities on the initial data [4]. The main advantage of such method is a basis for understanding of geomechanical processes caused by mine formation. The main disadvantage of this method due to which some important specifications of rock massif are lost is idealization of rocks to homogeneous isotropic or anisotropic massif with the simplest geometry of the underground structure [Kononenko, Khomenko 41].

But it is impossible to use analytical methods in the case of a more complex cross-sectional shape of the investigated workings in the presence of an adjacent mined-out space and structural heterogeneities of the rock mass. The solution may be obtained by numerical methods, which allow determining the stress state at certain points of massif using a number of assumptions.

Simulation methods are widely used in various fields, being one of the main components of complex research. Applied to the mechanics of underground structures, these methods allow finding out the main qualitative elements of the mechanism of displacement, deformation and destruction processes of rocks during coal-face work and primary mining. Simulation is a necessary stage during development of new theories as well as during checking solutions obtained by analytical methods.

Rock massif is a very complex structure where during conducting of mining operations, various types of deformation processes such as elastic deformations, elastoplastic displacements and rock failure with fracturing occur simultaneously. In that reason theoretical calculations of rock mass deformations, strength and stability of workings and another underground structures can be hard-solving task. In-situ methods are cost-demanding, time-consuming and have considerable labor intensity. In addition to this, during in-situ tests, the possibilities for varying the system parameters, technology, and sequence of mining operations are usually very limited, while during simulation it is possible to observe the influence of the main parameters in the large-scale ranges.

As a result, simulation method has a number of advantages that are absent in comparison to other methods. However, it is impossible to simulate all the details of simulated objects. So there is a certain simplification of processes and schematization of full-scale bodies. This allows making result's interpretations of in-situ experiments easier, making it possible to control and specify the mechanism of processes with a greater generalization and to study the influence of various factors on these processes.

The biggest benefit of such methods is the possibility to investigate the mechanical properties of a faulted rock massif, as it allows simulating rock jointing and take into account the contacting between rock blocks. In particular, software products based on the usage of the FEM are used quite efficiently, allowing to obtain solutions to many important geotechnical problems.

The stability of workings has a huge impact on the mining enterprises. Due to the continuous development of geomechanical models of the rock massif interaction with the support of the mine, it becomes possible to research the stress-strain state of the rock massif and more fully take into account the physical-mechanical properties of rocks and the constructive distinctions of the elements of the support system.

It was conducted by a number of scientists (Fomychov, Pochepov and Lapko) numerical simulations of scaly massif of Western Donbass and specifying methods of prediction techniques of rock pressure manifestation in the system of "laminated massif-roadway support" considering anisotropy, heterotrophy as well as mechanical, elastic and rheological characteristics of the system [42]. Also, it was applied numerical-based method of finite element (M. Toderas, R. Moraru, C. Danciu) [43] for the analysis of mine workings stability in strongly metamorphosed rock and specification of mine pressure with considering of inhomogeneous stresses effects and directional properties of rocks.

During numerical modelling of zonal structuring of massif [Kononenko, Khomenko, 41] around underground working, it was determined that method of finite element can be widely applicable for researching the zone structuring parameters of the massif.

In addition to this, it was determined [A. Olovyannyy, V. Chantsev, 2018, 44] that using mathematical simulations it can be observed the process of deformations in the rock sample. After getting a result obtained by finite element method, the graphs with stresses and deformations relationship of rocks during deformation or collapsing from laboratory investigations were compared. The received results confirm that mathematical simulations and finite element method can be applicable in tasks of mine pressure.

Rapid improvement of computer technologies gave a huge impulse to the enhancement of numerical simulation methods that determine and investigate strength and deformation properties of rock massif and in particular faulted rock massif. It is an item for a lot of investigations since there is a number of uncertainties that are associated with moving from rock test results to the physical-mechanical parameters of the rock massif and complexity of the fractured massif structure as well as development of numerical methods and their interaction with highly used analytical methods, for solving the problem of determining the degree of size effect on the value of the massif mechanical properties [40].

Numerical methods allow calculating stress fields, deformations and movements that occur in the supporting structures, reinforcing elements and in the inclosing massif. Based on the obtained information it is possible to predict the rock massif behavior, behavior of rocks around workings, their stability assessment, select of the most suitable construction materials for variable structures, to make an argumentation of the most stable structural schemes, that can allow reducing computational costs [45,46].

Summary

It can be concluded that nowadays numerical methods of simulation are enough modern, effective and necessary tool for solving a number of tasks regarding assessment of rock massif stability, mechanical rock characteristics and other geomechanical problems. In addition to this, there is a number of modern software that allow making all the calculations and assessment enough quickly and precisely.

The most widely used and developed method in geomechanical research is the finite element method. It is used in many modern computer programs. Such a numerical simulation method allows us to provide a stable computational process and to form an extensive database of considered parameters of the geomechanical system.

And for the most accurate simulation process, it is possible to simultaneously fulfill conditions such as spatial modeling of objects, reflection of the real structure of the rock massif with a separate description of the mechanical properties of each structural element and the most reliable simulation of all structural features of the support system and material properties.

1.6 Target selection and formulation of research objectives

In such a way, the target of the study is in a substantiation of the support system's parameters, taking into consideration the regularity of changes in the stress-strain state of the surrounding rocks and support elements to ensure the rock massif stability.

And research objectives of the study are carrying out analysis of simulation methods to study the mechanical properties of the rock massif, to design and give an argumentation models of simulation experiment on the basis of chosen simulation method, make an analysis of the received stress-strain states and to substantiate the rational parameters of the support system.

PART 2

THE CHOICE OF THE METHOD OF SLIGHTLY METAMORPHOSED ROCKS RESEARCHING AND THE CONSTRUCTION OF THE RESEARCH MODEL

2.1 Key features of the Finite element technique and program selection for investigation

At first simulation models have been used long time ago for describing simulation appearances in Math, Mechanics, Physics and other exact sciences. Such great scientists as Newton, Euler, Gausse, Chebyshev were developing numerical methods. Newton proposed the efficient numerical method for solving algebraic equations, and Euler - the numerical method for solving ordinary differential equations. With the advent of electronic computing machine, the speed of computing operation has largely increased. This allowed to solve a wide range of mathematical tasks that before were unsolvable and to use wider numerical simulations during scientific and theoretical calculations [53,54].

Differential numerical methods are the basis for a lot of modern simulation experiments, that are applied in such areas as structural mechanics, geomechanics, elasticity theory and others. Main advantage of simulation method in comparison to the in-situ method is the opportunity to accumulate the results obtained by studying of tasks and after that to apply them quickly in completely other fields [54].

The first method among numerical methods that use mesh is **method of finite difference** that is based on replacing ordinary and partial derivatives included in the differential equations and relations describing a particular physical problem, by their approximate expressions, in which the differentials of the arguments and functions are replaced by finite increments.

Comparing this method to the finite element method, there is a number of advantages of FEM such as more stability of the method, ability to work with geometrically more complex areas, the solution represents a function and values at any point can be calculated immediately, however it is faster to build a difference scheme for simple task in FDM.

In these days there are such the most popular methods of numerical simulations as discrete elements method (DEM), boundary element method (BEM) and finite element method (FEM) [4].

The field of **DEM** application are soils, foundations, rock massifs with a complex structure, mechanical systems, non-linear dynamic processes for which this method is the most effective and many others. The point of this method applied to the simulation of rock massif with complex structure that contain workings is in replacing of real rock massif on discrete elements, the shape of which should preferably be similar to the outlines of the elements of the massif structure. Due to the simplicity of creating a mathematical model of a discrete medium, the model elements are represented in the form of balls, which have different diameters in general case.

In the result of simulations based on DEM method, it is obtained a stress-strain state of the rock massif with determination of main stresses, displacements and deformations, caused by a cavity formation of a given size in a continuous medium.

In comparison to BEM and FEM that can be applied for simulation of a continuous medium, the distinctive feature of DEM has an ability to describe non-linear dynamic processes that occur in a deforming rock massif that contains cavities and sections of destroyed rocks. However, for processing of large data files, it is necessary to have powerful computers and a large amount of operating storage [4,55].

Regarding **boundary element method**, unknown functions of the integral equations are real variables that have physical meaning. For example, in tasks of the theory of elasticity, such a solution of the integral equation should immediately give all the forces and displacements at the boundary, but inside the body they should be obtained from the boundary values by numerical integration. This method is based on the fact, that it is easy enough to obtain an analytical solution corresponding to the point disturbance in the continuous homogeneous medium. This disturbance, for example, can be represented like concentrated force in an elastic body.

The BEM method is more cost-effective than FEM. The system of equations has a degree of order much smaller, since it is formed only for a certain number of elements defined on the boundaries of workings. However, the BEM has fewer possibilities for simulations of various medium heterogeneities and nonlinearity of physical relations [56,4].

The main idea of the **FEM** is that any continuous quantity (for example pressure or temperature) can be approximated by a discrete model, which is built on a set of sectionally continuous functions defined on a finite number of subdomains. This method is based on division of the area on elements of simple geometric form. It is also efficient to use rectangular and triangular elements for calculations. Finite elements interact with each other in nodes through nodal forces and nodal displacements. In nodes the conditions of equilibrium and displacement continuities are completely satisfied. The basic concept is in fixing of internal points displacements of each element with nodes displacements. This relationship is expressed by shape function. This shape should satisfy boundary conditions on the element outline and continuity condition of deformations [4, 57].

This method has a great potential for mathematical simulations of various heterogeneities. For each element, elastic properties can be set and stratification can be easily simulated. It can be simulated area of decayed rock by setting corresponding elastic modulus values or consider the presence of more rigid elements. Many solutions to various geomechanical tasks have been obtained on the basis of FEM, not only in the elastic approach and not only for plane region. Three-dimensional area also can be divided into subareas and three-dimensional area. But in that case the relationship between the movements of the element’s internal point and the movements of nodes will have a more complex shape and stiffness matrix of the system will have higher dimension [57, 4].

Software systems based on the FEM are divided into 2 main groups. The first one are finite element analysis programs introduced to the menu in computer-aided design (CAD) programs and they have all necessary tools for quick calculations of elements or assembly components in the environment of their development. The second group are programs focused on the preparation of full-scale finite-element model of the researching object with maximum possibilities of simulation, taking into account all features and realization of different types of calculations [58].

Nowadays there is a number of modern computer programs based on the FEM that cover almost all areas of engineering calculations, analysis and simulation of processes. For example, strength, vibrations, stability, dynamics, acoustics, hydrodynamics, aerodynamics, etc. There are such famous programs as ANSYS, AutoCAD, NX Nastran, Abaqus, FLAC3D and SolidWorks.

One of the best FAE-programs is **ANSYS**. The biggest advantage of the system is the most complete documentation and guiding system, which allows to use this program without customer support having good basic knowledge, for example in mechanics. There are also more than 100 finite elements in the program. It is a universal software system of FEM analysis, that is popular enough for automated calculations, for example computer-aided design (CAD) or computer-aided engineering (CAE) for solving both linear and nonlinear, stationary and non-stationary spatial tasks of mechanics. Program developers constantly improve it and make it easier to use, to customize and managing all the complexities [59].

The next widespread program based on FEM and originally developed by NASA is **Nastran**. Now the main NX calculation modules such as NX Nastran, NX Thermal and NX Flow are produced by Siemens PLM Software. In 2016 the company presented Simcenter product portfolio. It is powerful high-performance set of systems that combines a solution for simulation and in-situ testing with intelligent reporting and data analysis tools. This allows developing models that predict with high accuracy the characteristics of the future product at all stages of production preparation. The NX Nastran provides a number of engineering calculations including stress-stain state determination, proper frequencies and vibration mode, dynamic linear and nonlinear tasks of engineering analysis [60].

There is also **Abaqus** FEA. It is a set of programs for finite element analysis and CAD. Abaqus has 5 major software products such as Abaqus CAE, Standard, Explicit, CFD, Electromagnetic. Abaqus was originally developed for solving nonlinear physical behavior and realization of complex models taking into account all the forms of nonlinearities, and also to conduct multitasks static and dynamic analysis. This program is popular among both research and academic institutes. It is possible to apply the Abaqus on all steps of designing the product all the calculated, design and technological services of the company. This program is also reliable enough, with special control over the error of the results of computed tasks, auto-selection of integration gaps, as well as many other functions. In contradistinction to ANSYS that does not have a model of poroelasticity in standard tools, Abaqus allows solving tasks of mechanics of porous media directly [61].

There is such widespread program as **AutoCAD** developed by Autodesk company. It is two- and three-dimensional computer-aided design and drafting system that is highly applicable in mechanical engineering, construction, architecture and other industries. AutoCAD is certified to work in both Windows and OS X operation systems. The program also has an effective data exchange and documentation preparation tools allow carrying out all stages of the project - from concept development to the final stage. There are convenient interface and realistic visualization that create visually-filled images. The main disadvantage of the program is the difficulty of linking information from the database to graphic objects. Nevertheless, it should be noted that the absence of three-dimensional parameterization does not allow AutoCAD to compete with such program as **Autodesk Inventor**, that is three-dimensional system of solid-state and surface parametric design. The program tools also provide full range of design documentation creation.

One more program for simulation is **FLAC3D**. It is numerical modelling program for investigation of rocks, constructions and supports. This software also analyses integrated behavior of the model, shows strains and deformations. This program designed to the solution of geotechnical tasks and is highly used in civil engineering and mining. It also deals with the simulation of faults, joints, intact rock and planes of stratification, making it especially effective for usage in simulations [66].

SolidWorks is a program that based on the technology of three-dimensional parametric simulation, it is a simulation with usage of the model elements parameters, changing which, it is possible to view various structural schemes. There is a standard

Windows graphical interface and interaction with Windows applications. There are also more convenient computational and analytical modules, for example, a stress analysis module. SolidWorks was the first CAD system to support solid-body simulations for the Windows platform. In SolidWorks, a single sketch can be used for multiple operations. It can be selected both the outline of the sketch and the edges of the models and apply forming operations to them [62].

The design system SolidWorks covers the entire product development and helps to solve complex problems in the design and simulation. In this investigation, it was chosen to use the SolidWorks program due to its proper interface, accuracy of results and the possibility of a solid body simulation.

In this work, it is chosen program SolidWorks. This program has a number of opportunities and useful tools for usage in the mining industry. Among them, there such possibilities as consideration of:

- the entire thickness of the rock massif;
- geometric nonlinearity;
- static load;
- stratification of the rock massif;
- isotropy of the deformation and strength properties of rocks;
- the user choses the size of the mesh by himself (fine or coarse) and it is possible to set the different mesh for each component of the simulation model;
- setting the physical parameters of the working support elements;
- spatial arrangement of processes that occur in the rock massif;
- the ability to simulate quickly a large number of alternative design options, for instance support systems;
- based on the initial conditions, stresses in the structure, deformations, shears and strength characteristics are calculated.

All these program’s features make the choice for its usage more suitable and convenient.

2.2 Argumentation of the simulation method selection

There is a wide variety of studies of the mechanical characteristics of rock massif. The main groups for the study of properties, as it was mentioned earlier, are field, laboratory and numerical methods.

However, the numerical methods have a number of advantages. By the simulation of the rock massif and mining-and-geological conditions in the vicinity of the working with the help of substitutions, it is also possible to research the deformation process in the massif and mutual influence of all the parts and elements of the simulation model of the investigated geomechanical system.

The numerical simulation allows to take into consideration a lot of parameters. For instance, different structures of rock seams which have various geometric sizes and mechanical characteristics. All rock layers can have different degree of fracturing, water saturation and rheology.

All the structural models consist of 3 main groups of characteristics, that should be taken into consideration during simulation. The first group is geometric features (flat or spacing solution, multiply connected regions or simply connected regions). The second group is external loads and mechanical properties of the rock massif (isotropic or anisotropic substance, elastic or inelastic deformation, with or without consideration to the rocks weakening). And the third group is constructional and technological aspects of the working development, design and operation, extraction of minerals, design of workings [67]

It is quite necessary to design the simulation model which will be capable to represent the most adequate and real configuration of the rock massif, the working construction and the working support. Due to the numerical simulation, in particular the finite element method, it can be taken into account the huge variety of parameters and characteristics [68,69].

In this investigation, the research of the slightly metamorphosed rock massif is based on the following structure:

1. Development of the simulation model.

2. The simulation model design with different mechanical characteristics for each element.
3. Sequential experimentation of the simulation models.
4. Background of the obtained diagrams of stress-strain state of the simulation model elements.
5. Signification of the important elements of the simulation results.
6. Analysis of the obtained stress-strain state of the rock massif and elements of the working's support.

During simulations, it was applied spatial (three-dimensional) model. Such model has a number of advantages in comparison to planar model. There are such advantages as consideration of heterogeneity of the rock massif in 2 vertical normal planes, fairly presents contact relations between the rock massif and mine working support, allows representing all the possible physical characteristics of the real rock layers. In such a way, the spatial model minimizes the possibility of mistakes during simulation.

The construction of the simulation model starts from the rock massif construction around the working. All the rock seams are designed in the three-dimensional model; in that reason it is quite important to apply spatial model of the simulation.

Also, one of the disadvantages is investigation not only border zone of the rocks, but also entire stress field around the working and analyzing of the stress state of the roof, bottom and walls of mine.

2.3 Selection of the initial and boundary conditions in relation to the mining and geological conditions

In this investigation it was chosen geological settings of "Dniprovska" mine that is located in the Western Donbass of Ukraine. According to the large-scale studies, coal deposits of the Donbass are mainly represented by terrigenous rocks. These sedimentary rocks consist mainly of rock fragments and minerals of silicate composition that arising from the transfer and accumulation of land denudation products. There are such widespread rocks as sandstone, siltstone, and mudstone in the

Donbass. Coal seams and limestone layers are enclosed in the form of rock layers among them.

Support of workings in the Western Donbass is carried out by a combination of different types of support systems and the parameters of the elements vary depending on the manifestations of rock pressure. The experience of stope's support in the mines of the Western Donbass confirms the feasibility of using "frame support-rock bolt" system.

In addition to this, such support system of rock prevents the specific phenomenon to the region of Western Donbass, during which it is occurred the stamping of easily deformable rocks of the mine walls and bottom due to the increased stiffness of the coal seams. Also, in view of the extensive stratification zones of the soft rocks of the mine roof, the feasibility of the massif support with rope rock bolts is explained.

The longwall № 543 of seams C_5 and C_5^u was prepared on the eastern wing of mine-take of the inclined drift. The length of the longwall is 182 meters. Extraction pillar of the working area was prepared by main roadway № 543 and extraction pillar № 543, that were constructed in 2017 - 2018 years. The depth of mining is in the range of 120-210 meters.

Boundary entry № 543 in over the range of extraction pillar is conducted on coal with coal-cutting of rock walls by frame-anchorage lining. The working is fixed with a metal arch three-link marquee pliable uniform support (MPUS-11.7). According to the mine specifications, the setting increment of the support is 0.8 – 1.0 meter. Anchor bolts with a length of 2400 mm were installed in the spaces between the frames of the arch support. It was taken 4 anchors bolts in a row with stepping increment of 0.8 meter of installing roof supports and 6 anchors bolts in a row with step increment of 1.0 meter of installing roof supports. During mining of the working area of the 543 longwall face, it is needed to maintain the boundary entry during the entire period of its mining for providing a direct-flow ventilation scheme. Boundary entry № 543 is designed to deliver materials and equipment to the longwall. Mining sequence is long-pillar. Extracting coal seam thickness is 1.05 meters. Coal seam angle along the longwall is 1˚ - 2˚, in the direction of mining 2˚-6˚.

For mining simulations, it was necessary to establish sizes of the simulation model and determine its height and width. According to the number of scientific researches, it was revealed that perturbations of the stress-strain state of the rock massif extend to a depth (height) of 15-20 meters and outside the zone of influence of coal-face works, the stability of the in-seam working is determined by the state of the adjacent rocks of seam roof and bottom. According to these researches and previously conducted experiments, for construction of the model in the appropriate ranges and obtaining the reliable assessment, it is accepted the model of rock massif with height and depth of 20 meters each, the width is also 20 meters, the thickness of the model is 5 meters. Taking into account these quantities, it was taken 11 rock layers upper the coal seam and 7 layers below. There are such types of rock that form these seams as sandstone, siltstone, mudstone and coal.

Depending on the various characteristics such as bedding conditions of the rock, type of mine rock, grain texture, rock structure, there are different mechanical characteristics of rocks that were used during simulation. In the Table 4 there are all mechanical characteristics of the rock massif, that were used in the study.

There are such characteristics as thickness of the layers, compression and tensile stresses, elastic modulus, Poisson’s ratio, shear modulus, rock mass density.

According to the characteristics of the rock massif, it can be seen that thickness of the layers is in the range of 0.2 to 7 meters. Compressive stress is in the range of 18 MPa to 27 MPa in the unrestrained (natural) condition without water saturation, the tensile stress is in the range of 1.8 MPa to 5 MPa, the Poisson’s ratio is from 0.21 to 0.26 and the rock density is from 1240 kg/m^3 to 2600 kg/m^3.

Rock seams around the working are consist mainly of mudstone and siltstone, which are soft rocks, prone to slaking and have expressed rheological properties. With the presence of sandstone, that is a stronger rock, higher strength and deformation characteristics are appeared. Also, with moving away from the coal seam, the mechanical characteristics of mudstone increase.

Number of layer	Type of rock	Layer thickness (m)	σ_{compr} (MPa)	σ_{ten} (MPa)	ν	E (MPa)	G (MPa)	P (kg/m^3)
1	Mudstone	0.8	23.6	2.3	0.21	5000	6000	2370
2	Sandstone	1.45	26.5	5	0.26	12000	9500	2600
3	Mudstone	2	20	3	0.21	7000	6000	2370
4	Siltstone	1.7	18	1.8	0.25	6000	6000	2510
5	Coal	0.36	27	2.7	0.25	3.6000	1500	1240
6	Siltstone	0.6	20.2	2.2	0.25	4000	6000	2510
7	Sandstone	3.2	26.5	4	0.26	10000	9500	2600
8	Mudstone	1.7	24	2.4	0.21	7000	6000	2370
9	Siltstone	3.4	20	2.0	0.25	7000	6000	2510
10	Mudstone	2.9	23.5	2.5	0.21	8000	6000	2370
11	Mudstone	2.5	22.6	3	0.21	8000	6000	2370
12	Coal	1.05	27	2.5	0.25	37000	1500	1240
13	Mudstone	2.25	24.5	3	0.21	8000	6000	2370
14	Siltstone	2.55	19	2	0.25	5000	6000	2510
15	Coal	0.27	27	2.5	0.25	37000	1500	1240
16	Siltstone	3.4	20.2	2.5	0.25	4000	6000	2510
17	Mudstone	7.0	24	2.4	0.25	9000	6000	2370
18	Coal	0.2	27	2.5	0.25	36000	1500	1240
19	Siltstone	5.8	20.2	2.2	0.25	6000	6000	2370

Table 4. Mechanical characteristics of the rock massif

In addition to this, such weakening factors as water-bearing and rheology also act less intensely. However, it should be taken into account that the rocks of the Western Donbass consist mainly from the slightly metamorphosed rocks. In addition to this, the connection of lithological variety of coal formation is weak due to the mining operations that perturb the initial field of the stress-strain state of the rock massif, which leads to the deformation of rock seams in both vertical and horizontal directions and intensifies the manifestations of rock pressure [40, 63, 64].

The seams of the working have an angle of dip 1˚, the depth of the working is 200 meters.

The following simulation algorithm was chosen. At the first stage, it was performed the calculation of the stress-strain state of the base model working without support and anchors bolts. At the second stage, the stress-strain state of the rock massif with the support is calculated. In the third stage, simulation was performed both with 6 rock bolts and frame support. All tasks are considered in an elastic approach.

It is determined the components of the stress field (both horizontal and vertical stresses) and value of stresses intensity. The investigated geomechanical system is divided for basic components such as rock massif, frame support and rock bolts. Each of these elements is analyzed by three components: horizontal stress, vertical stress and intensity. The requirement of such analysis is in the mechanical, physical and geometric differences of all the elements of the system. Each component of a stress has an information about condition features of various elements, for instance rocks of bottom and roof of the working, walls of the working, roof timber and legs of the support, rock bolts in the roof and walls of the massif. Each component of stresses gives the basis for a reliable estimation of the rock massif.

The difficulty of simulation the working support and rock bolts are in reflecting the real design features of these elements and taking into account the fact that the linear dimensions of the both working support and rock bolts are much smaller than the average sizes of the elements of the rock massif. During the simulation, it is necessary to design finer mesh, that increases the time for simulation.

2.4 Sequence of the model development

The design of the simulation model should be started from the researching of the geological conditions of the mine and rock massif structure. As it was described in the Table 4, there are all the necessary parameters of the rock massif. For the beginning, it is constructed rock seams with setting the geometric parameters and mechanical characteristics of each layer. The rock massif in this investigation for the simulation model is formed from 19 layers.

It was such a sequence of operations as: new – create - part (a 3D representation of a single design component) - sketch, and creation of each layer with setting the height, width, length and angle of inclination. The width of the model is 5 meters, so each seam was elongated for 5000 mm (Figure 2).

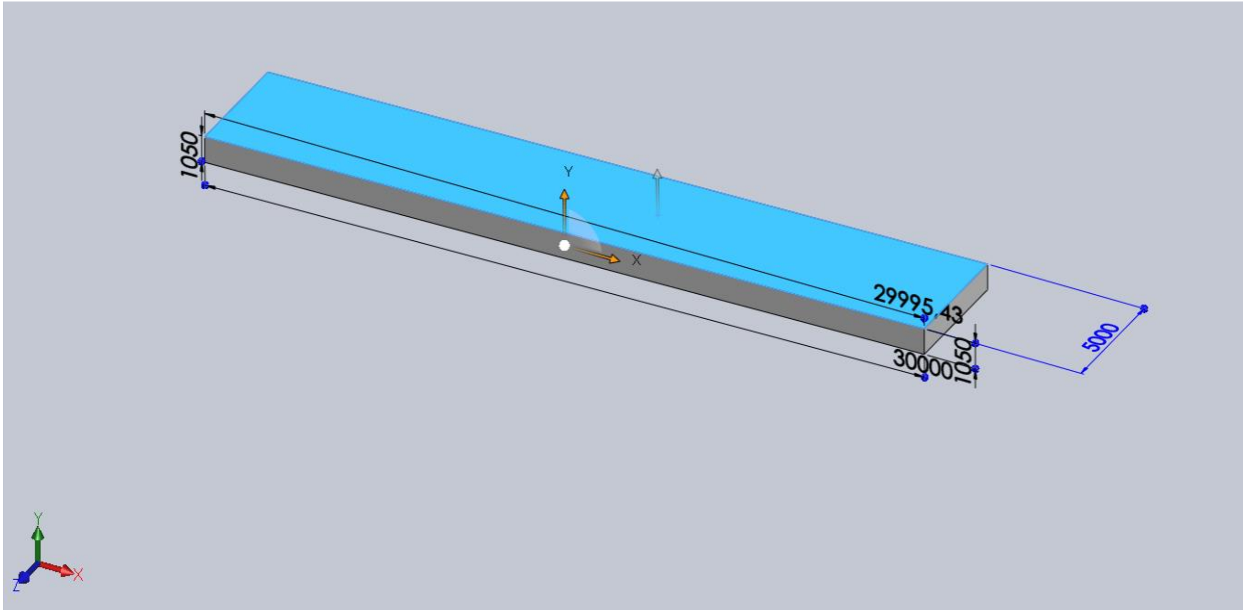

Figure 2. Coal seam in the “SolidWorks” with its dimensions

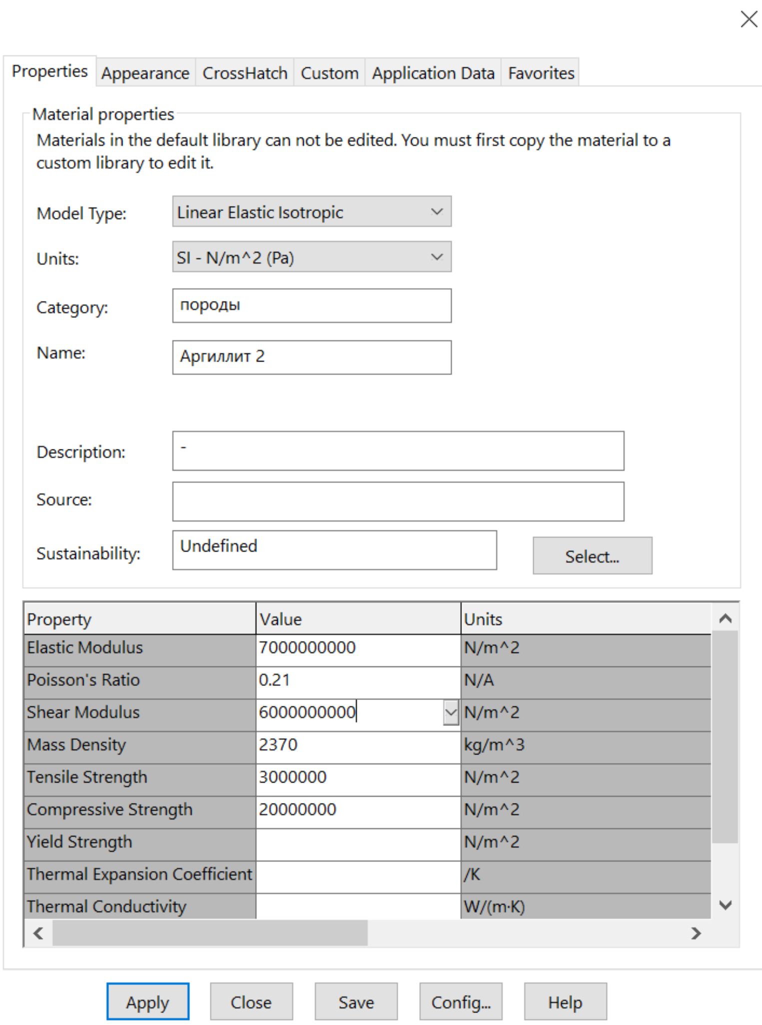

Figure 3. Setting of the material properties for coal seam

After the seam development, by the instrumentality of smart dimension function, the sketch was determined. After that in the "material specifying", all the mechanical characteristics of each seam were determined. Each seam has own parameters, but some of them were taken the same for the same seams, for example mass density or shear modulus for each type of rocks (Figure 3).

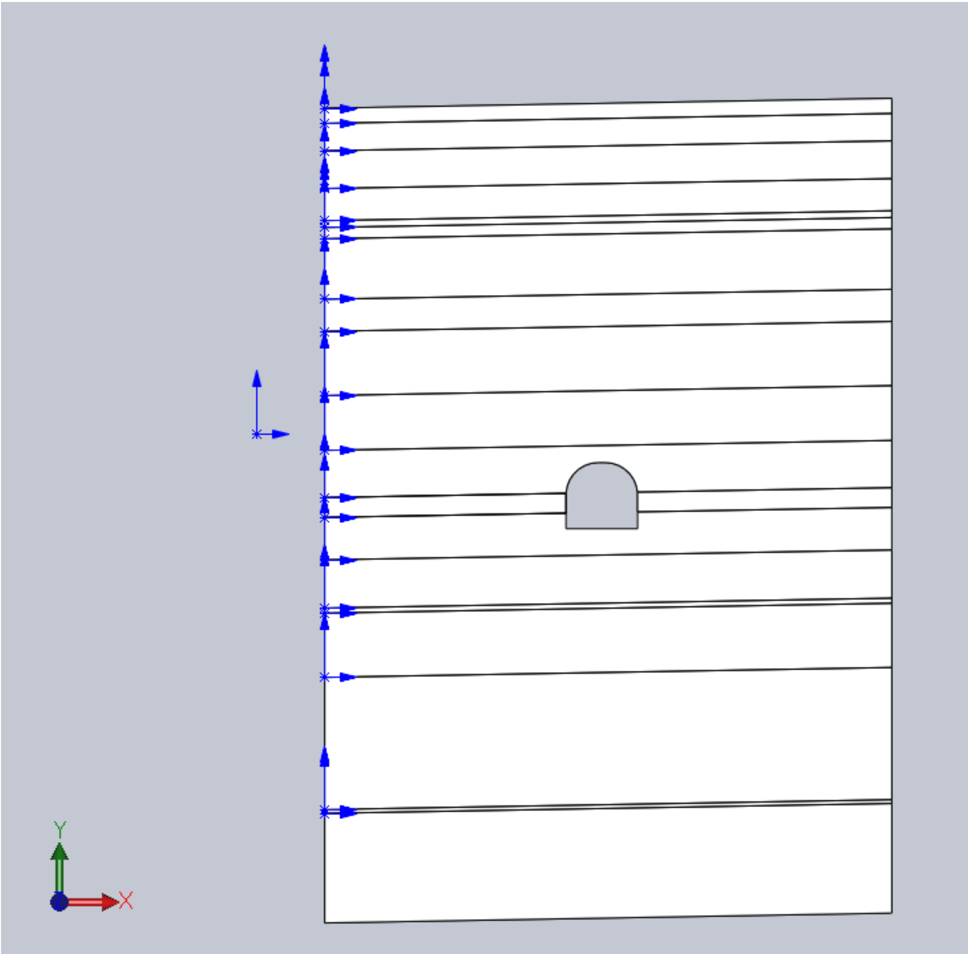

Figure 4. The assembly of rock massif with connected seams between each other

After the creation of all seams, it is needed to create a new file – assembly, where it is inserted all 19 rock seams that we mate with each other by "coincident" function (Figure 4). After the combination of all seams and creation of the rock massif, it is created a cutting of the working in the massif, that was done by means of the frame support with the backfilling.

For the creation of the frame support and cutting out in the working, it was simulated a frame support according to the dimensions, stated in the specification from the Dniprovska mine. The taken width of the frame is 3500 mm and the height 3360 mm. Substitutable special section (SSC) of the frame support– 22. (Figure 5).

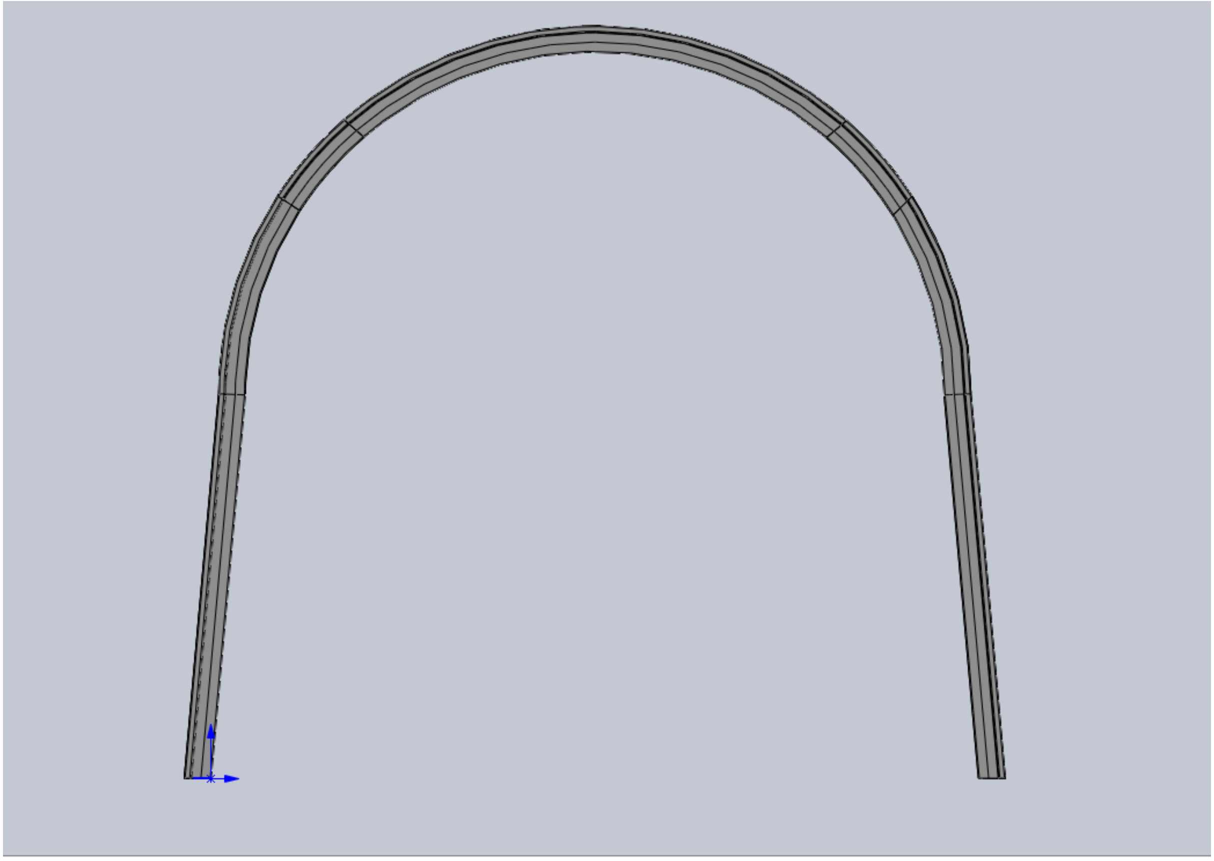

Figure 5. The frame support

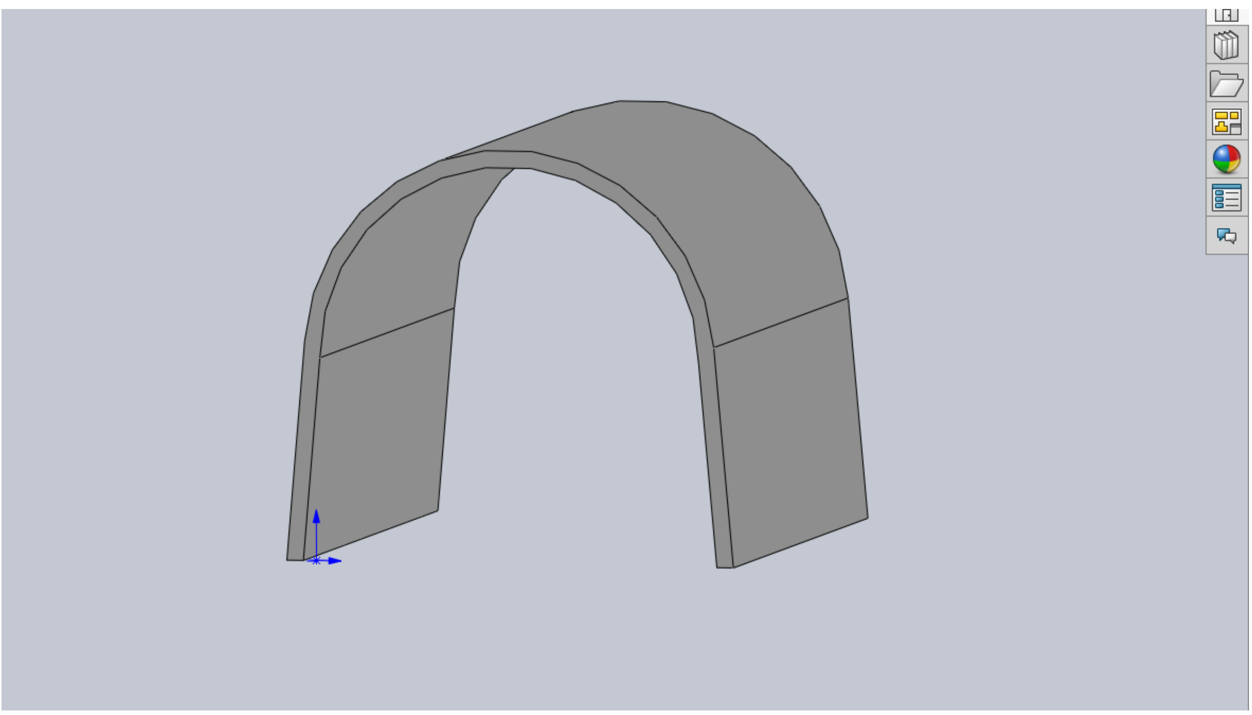

Figure 6. Backfilling

For maximum approximation to real mine conditions, a construction of the backfilling space between the frame support and the rock massif was constructed. It is applied for the increasing of stability of the rock massif and more homogeneous distribution of stresses. The length of the backfilling corresponds to the length of the model and is equal to 5 meters (Figure 6).

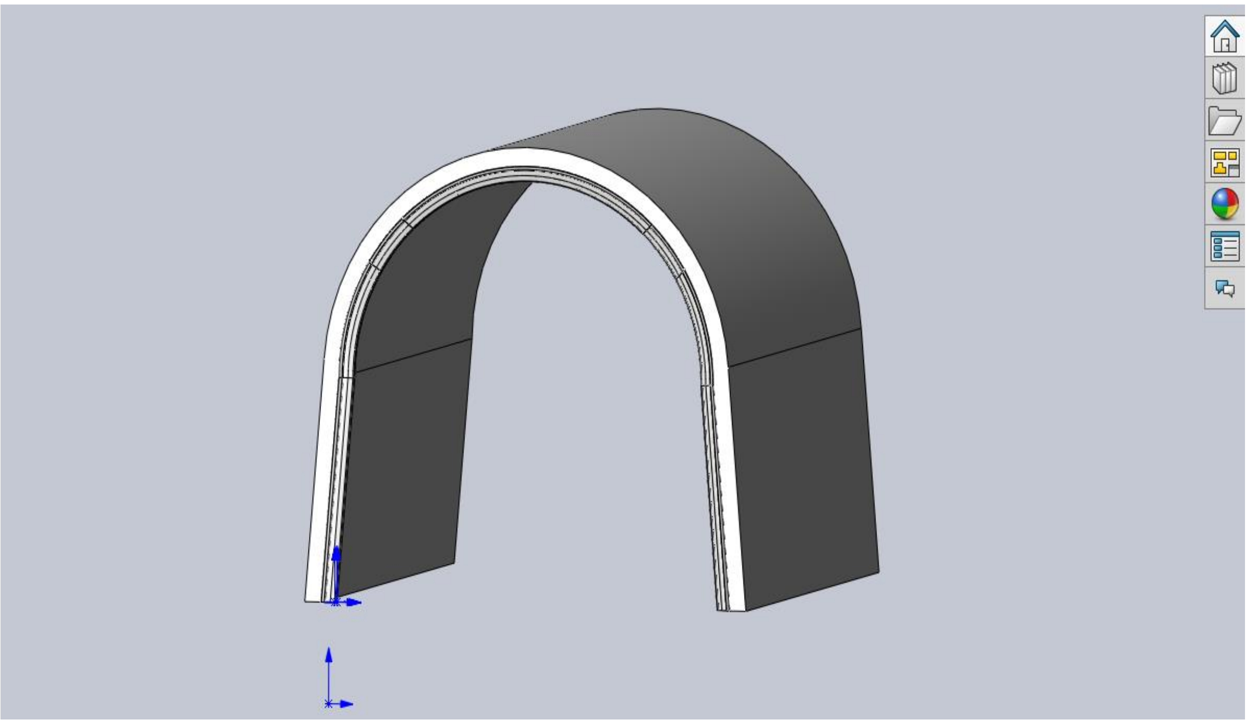

Figure 7. Backfilling in assembly with the support

For the creation of the cutting out in the rock massif and conduction 2 simulations (with the working support and rock bolts), it was created the new assembly, where the frame support was mated to the backfilling. After that, according to the outline of the sketches, it was made the cutting out (Figure 7).

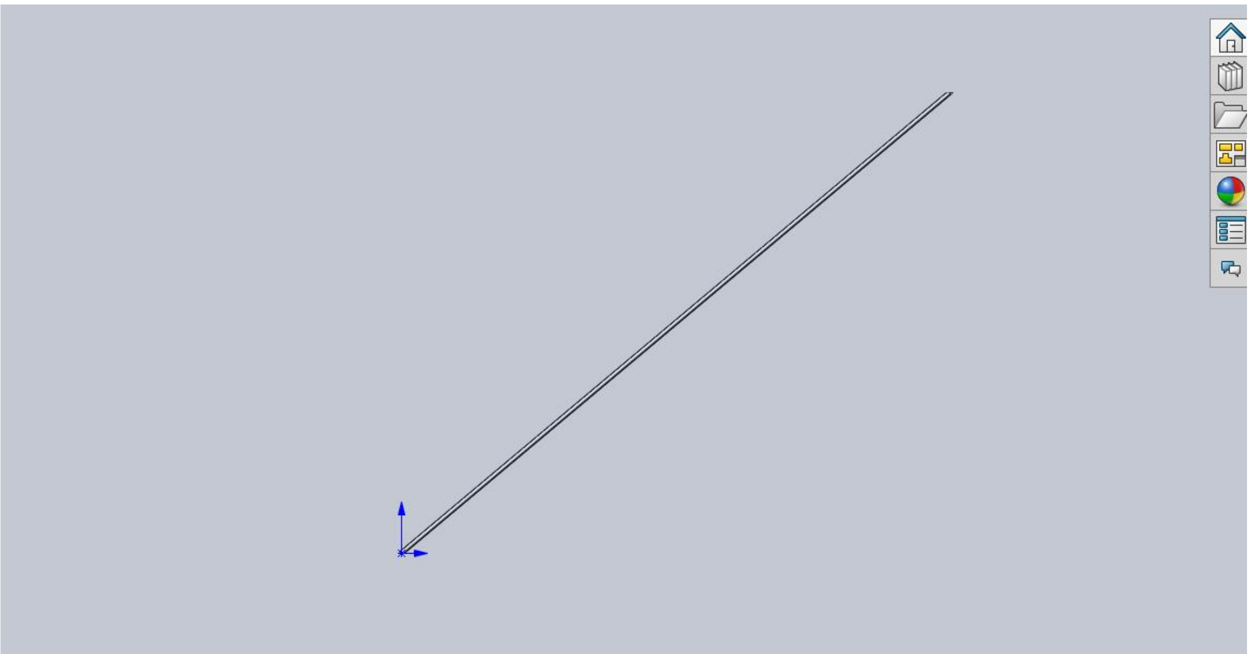

Figure 8. Rock bolt

For the simulation of the rock massif with the frame support and rock bolts, there were designed models of 6 rock bolts. Among them 2 central rock bolts are rope bolts, and 4 in the lateral sides of the roof are resin-grouted rock bolts, 2 on each side (Figure 8).

After both assemblies are prepared (the rock massif assembly and the assembly of backfilling with the support) and all the characteristics of seams are set and all the seams are interconnected, it was chosen function “SOLIDWORKS Add-Ins” and SOLIDWORKS Simulation. For the simulation process it is created a new research “static 1”. (Figure 9)

For the beginning, it was created a new research “static 1”, where the mesh for the rock massif was built. After that in the insert “fixtures”, it was made a fixed geometry for the rock massif and lastly it was set in the “external loads” the pressure of 5 MPa (Figures 9, 10).

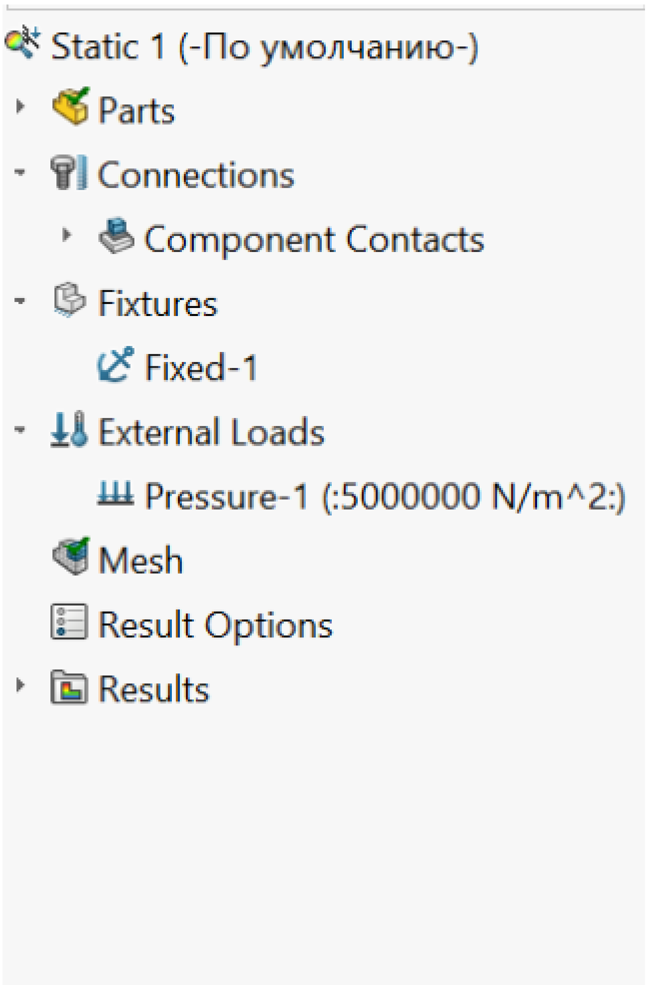

Figure 9. Components of the Static 1.

During the second and third simulations, where there are rock bolts and frame support, more fine mesh was created separately for these elements around the working, and for adjacent rocks around the working for more detailed and precise simulation results, including coal seam.

Also, it was set the mechanical characteristics of the frame support and rock bolts according to the chosen substitutable special section and grade of steel (steel 5). And, it should be taken into consideration, that were set different characteristics for each type of rock bolts and frame support. For the roof timber and legs characteristics are the same, but for frame locks they differ.

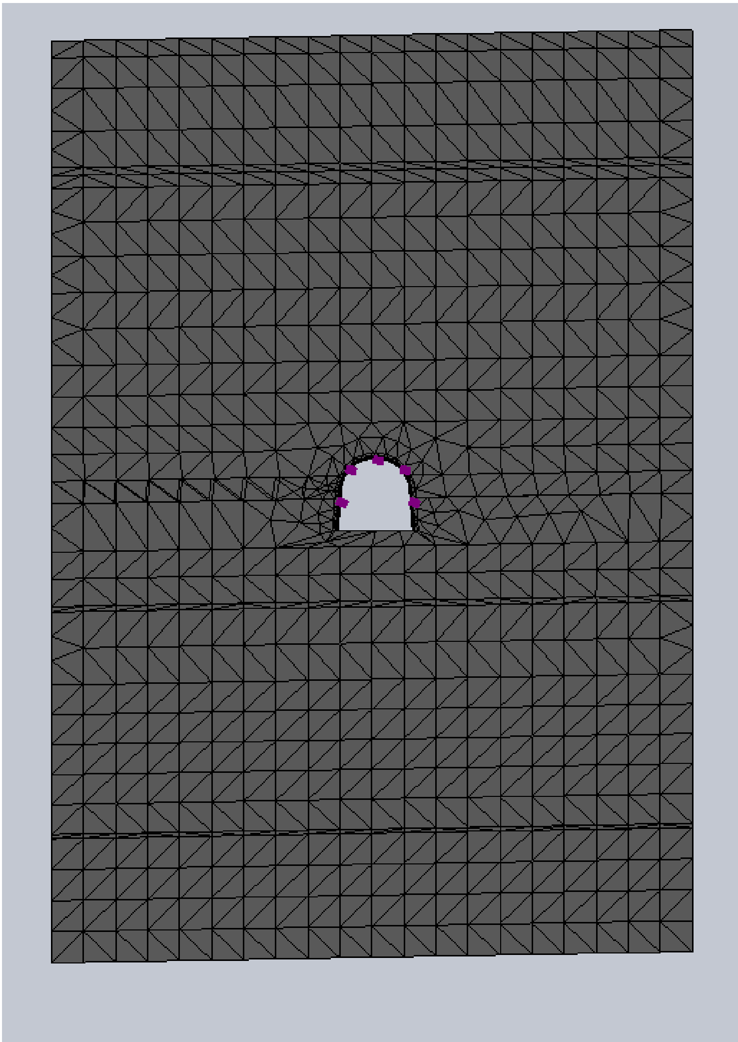

Figure 10. The mesh

After that, the bottom “run this study” is pressed and the simulation process is started.

2.5 Strain-stress state analysis

The first simulation model of the working is a driven working in the rock massif. It was applied a load of 5 MPa, calculated according to the formula:

$$\sigma_y = \gamma * H,$$

where γ is a bulk density of rocks (in average 2500 kg/m^3), H – is a depth of working (200 m) deduct the upper part of rock massif, that that is researched (20 m).

Strain-stress state analysis of the rock massif upon the action of horizontal and vertical stresses depends on the Poisson's ratio of each rock seam, that were installed in the program SolidWorks during the setting of the number of parameters and mechanical characteristics of the seams.

One of the most important characteristics that determine the stresses distribution in the rock massif is the lateral pressure coefficient. During the changing of the load, the coefficient values also change. With the increasing of the load, the coefficient also increases and only in a certain load interval remains constant. We can count the lateral pressure coefficient using Dinnik's formula for elastic rock model:

$$\lambda_{el} = \frac{\nu}{1-\nu}$$

where ν is a Poisson's ratio of the needed seam.

The vertical component (vertical intensity σ_y) at a given depth is determined by the weighted average density of overlying rock seams, and the horizontal component (horizontal intensity σ_x) is proportional to the geostatic pressure and is found by the formula [65]:

$$\sigma_x = \lambda_{el} * \gamma * H$$

2.5.1 Strain-stress state analysis of the working in terms of vertical stresses

Stress-strain analysis of the working considering the vertical intensities σ_y shows that in the bottom of the working the area of lower compression stresses changes over tension stresses closer to the outline of the working bottom. And this area is much bigger than in the roof of the working. Such stress distribution is characteristic for the conditions of the Western Donbass, since the manifestations of rock heaving are usually more intense than the roof subsidence. It is obtained tension stresses signed "+" and compression stresses signed as "-". Tension stresses create weakening area in the mine bottom because the rocks weakly resist tensile forces, taking into account softening factors such as the moisture content, fracturing, and rheology. For the vertical stresses, it was set such range of -9 MPa for compression stresses to +2 MPa for tension stresses.

In every simulation model, it is chosen the most suitable range of values that modifies and selects for the most appropriate reflection of the stress state.

In the bottom of the mine, the stresses act in the area with width around 3 meters and 2 meter in depth. The stresses act from -1 MPa of compression stresses which go into tension stresses +1 MPa closer to the outline of the mine bottom. Due to such stresses, it is appeared the rock heaving in the bottom.

In the roof of the working, in the central part of the seam, there is tension stress that bending the rock into the working, and in the higher part of the seam there is compression stress. The area of these stresses action is approximately 1-meter width and 1-meter height that is less than in the mine bottom. The tension stresses act in the range of -0.75 MPa to +1.5 MPa.

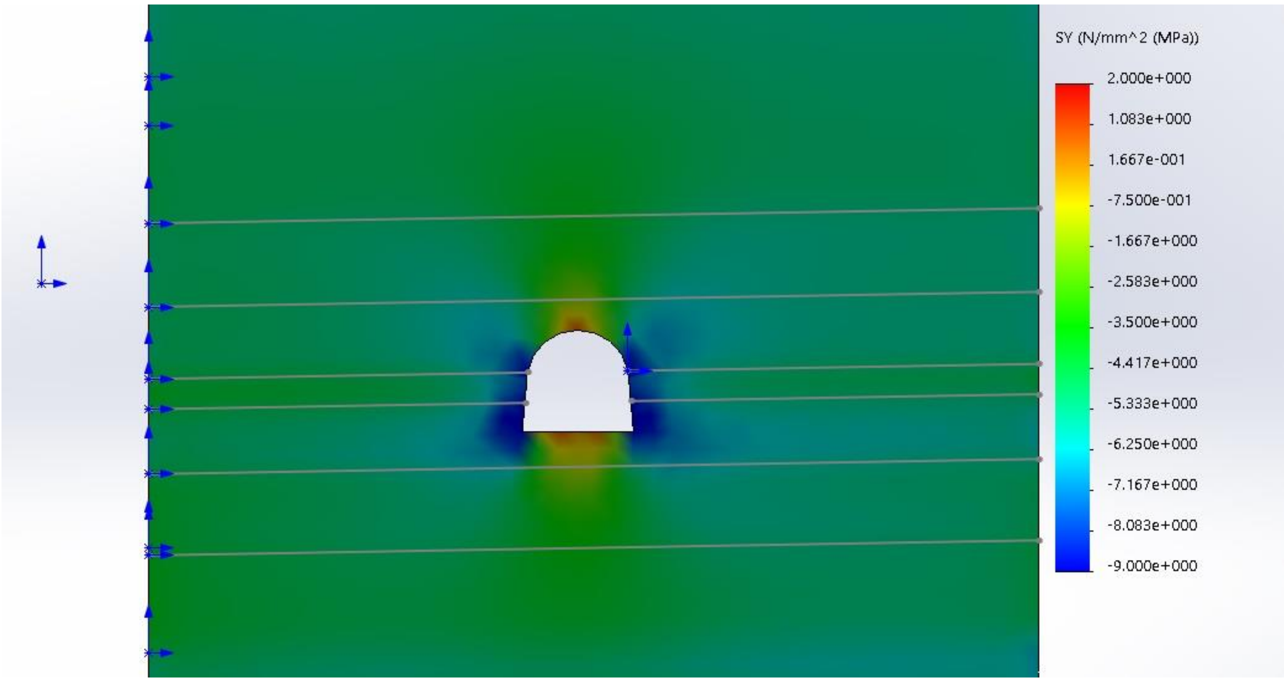

Figure 11. Vertical stresses of the working

In the walls of the workings acts bearing pressure and there are areas of the increased rock pressure. The areas of the compression stresses in the walls of the working has an area of the 5 meters' height and 2.5-meter width. The range of compression stresses is from -6 MPa and to more than -9 MPa near the mine walls. The area of compression stresses has large sizes and engages coal seam, its roof and bottom. Moving away from the mine walls, it is appeared a zone in unloaded condition.

According to the quality factor, of the horizontal stresses corresponds to the existing ideas about the distribution of geostatic stresses around the working. In the roof and bottom of the mine, the areas of tensions (σ_y) are formed, and in the walls of the working – zones of high rock pressure.

2.5.2 Strain-stress state analysis of the working in terms of horizontal stresses

It was set such range of load values from +3 MPa for tension stresses to -6 MPa for compression stresses in terms of horizontal stresses. Considering the simulation results of the working, in particular the horizontal intensity, it can be seen that in the walls of the working acts such compression pressure within the limits of -2.25 – 0.75 MPa.

Viewing the upper and lower layers of the coal seam, which are represented by the layers of mudstone with the Poisson's ratio equals 0.21, it is also can be concluded that in the walls of the working act compressive strength. Taken for example the mudstone layers upper and lower the coal seam and the calculated horizontal pressure according to the formula, we will get such lateral load of the working for this layer as -1.35 MPa.

Analyzing immediate mining roof that is represented by mudstone, it can be observed tension stresses. In the central part of the rock arch, the tension stresses act in the range of 2.0 to 2.5 MPa. The destruction of the continuity of the immediate roof may occur in the area, which has approximate dimensions of 1.8 meters wide and about a meter high. Considering the sides of the immediate roof, tensile stresses in the range of +0.75 to +1.5 MPa act there. Taking into account fracturing rocks of the Western

Donbass, that cannot resist tension, it is creating the weakening zone, that have approximate dimensions 3.5 meters high and 2 meters wide from both sides of the workings.

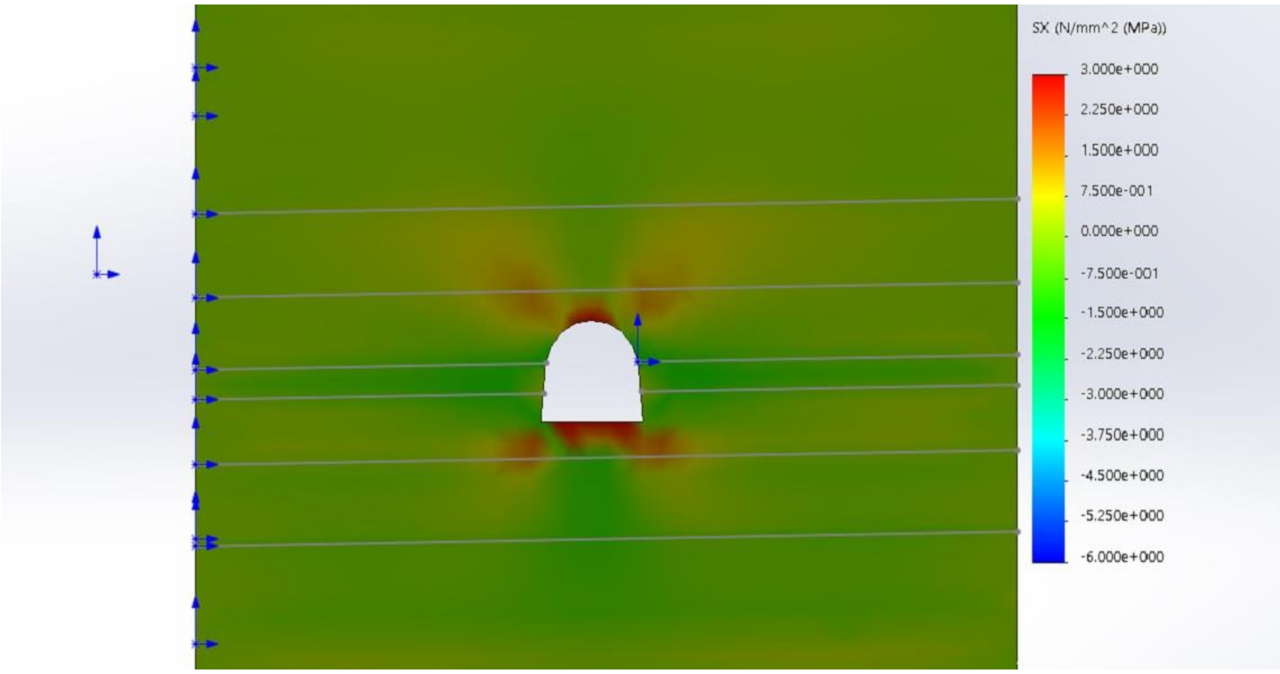

Figure 12. Horizontal stresses of the working

Considering the bottom of the working, the tension zone has less width than the width of the working and accounts for approximate 3 meters and depth of 2 meters. The tension stresses in this area are in the range from 2.0 to 2.5 MPa. It is also arising weakening zone in the bottom of the working, taking into consideration the fracturing factor and water saturation from the coal seam. In the right corner of the working, it can be observed stronger tension loads than in the left corner and this part of the right corner has sizes around 2 meters' width and 2 meters' depth. There are also weakening zones on both sides from the bottom of the working. This area has around 5 meters' on the strike and 3 meters' depth with the range of tension stresses from +0.75 to +1.5 MPa. Due to the tension stresses in the bottom, it is foreseen a rock heaving.

The 2 tension stresses zone occurs in the area with dimensions around 4 meters' width and 4 meters' height in the lateral sides of the working above the coal seam, and 2 zones about 5 meters' width and 3 meters' depth below the coal seam. Regarding that the enclosing rocks have a high degree of fracturing and water content, but the coal seam has a high strength, according to the action of compressive loads, destruction does not occur.

According to the quality factor, parameters of the distribution components curve of the stress-strain analysis are compliant with existing representation of deformation processes of the rock massif around the working. Moreover, the obtained results do not contradict to the numerous geomechanical investigations in this field of studying. It confirms the equivalence of the simulated model to the real conditions of mine workings maintenance in underground mines of the Western Donbass.

2.5.3 Strain-stress state analysis of the working in terms of intensity

Considering stresses intensity in the stress-strain analysis of the working, it was researched the stresses range of 0 MPa to +14 MPa. Stresses field σ allows to assess rock conditions from the synergetic action of the stress components (both vertical and horizontal) in the field of compressive forces.

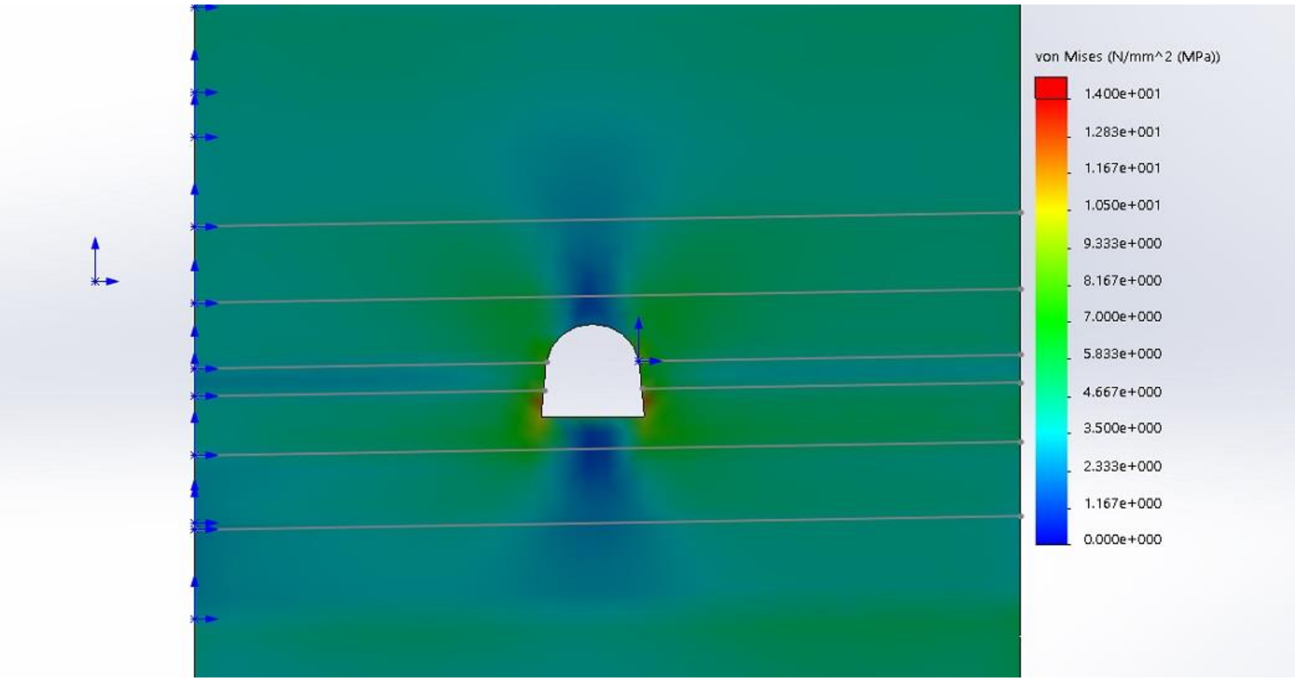

Figure 13. Intensity of stresses of the working

In this case, in 2 layers of mudstone and siltstone below coal seam, which are subject to factors of water saturation, jointing and rheology, there is significant area of rocks which are likely exposed to weakening. This area has approximate sizes of 1.5-meter width and 5 meters' depth. The range of stresses is from 1.3 MPa to 3.5 MPa. It is possible rock heaving in the bottom of mine and increased stresses on the lower part of mine legs. In the corners of the working, there are increased stresses, in the range of values from 11.0 MPa to 12.5 MPa. In this places, it is possible weakening rock.

Unloading areas (according to the vertical stresses and intensity) due to the extremely lowered stresses are dangerous in terms of the possible loss of stability in the bottom of the working, which can provoke a more intense heaving process.

2.6 Stress-strain state analysis of the support

In this analysis on the stress-strain state of the support have influence such parameters as support design, angle of bedding, frame lock and consolidated space.

In this investigation, according to the specification of the mine, it was simulated the metal arch three-link marquee pliable uniform support with prolonged legs of the support.

For the simulation and the most accurate reflection of the real design features of the support and lagging, the special significance has the shape of interchangeable special section, from which the shape is constructed. It was used the ISS – 22. In addition to this, it is important to design the frame lock, that provides the sustainable interaction between support and soft adjacent strata. The construction of the base plate under the frame leg is also important due to the limitation of the support indentation in the rocks of the bottom.

The frame support serves as a visual indicator of the intensity of the rock pressure manifestations in different parts of the working outline. According to the deformation of the frame support, it can be evaluated the degree of stability of the working and, if necessary measures can be provided to strengthen the frame support. In the number of works, it was investigated how geomechanical factors can influence on the working support and its yielding property [70,71].

For conducting of more reliable stress-strain analysis of the frame support, the range of values for stresses changes. In addition to this, the figures of horizontal, vertical curve diagrams and intensity curve diagram are made in the increased scale, since the frame support is under the action of stresses from shifting border rocks and the frame support dimensions are much smaller than dimensions of the rock massif.

As for the frame locks, due to the difficulty of the accurate reflection of the frame lock design, the constructional simplification was introduced to the simulation model, that allowed to calculate the stress-strain state more stable. In the area of the frame lock location, the filling from the easily deformable material was introduced. This ensured the flexibility of the frame under compressive stresses. The frame lock simulation in the form of a cross section of the corresponding special section is located at the coordinates of the frame locks and has length of 400 mm. The integral support was simulated with 2 lock inserts, that has mechanical characteristics of grade of steel № 5. The only one difference in the characteristics is in the decreased limit of the stretching strain (yield strength) in the frame locks. This allows to make simulation more stable.

It was set such mechanical characteristics of the frame support as $\sigma_t = \sigma_{compr}$=300MPa, elastic modulus = $21*10^4$ MPa, Poisson's ratio ν=0.3, shear modulus = $8.1*10^3$ MPa and mass density 7700 kg/m^3. For the frame locks, the elastic modulus was set 10^3 MPa, Poisson's ratio is 0.3, shear modulus $3.8*10^3$ MPa, $\sigma_t = \sigma_{compr}$=300MPa.

It was designed the consolidated space (backfilling) between support and rock massif, with the width of 0.15 meter. The backfilling is used for the improving of the connection between support with walls of the working, it helps to increase the stability of the rock massif by the increasing of the load-bearing strength of the support and conducts equilibrium distribution of the stresses. The mechanical characteristics that were set for the backfilling are: elastic modulus = 10 MPa, Poisson's ratio = 0.1, mass density = 2000 kg/m^3, σ_t = 0.1 MPa, σ_c = 3.5 MPa.

Analysis of the horizontal σ_x and vertical σ_y stress distribution of the support corresponds to the dispositions of mechanical engineering of the underground structures.

2.6.1 Stress-strain state analysis of the frame support in terms of vertical stresses

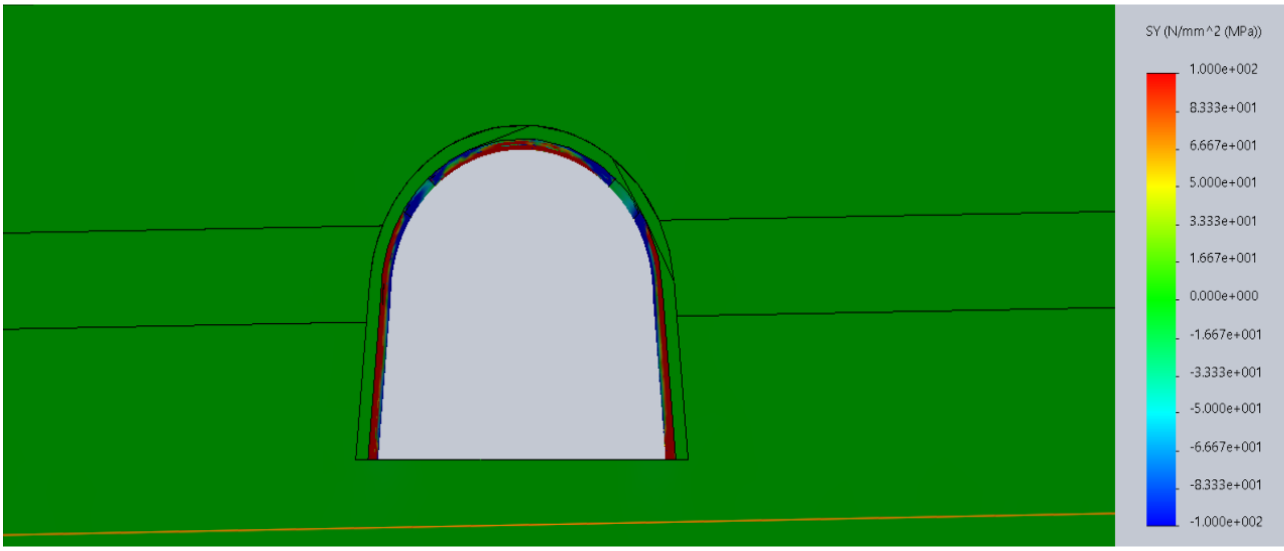

Figure 14. Vertical stresses of the support

In terms of vertical stresses, it was set such range of values from -100 MPa for compression stresses to +100 MPa for tension stresses. Inside the roof timber, it can be seen considerable tensile stresses in the central part of the support, the values are more than 250 MPa. This area has a width of around 1 meter. On the external part of the roof of the working, compression stresses act to more than + 200 MPa. However, these values are still lower than yield stress of the steel support.

In the areas of frame locks of yielding, it can be seen significant decreasing of stresses. On the both frame locks act compression stresses to more than -100 MPa. This is due to the effect of reduced deformation properties of the frame locks of yielding that have a length of 400 mm. The compression stresses act also on the distance of around 200 mm from both sides of frame lock with range of values from -30 MPa to – 100 MPa.

In the legs of the working, in the area that begin lower of the frame locks level, there is an action of considerable tensile stresses, more than 400 MPa. In such a way, it can be concluded that frame support will have possible destruction and such stresses provoke the development of the bending of the legs inside the working.

2.6.2 Stress-strain state analysis of the frame support in terms of horizontal stresses

For the horizontal stresses, it was set such range of values from -50 MPa for compression stresses to +50 MPa for tension stresses.

Considering the roof of the support, it can be observed high tension stresses on the internal side, that has a width of 2 meters. And on the external side of the support, act high compression stresses on all the area of the roof support between both frame locks. Such action of stresses characterizes the bending of the roof support into the cavity of the working.

On the level of frame locks, it is also can be seen the action of low compression stresses on the length of 400 mm. The range of values is more than -100 MPa.

In the legs of the working, act high compression stresses. These compression stresses are higher than rock resistance to compression. Due to this process, in the bottom of the working, the heaving of rocks takes place. In the both corners of the legs of the support, there are small areas of tension stresses.

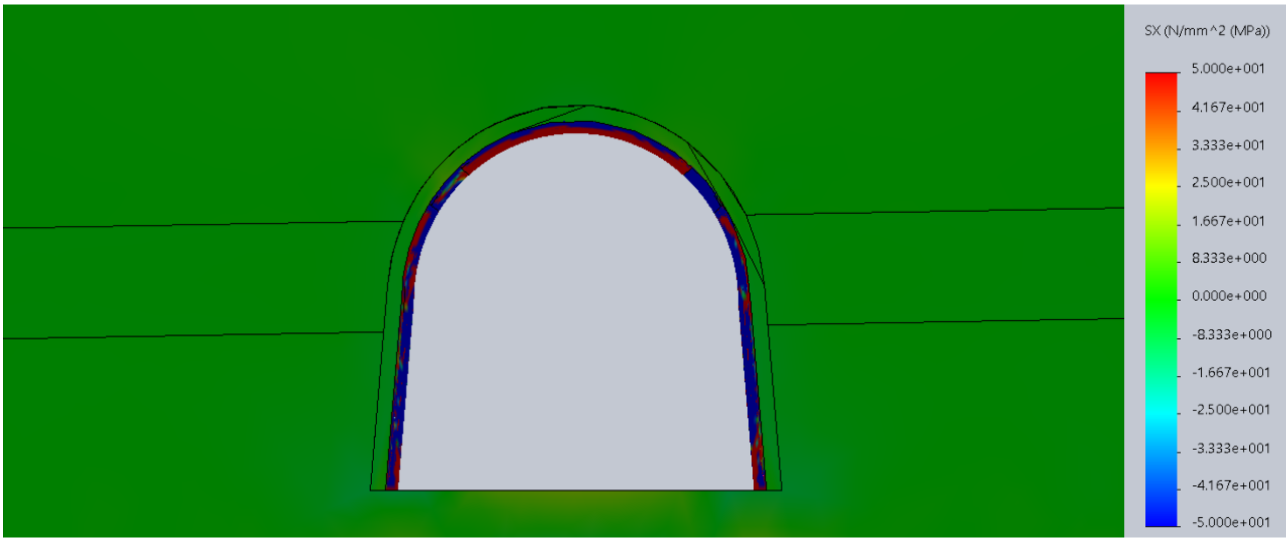

Figure 15. Horizontal stresses in terms of support

2.6.3 Stress-strain state analysis of the support in terms of intensity

The roof timber is exposed to local tensile and small compressive stresses and legs of the working are exposed to considerable compressive stresses, since they impound all the vertical loads. The horizontal stress has a high degree of compression in the roof timber because of the accumulation of wall pressures.

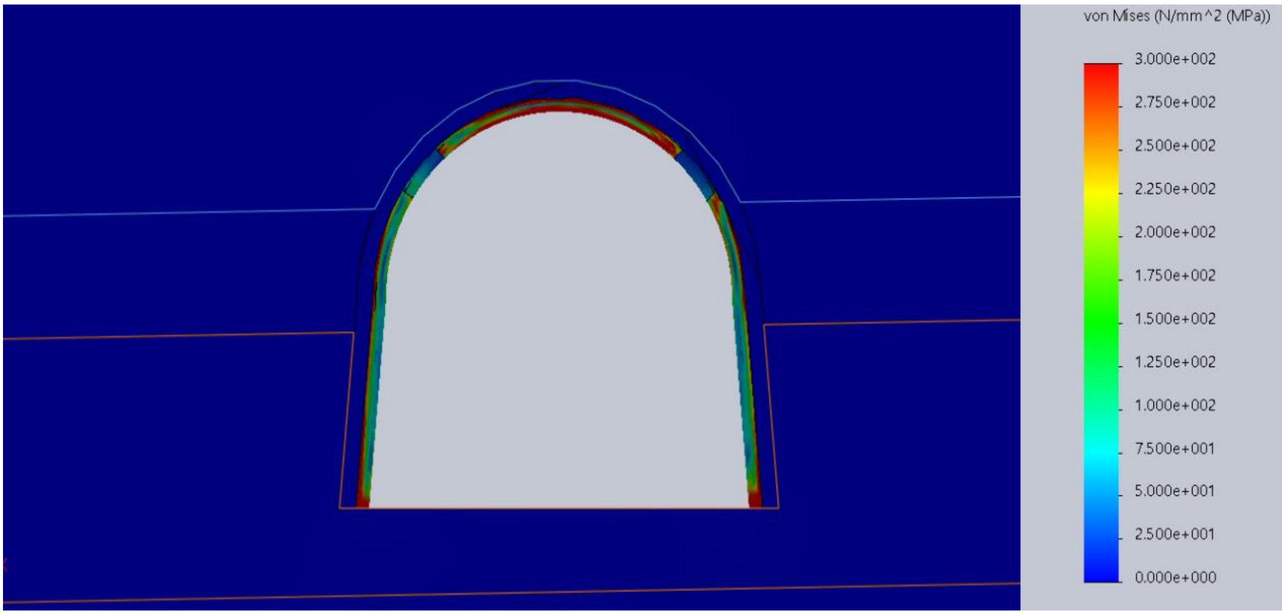

Figure 16. Intensity of stresses in the frame support

The frame support was loaded by 300 MPa and it can be seen local areas of tensile stresses in the frame timber and legs of the working, that are marked by red color.

And in the legs of the frame support, under the level of frame locks, the compression stresses act in the support.

It can be seen that legs of the frame are under the action of increased vertical compression stresses and decreased horizontal stresses. This stresses in common form extended areas of the yield state of the frame support, which leads to a loss of stability of the legs of the frame and the frame support as a whole.

It should be taken measures to increase the stability of the frame and limit the areas of the plastic state. In such a way, the next step of simulation is adding rock bolts to the model for providing more stable state of the support and rock massif.

2.7 Stress-strain state of the rock bolts (anchors) and support

In the mines of Western Donbass, it is highly used the combination of different types of rock bolts with the frame support. The most popular type is resin-grouted roof bolts that has length to 2400 mm. During last years, it is started also to use rope bolts in the roof of the working that have length to 6000 mm. In such a way, the support system of the mine working consist from frame support, rope bolts and resin-grouted roof bolts. Each of the elements of the support system interacts with each other and works in the same area of the bordering rocks of the massif and resists the same manifestations of rock pressure.

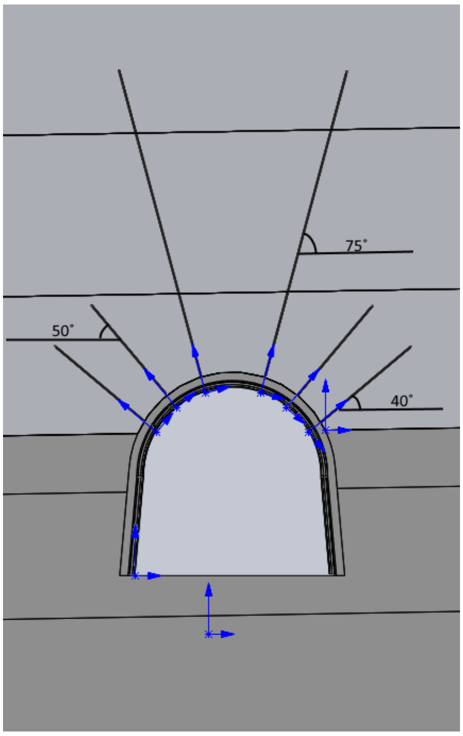

Figure 17. The layout sketch of rock bolts

For the qualitative assessment of the stress-stain state of the support system, it is important to choose the appropriate scheme of roof bolts position. According to the number of field experiments, there is a high difference between stress-strain state of roof bolts during changing of their position in the working, for instance their location in the roof timber or in the walls of the working, and the type of roof bolts, for example rope bolts or resin-grouted roof bolts. According to the number of simulations, it has been discovered, that resin-grouted roof bolts have higher extent of loading in the area of frame locks location and in the walls of the working. And the rope bolts are better to locate in the roof timber. In addition to this, the depth of the working has an importance.

As a consequence, there are 2 main points for the choosing of the roof bolts position. The first one is different extent of rock bolts loading in the rock massif. It was found that, resin-grouted roof bolts have low extent of loading in the roof timber and walls of the working. And the highest level of the loading resin-grouted roof bolts has in the walls of the working, lower and higher of the frame locks area. And the second reason is in the high level of rope bolts loading in the central part of roof timber. Due to the length of rope bolts that reaches 6000 mm, there is a significant difference between the displacements of the rocks deep in the rock massif and near the working outline. In such a way, there is a significant resistance of rope bolts again subsidence and their effective support influence can be seen [70].

One more important reason, that is need to be highlighted for the support system choosing, it is the increased lateral rock pressure in the workings of the Western Donbass from the immediate roof of the working and bottom of the coal seam. This feature is explained by creation of stronger plate by combination of rope and resin-grouted bolts in the immediate roof and coal seams have higher compression resistance in comparison to immediate roof and bottom. As a result, soft and water-saturated rock of the immediate roof and bottom presses into the cavity of the working due to the action of bearing pressure.

In this investigation, it was chosen the layout sketch, where it is used 6 anchors. Among them 2 centrals are rope bolts, and 4 bolts in the roof of the working, 2 on each side are resin-grouted rock bolts. Such layout of rock bolts is especially suitable for the conditions of the Western Donbass mines for the rock pressure control and workings with reuse. The diameter of both types was taken 22 mm. The length of rope belts is 6000 mm and length of resin-grouted bolts is 2400 mm. The rope belts are conducted at an angle of 75˚ and approximate distance between them is 1000 mm. The upper resin-grouted rock bolts are conducted at an angle of 50˚ and are situated just above the level frame lock, and the lower resin-grouted rock bolts are conducted at an angle of 40˚ and are situated just below the level of frame lock.

There were set such mechanical characteristics for rock bolts as $\sigma_{compr} = \sigma_{ten} = 300$ MPa, elastic modulus is $2.1 * 10^5$ MPa, Poisson's ratio is 0.3, shear modulus is $8.1 * 10^4$ MPa and mass density 7800 kg/m^3. These characteristics were set for both types: resin-grouted and rope rock bolts.

According to the number of investigations, rope bolts are highly suitable for the layered rocks and slightly metamorphosed massifs.

However, the lateral soft rocks complicate the application of rock bolt systems.

Consolidation of the working by rock bolt’s systems is highly effective and applicable in the number of mining enterprises in the whole world. Rock bolts serve for prevention of the increasing of disturbed rocks area around the working. The most effective system is combination of usual-sized rock bolts with deep-laid rock bolts.

In this work, for the third simulation model, it is used 2 deep-laid rope bolts. These rock bolts are made in the form of a cable consisting of flexible steel cores, which allows to use them with higher length than the working height. Functioning principle of rope bolts in the fixing of rocks that are expected to be in the failure zone to overburden and more stable rocks [72-74].

Due to yielding bracing of rope bolts and frame support, bearing ratio of the frame support is increasing. The combination of rope bolts and support changes the rock massif behavior and leads to the joint deformation processes of rock massif and frame support. In addition to this, in areas of increased rock pressure, there is a process of self-adjusting changing of resistance to stress [70].

The symmetrical installation of rock bolts in relation to the vertical axis of the mine working is grounded by the fact that there is the opportunity of reusing the mine working and aligning the asymmetry of the geomechanical processes behavior [70].

Another 4 rock bolts are resin-grouted roof bolts. They consist of a steel bar with a ring seal. At the end of a roof bolt that is situated in the working outline, there is a screw and back plate. The deep end of the roof bolt is fixed in the bore hole by a polymer concrete [75].

2.7.1 Stress-strain state analysis of the rock bolts and support in terms of vertical stresses

For the vertical stresses, it was set such range of values from -50 MPa for compression stresses to +50 MPa for tension stresses. Analysis of the vertical stresses distribution shows that on the external face of roof timber act tension stresses, which marked with green color. This part of the roof timber is under tension stresses around +30 MPa. The internal part of the roof timber is under action of compression stresses, which values range to -60 MPa. Such unloaded state of the roof timber is explained by the support of rock bolts.

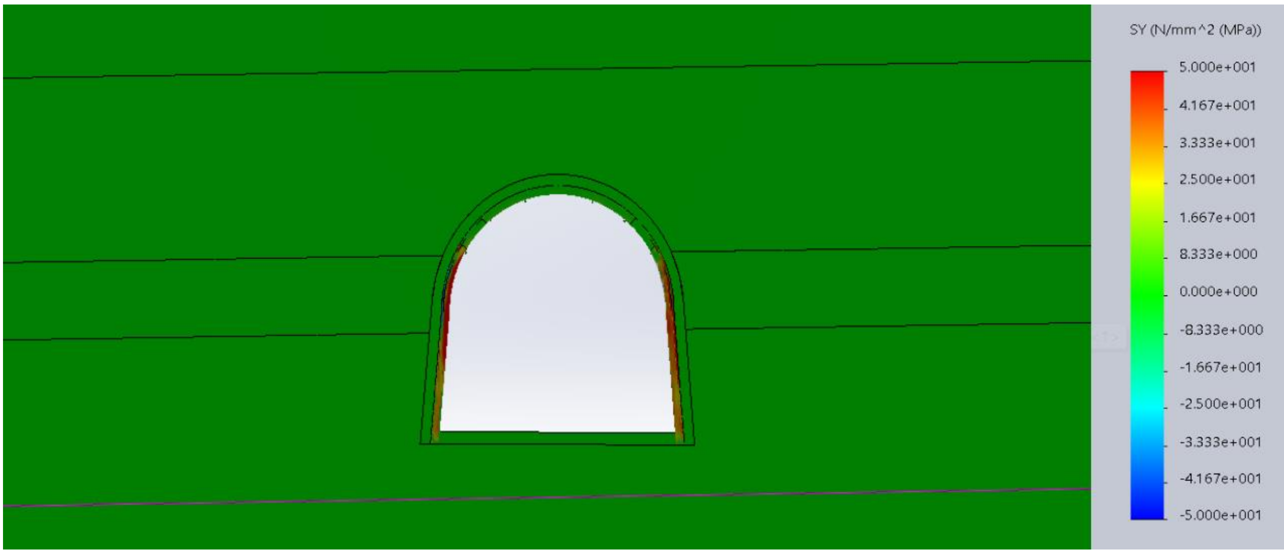

Figure 18. The vertical stresses in the support

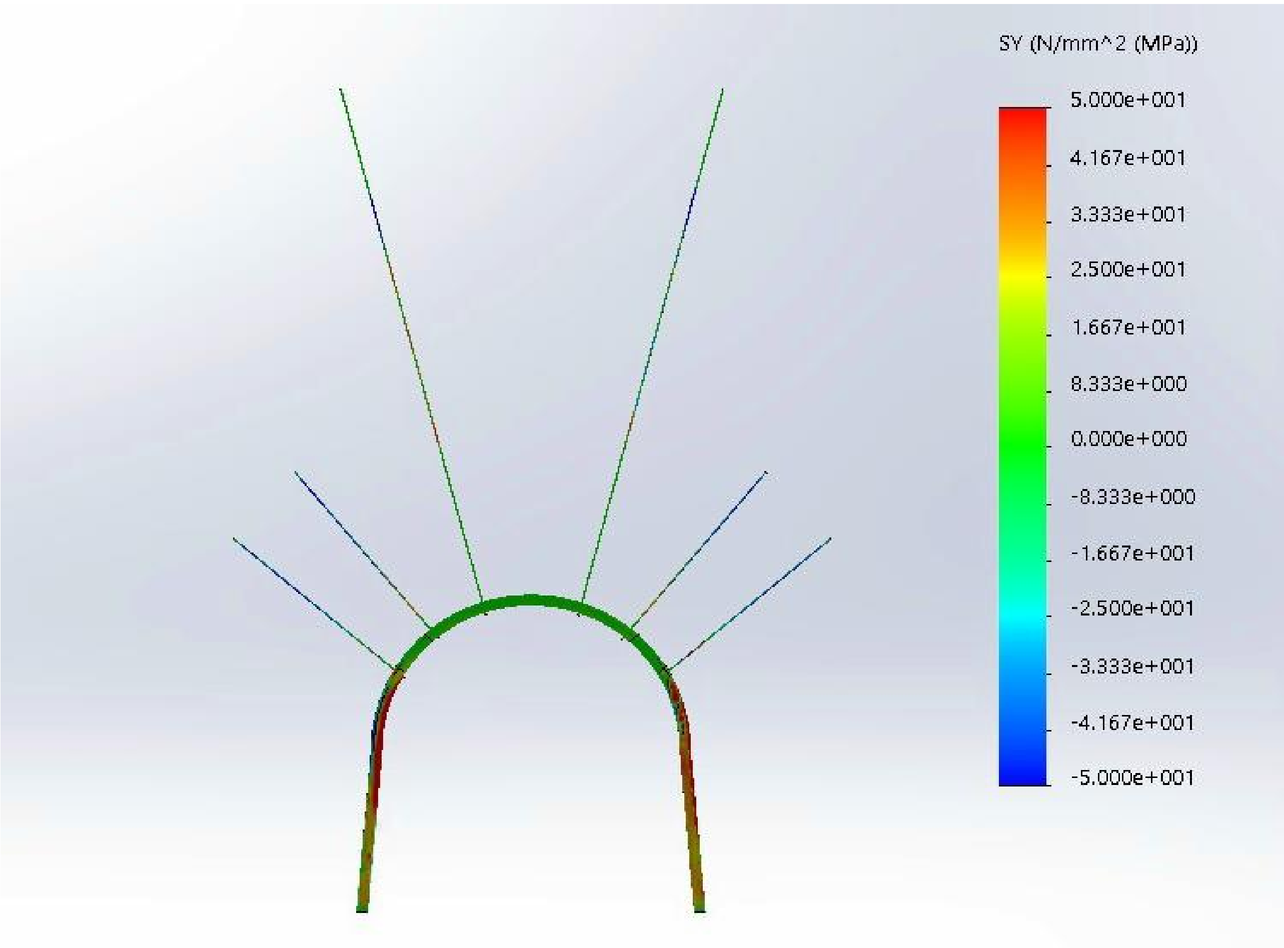

Figure 19. The vertical stresses in the rock bolts

In the internal part of the frame legs, the tension stresses act in the range to +130 MPa. The legs of the working are marked with red color. In the corners of the legs, it can be seen also low tension stresses, that are in the range from +10 MPa to +30 MPa. On the external part of frame legs act low compression stresses in the range from -10 MPa to -50 MPa.

It can be seen compression stresses in the internal part of frame locks that smoothly turned into tension stresses on the external part of frame locks in the range from -10 MPa to +15 MPa.

In such a way, the vertical stresses that act on the support system of the working do not exceed yield stress of the steel and the working is in the stable condition.

According to the layout of rock bolts, it can be seen that 4 resin-grouted rock bolts in the walls of the working are under the action of high tension stresses in the area of immediate roof and marked with red and green color. The tension stresses are in the range from +30 MPa to +50 MPa. Above that level of immediate roof, in the main roof, that is represented by mudstone as well as immediate roof, the resin-grouted rock bolts are under the action of compression stresses in the range of -35 MPa.

Considering the 2 rope bolts in the central part of the roof timber, they are under the action of low tension stresses at around +25 MPa in the immediate roof and turns into higher tension stresses in the area of main roof. In the upper seams, where the length of rope bolts is from 3 to 6 meters, there is an action of low compression stresses that turns into low tension stresses in the range from +15 MPa to -10 MPa.

2.7.2 Stress-strain state analysis of the rock bolts and support in terms of horizontal stresses

For the curve or horizontal stresses, there were set such range of values from -100 MPa for compression stresses to +50 MPa for tension stresses.

Inside of the room timber, the tension stresses act in the range from +10 MPa to +50 MPa in the central part. And on the external part of the roof timber, act low compression stresses in the range from -50 MPa to -80 MPa.

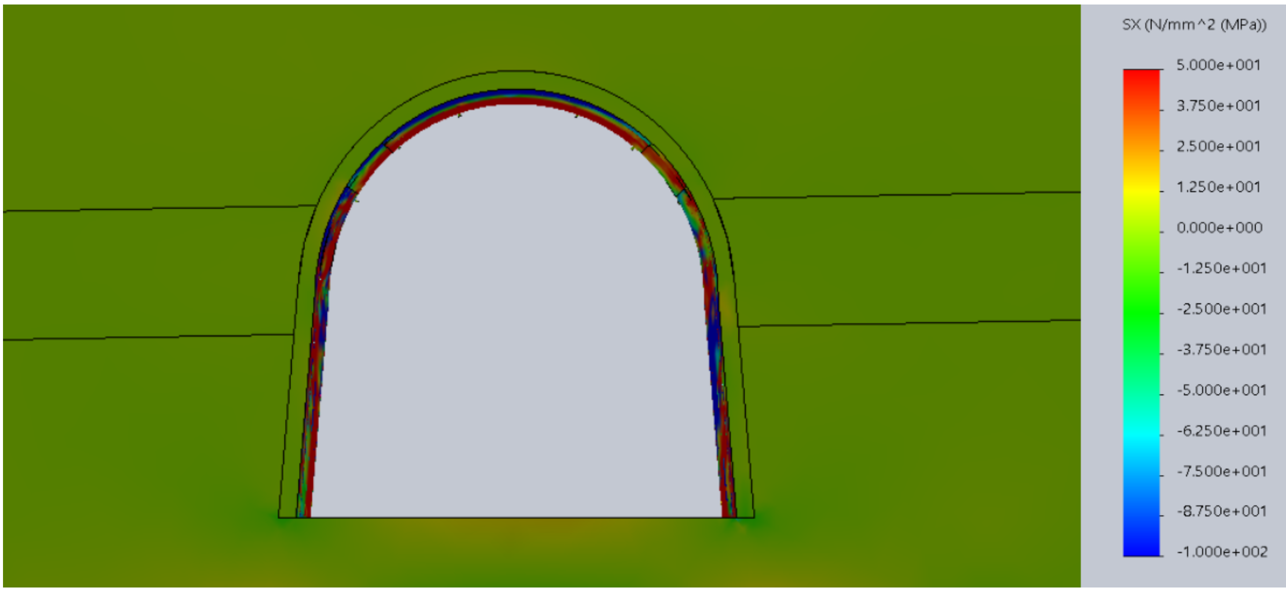

Figure 20. The horizontal stresses in support

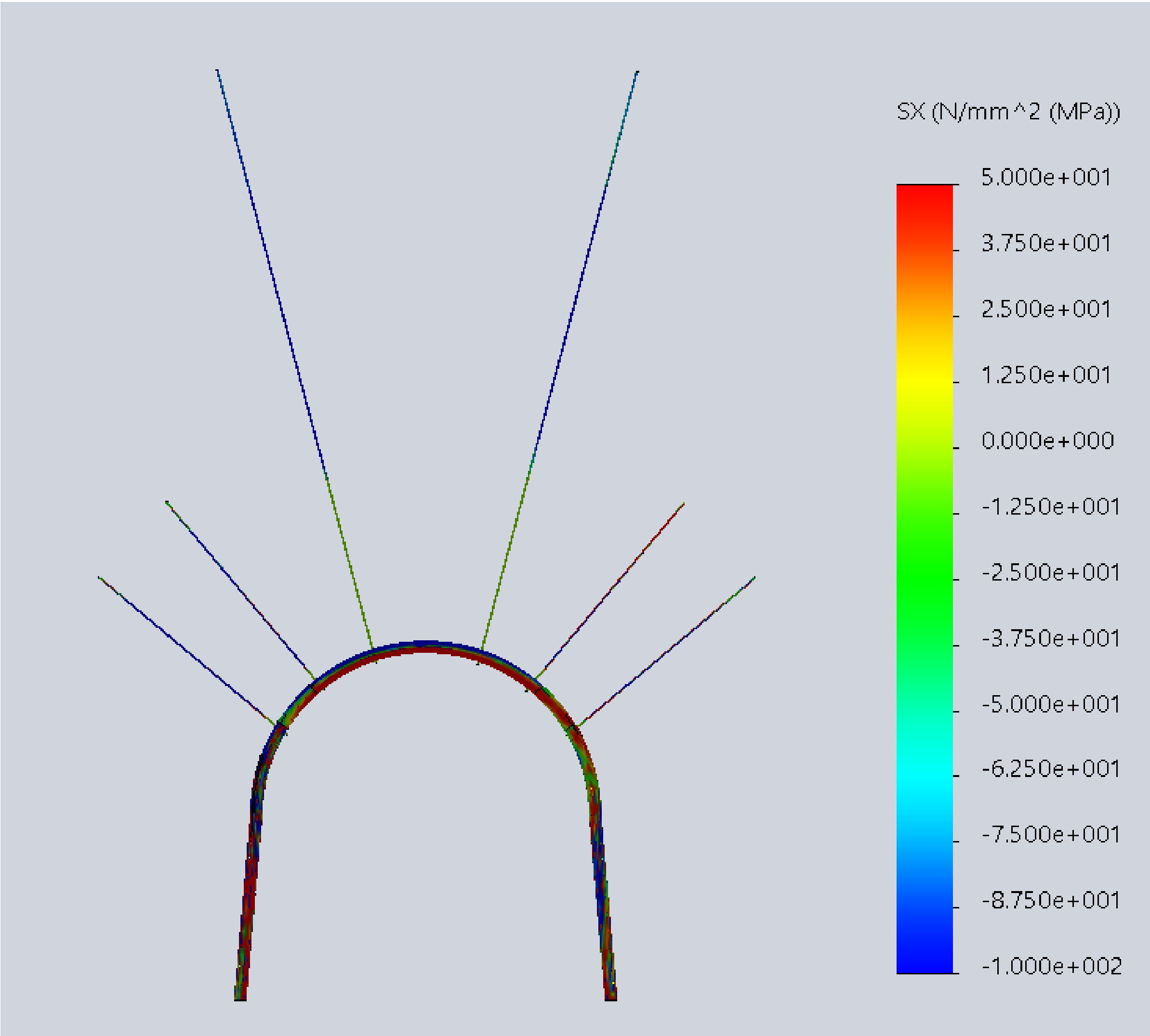

Figure 21. The horizontal stresses in the rock bolts

In the legs of the working, in the internal part compression stresses act in the working, marked with light blue colour. Closer to the working outline, it can be seen tension stresses in the legs of the working, colored with green and red colores. The tension stresses act in the range of +30 MPa and compression stresses on the internal part act in the range to -40 MPa.

In the area of frame locks, act most of all low compression stresses in the range from -5 MPa to the -20 MPa.

The assessment of the horizontal stresses proves the curve of vertical stresses, that the roof timber and legs of the working under the action of stresses flexes into the working.

In 4 resin-grouted rock bolts in the walls of the working, closer to the working outline act tension stresses on the length of around 2 diameters of rock bolts. Further on the roof bolts act compression stresses in the range of -80 MPa.

Considering the rope bolts in the area of immediate roof they are under the action of low compression stresses in range of -20 MPa and in the area of main roof the action of compression stresses increases to -60 MPa.

2.7.3 Stress-strain state analysis of the rock bolts and support in terms of intensity

For the intensity curve it was set such range of values from 0 to 200 MPa. In the roof timber it can be seen increased stresses in the internal part of the support. Because the action of vertical stresses, the shape of roof timber tends to flexing. The vertical stresses act in the internal part of the support as tension stresses.

Inside the support, it can be seen the highest stresses. In the legs of the working, in the internal part, the stresses act to the values of almost 180 MPa.

It can be concluded that the roof timber bends inside the working under the action of stresses, legs of the working also bend into the working under the increased stresses.

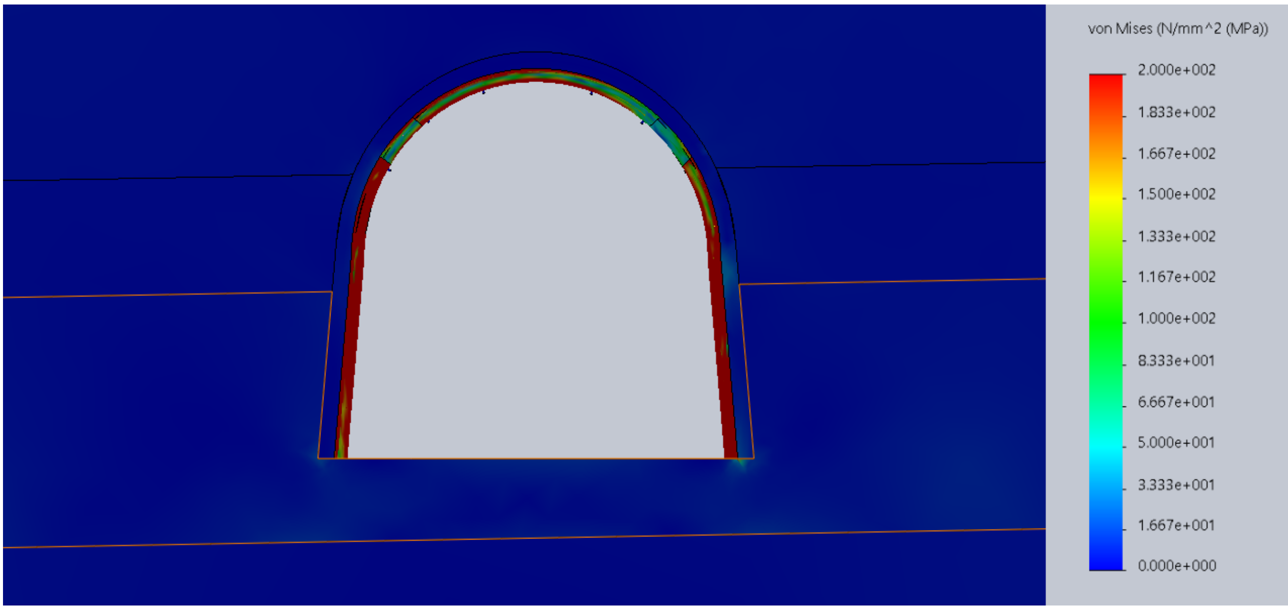

Figure 22. The intensity in the rock bolts and support

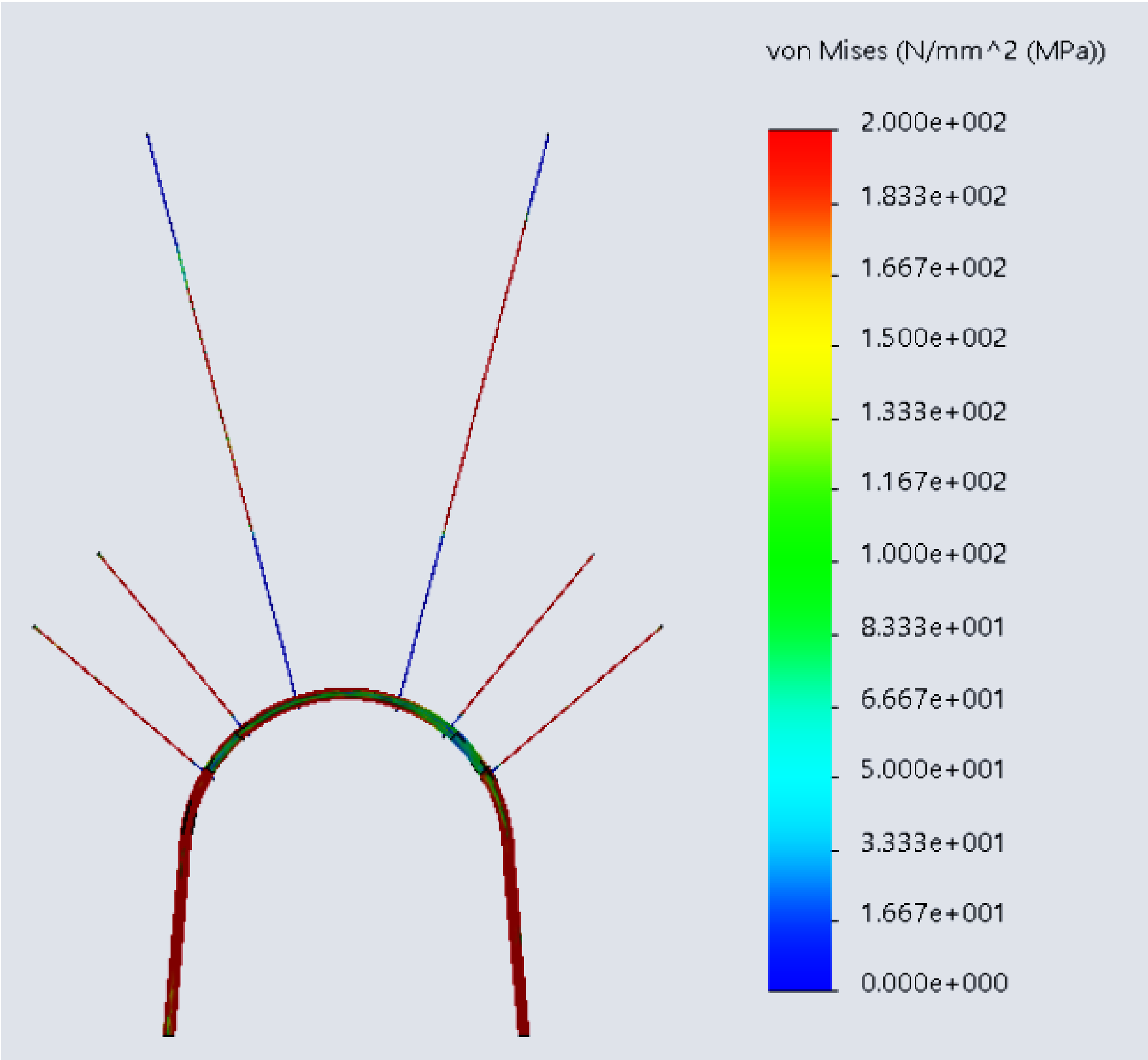

Figure 23. The intensity of rock bolts

Considering the simulation model of rock bolts (the intension curve), it can be seen that 4 anchors in the wall sides of the working are under the action of tension stresses and marked in red color. Closer to the support, on the level of the backfilling, they are under the action of compression stresses and colored in the blue color. These 4 anchor bolts are resin-grouted and loaded in the level of immediate roof.

And 2 upper rock bolts are under the action of compression stresses from the level of working support to the level where 4 resin-grouted rock bolts end. But higher of this level, it can be seen that anchors are under the action of tensile stresses and loaded in the level of main roof, that is represented by mudstone as well as immediate roof. Higher this level, in the end of anchor bolts in the layer of siltstone, they are colored in blue and are under the action of compression stresses.

The obtained results indicate that the "frame support-rock bolts" system is in a stable condition and does not exceed the yield strength of steel. The obtained results of vertical loads indicate increased stresses in the frame legs, which can be strengthened with resin-grouted rock bolts below the level of coal seam (at an angle of around 170°). Rock bolts withstand a given load. However, the maximum load on the rope bolts is in the middle of the length of the bolts. As for resin-grouted rock bolts, they are in a loaded state, but the load on them does not exceed their limits. Thus, according to the results of computer simulation, it can be seen that the working is in a stable state, the support system copes with its main task in the given geological conditions.

2.8 Workplace safety during roof-bolt setting

The staff personnel involved in the construction of the anchor bolts must go through specific training. All work on the construction of the roof bolting should be carried out under the protection of the previously installed excavation support.

It is forbidden to carry out installation of roof bolts from a combined machine.

During drilling boreholes and construction of roof bolts, at least two people should be in the mine at the work place. It is forbidden to walk around the support setting of the tail chain during shifting of lining.

Drilling of boreholes should be carried out under the protection of the grid-tightening, which is pressed to the roof of the mine with a crown tree that is hold by the auxiliary support.

Alongside with the revision of face, it is necessary to inspect the ventilation pipes, eliminate the discovered defects and hang the cables. In gaseous mines, before starting work on drilling boreholes and construction of rock bolts, it is necessary to determine the concentration of methane in the atmosphere of working.

Drilling of rising blast holes should be accompanied by dry or wet dust allayment, otherwise workers should be provided with respirators and safety goggles.

It is forbidden to drill boreholes through exfoliated pieces of rock in order to avoid collapse of the rock.

Drilling rigs are systematically tested for their ability to provide the required rotary moments.

The diameter of the bore bits for drilling bore holes should correspond to the diameter of the used rock bolts, checking the diameter of the bore bits is also made after their renewal (sharpening).

It is necessary to check the length of the drill rods: the length of the holes should not be more or less than required.

It is necessary to use drilling bits that correspond to the strength of the rocks.

It is necessary to carry out careful quality control of drilling bits during the purchase. The dimensions of the drilling bits should provide drilling holes for all types of work where rock bolts are used.

It is necessary to check the quality of the drill rods, to fix all deviations and deficiencies. This applies in a greater degree to the drill rod carrier.

The installation of polymer ampoules must be done in special gloves.

Ampoules with damaged covering that were found at the workplace should be placed in a double plastic bag, vacuum-packed, taken to the surface and disposed. The number of damaged ampoules should report in acts. It is forbidden to fault (casing breakage) of the ampoules before introducing them into the hole.

During installing of chemical ampoules into a hole, it is necessary to make sure that they are equally located in the well along the entire length.

Prior to fixing agent setting, it is prohibited to suspend equipment and other objects to the elements of rock bolts support.

During installing of rock bolts and tightening of locknuts, the crown trees should be firmly pressed to the roof and side rocks.

The adapter for tightening of rock bolts must be checked for suitability and run-out, which will ensure the appropriate moment of rotation of the rod.

The construction and operation of a mine without safety indicators are prohibited.

2.9 Measures for safety work

Dust generation ability of coal seam C_5+C_5^u according to the catalog of mine seams by dust factor in the "Guide to the fight against dust in coal mines", it belongs to group I of dust formation. The specific dust emission per ton of coal extraction is 15 g/t. The dust content in the loosened coal is 0.25%. Coal humidity (analytical) - 2.1%. Seam thickness (average) - 1.03 m.

The mine is classified as a dust hazardous. The specification of the mine provides additional measures in accordance with the requirements of the "Instructions for the Prevention and Localization of Coal Dust Explosions" (Collection of instructions for the Safety Rules in Coal Mines) [76].

If smoke is detected in the mine workings, it is necessary immediately to engage in the self-rescuer and move along the ventilation stream to the nearest workings with a fresh air stream and then along the fresh air stream to the auxiliary shaft and to the surface [77].

2.9.1 Measures to dust control of air. Dust removal of the return ventilation air. (cleaning of the ventilation air)

For continuous coal dust consolidation in the return ventilation air and reduction of dust deposition, it is necessary to use water barriers, which are installed dispersed in the area of the ventilation drift adjacent to the longwall face. The water barriers are installed on the return ventilation air no further than 20 m from the longwall face in the direction of the air stream.

Water barriers are installed in such a way that the cross section of the mine working is completely blocked by the mist spray. For every 500 m^3/min of passing air, one water barrier is installed. If it is necessary to install several barriers, the distance between them is taken equal to 3-5 meters [76].

Water barriers will be installed in the working area of the mine.

Irrigation during the operation of the combined machine will ensure residual dust content of the air at the level of:

$$C = \frac{1000 q_{de} v 16{,}7 K_g K_s P_c}{Q_a} K_u K_r K_{s'} \frac{mg}{m^3};$$

where q_{de} – is a specific dust emanation of the coal seam (15g/t);

P_c – productive capacity of the combine machine, t/min (2,67 t/min – for all longwall faces);

V – moving velocity of the ventilation air stream in the working face, m/s (1,6 m/s); Q_a – amount of air that is pass through the working face, m^3/min (252 m^3/min for the 543 longwall face);

K_g –index of the reduced degree of grinding, (0,11)

K_s –index, taking into account the change in specific dust emission depending on the combine configuration (0,9);

K_u - coefficient that takes into account the upper limit of dust fineness (0,5);

K_r - coefficient that takes into account the presence of dust removal measures;

K_s - coefficient that takes into account the influence of the speed of the ventilation air stream in the longwall face (0,9);

$K_r = (1\text{-}E_1)\ (1\text{-}E_2)$,

where E_1 - coal moistening efficiency in the massif, in a quantity;

E_2 - combine irrigation efficiency, in a quantity;

$K_r = (1\text{-}0)\ (1\text{-}0,\ 792) = 0{,}208$;

For the longwall face № 543:

$$C_{543} = \frac{1000 * 15 * 1{,}5 * 16{,}7 * 0{,}11 * 0{,}9 * 2{,}67)}{242 * 0{,}5 * 0{,}208 * 0{,}9} = 34{,}8\frac{mg}{m^3};$$

Due to exceeding the level of maximum permissible dust concentrations (4 mg / m^3), it is required to use the dust respirators. The efficiency of the water barriers is 80%. Since the residual dustiness of the air (C) after irrigation at the combine is more than 4 mg / m^3, the use of water barriers is necessary.

Determine the number of water barriers. The air dustiness after working development:

1st water barrier:

$B_1 = 38{,}4 – (38{,}4 * 0{,}8) = 7{,}68$ mg/m^3;

2nd water barrier:

$B_2 = 7{,}68 – (7{,}68 * 0{,}8) = 1{,}54$ mg/m^3;

The daily water consumption for the water barrier is:

$Q_d = Q_t * T$, l/daily;

Q_t - water consumption per unit time, l/min;

T - the duration of the barrier working per day, min.

$Q_t = V * q_{bar}$, l/min;

where: V is the amount of air passing through the longwall face and the water barrier, m^3/min;

q_{bar}– specific water consumption for air purification from dust, l/m^3 (0,1 l/m^3 - for the water barrier).

The duration of the barrier working is equal to the duration of the combine working per day, which is determined from the expression:

$$T = A/P_c, \text{ min}$$

where P_c – productive capacity of the combine machine (2,67 t/min);

A - daily production from the longwall face, t.

The excavation section of the 543 longwall face is ventilated according to a straight – through ventilation arrangement. Fresh air enters the longwall at 543 development drift. The outgoing air stream comes from the longwall to the 543 boundary entry, where it is refreshed with a fresh air stream. As the mining pillar of the longwall face is mined, 543 boundary drift is supported, 543 development drift is abandoning.

At the 543 boundary entry, at the distance of no more than 20 m from the the ventilation air stream exit from the longwall face, it is installed 2 water barriers for dust removal of the outgoing air stream. The number of barriers is calculated earlier.

The water flow rate of a water barrier B3-2 type is 50 liters at a pressure 1.2MPa. For dust removal of the outgoing air stream, it is accepted 2 air barriers of the B3-2 type.

To remove dust from an outgoing air stream:

$$Qt_{.out.} = 242 * 2 * 0,1 = 48,4 \text{ l/min};$$

The duration of the barrier working is equal to the duration of the combine working per day, which is determined from the expression:

$$T = 1836 / 2,67 = 688 \text{ min};$$

Then:

Q_d = 48,4 *688 = 33299 l/daily = 33,3 m^3/day.

All the values were taken from the "Mine dust removal instructions" [77].

2.9.2 Fire prevention and protection activity

According to the mine specifications, for the purposes of fire extinguishing and dust control, fire-irrigative line of pipes is set in the room entries and equipped with fire-extinguishing means.

To provide a total water consumption for fire extinguishing of at least 100 m^3/h, for workings equipped with belt conveyors at the 543 development entry to the conveyor drive 2LT-1000D, the water pipeline is constructed for the connection to the UVKP automatic fire extinguishing system. The diameter of pipeline is 150 mm.

Water barriers are used to localize coal dust explosions, that constructed according to the chapter 6, part 7, par. 16-17 of "Coal mine safety rules".

The wall sides and the roof of the entry should be washed periodically with water, according to the chapter 6, part 7, par. 20 of "Coal mine safety rules".

As the coal seam of $C_5 + C_5^u$ s not prone to spontaneous combustion (Conclusion of the «Respirator» Research Institute of Mining Rescue Affairs), special measures to prevent coal spontaneous combustion are not provided.

The calculation of the fire-irrigation pipeline for the 543 development drift and 543 boundary entry was carried out by the MRA research institute in the "Mine Fire Protection Project" and complies with the established Safety Rules.

The amount and arrangement of primary fire extinguishing means were taken in accordance with the requirements of the "Instructions for the fire protection of coal mines" (Collection of instructions to the Rules for safety in coal mines) [76,77].

PART 3

SELECTION OF THE WORKING SUPPORT PARAMETERS. ARGUMENTATION OF THE PARAMETERS OF THE ROCK BOLTS SUPPORT LAYOUT

3.1 Analysis of the used support structures

The choice of rational parameters for the placement of the support system elements of working is the most important task of this section. The choice of parameters of the support elements depends on a number of geological and mining conditions for workings maintaining.

The main point of installing resin-grouted rock bolts is their maximum unloading of frame support to ensure the minimum loss of the working cross section and provide the maximum resistance to the rock pressure.

The number of reason for installing both resin-grouted and rope rock bolts are described in the Section 2.7 of this work.

In this work, 4 resin-grouted rock bolts that have length 2400 mm were installed in the walls of working in the area of frame locks location. Such a support system is reasoned by studies of stress-strain state support systems in different mining and geological conditions of the Western Donbass. According to the studies, during installation of resin-grouted rock bolts in the central part of the working's roof, it is determined their reduced loading. In addition to this, the roof timber is loaded not so much as legs of the working.

However, in the walls of the working, the load on the rock bolts increase due to the action of bearing pressure. To ensure maximum resistance of rock bolts in the walls of working to the movement of roof rocks, the preferred direction of movement of rocks of the immediate roof is determined:

- 40˚ to the horizontal axis – in the area of coal seam bedding plane;
- 50˚ on the height of around 0.6-0.8 meters from bedding plane.

It is the angles of lower and upper rock bolts installation respectively.

Installation of resin-grouted rock bolts limits the negative effects of rock deformations or completely eliminates them. Dangerous factors include the following:

- excessive lowering of the roof and loss of the drift cross section;
- the formation of cavities between the immediate and the main roof, which can provoke a dynamic effect when the main roof collapses and the drift is removed from service;
- irregularly distributed load on the support system elements, since the load concentration transmitted to immediate bottom of working and increases the destruction possibilities of drift rocks;
- the formation of increased lateral pressure on the frame legs, heaving of the soil and subsidence of the roof rocks [70].

Considering the working support by rope bolts, their effectiveness and multitasking improves the stability of the workings. Their usage refines the number of factors, among which:

- increased safety of work;
- the working stability in the zone of actual mining influence;
- improves the ventilation level of the working;
- they limit the roof stratifying and provide its original structure up to the run of working face.

The following installation options for rope bolts are recommended. The rope bolt's shank is located at a distance of 0.5 to 1 m from the vertical axis of the working, which ensures the maximum decreasing in moment of resistance in the roof of the frame support. The angle of rope bolt inclination is 70-80˚, which coincides with the movement of roof rocks in this section of the working kettle and the rope bolt will create maximum resistance. The length of rope bolt should be no less than 6 meters.

Considering the rock bolts scheme that was applied in the "Dniprovska mine", there are 2 possible layouts: 6 resin-grouted rock bolts with stepping increment of 1 meter and 4 resin-grouted rock bolts with the stepping increment 0.8 meters (figure 24).

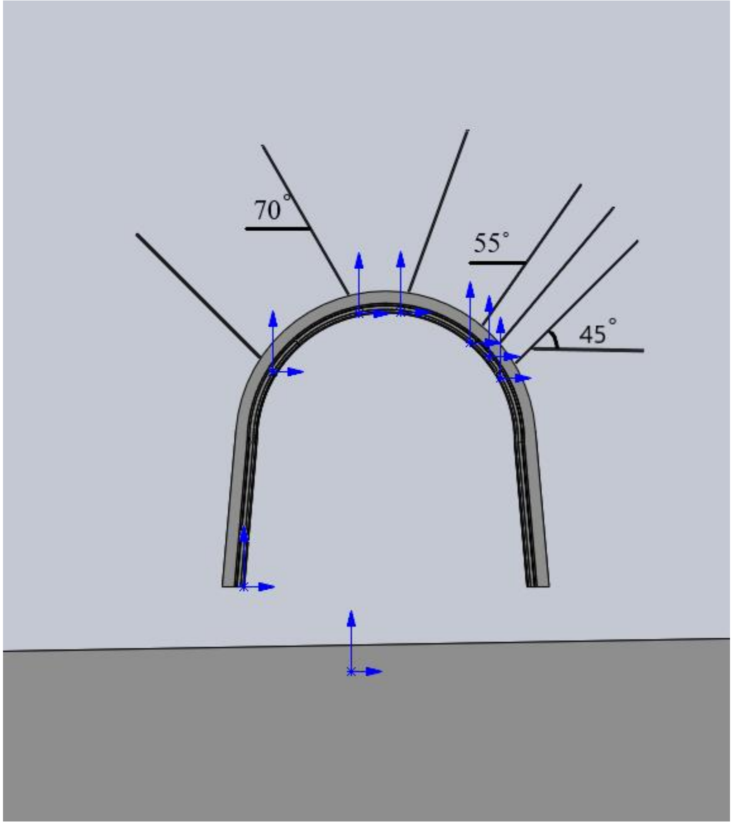

Figure 24. Rock bolts layout of "Dniprovska" mine

The support setting of the development drift with longwall face is made by strengthen advance support, which consist from 2 rows of hydraulic pops with diameter 18-22 cm, with compliancy elements installed under the metal baulk and frames of rock bolts support.

Strengthening the support system of the development workings behind the longwall is carried out by restoring wooden props with a diameter of 18-22 cm with compliancy elements (pieces of timber L=400 mm) under 2 central metal mud stills. The upper part of the wooden props should be fixed to the top of the frame support with a wire with a diameter of 6 mm.

Additionally, in the developed zone of inelastic deformations in case of unstable condition of the working's walls, rock bolts with a length of 1500 mm are installed at a distance of 250-300 mm below the coal seam.

3.2 Substantiation of the resource saving at the supporting of development drift

According to the investigation of influence of the substitutable special section type on the stress-strain state of the "stratified massif-working support" system [67], it was found that cost-effective use of resources is possible during reducing of metal consumption for frame support and as a result decreased labor costs. There are 2 ways of decreasing the metal consumption under the conditions of strengthening of the working roof by rock bolts. The first one is in increasing of the frame support setting increment and the second one is in applying more easy type of substitutable special section.

However, the first option is problematic due to the mining and geological conditions of the Western Donbass. Since rock massif is soft and metamorphic, the increased setting increment will lead to the increased volumes of rocks that are accounted for 1 frame support. The load on the frame support will increase. As a result, it is possible to have loss of bearing capacity.

Considering the increasing of the substitutable special section number and increasing of metal amount, this option will strengthen the frame support, but there will not be cost-effective use of resources. Since with the increasing setting increment of the support, it will be also increased amount of the frame support metal section.

Also, it should be paid attention that due to the soft rocks that are often under the action of weakened factors such as fracturing, water saturation and rheology, the increased distance between frame supports can lead to the loss of the massif stability and there are possible local rock failures.

The second option is changing of special section on the one with less metal consumption. In the investigation [67], it was changed from SSS-27 to SSS-22 profile and even SSS-19. The simulation showed not significant increase in action of stress on the frame support.

Applying this option to this work, it is possible to change the frame support profile from SSS-22 to the SSS-19. Reducing the number of special section's profile does not lead to a significant change in the stress-strain state in the frame support, which confirms the practicability of reducing the capacity of the metal in frame support and the complexity of its installation.

According to the previously mentioned investigations, the changing of the special section profile from number 27 to number 19 and with the same installation step of the frame support, allowed to decreased the metal consumption on around 30%. As a conclusion, in the given mining and geological conditions, where the depth is less (200 meters in comparison to 362 meters) and as a consequence less stress on the rock massif, such a resource saving option is suitable.

As a result, for the optimization of the development drift stability, it is effective to use “frame support-rock bolts” system for the mining-geological conditions of the Western Donbass. And for the cost-effective use of the resources, it is possible to decrease the special section profile with less amount of metal.

3.3 Economic evaluation and benefit of the system

Use of such a support system which strengthens a slightly metamorphosed rock massif reliable enough, allow to reuse the development drift and have more benefits, such as possibility to reduce the profile of substitutable special section.

During calculations of the elements of the rock bolts-frame support system, the cost of 1 meter of the drift supporting makes:

Element	Amount	Price (UAH)	Total (UAH)
Rock bolt 22/2.4	4	200	800
Rock bolt 22/6	2	675	1350
Ampoule 300	6	15	90
Ampoule 600	12	24	288
Locknut M 22	6	12	72
Grummet ø 200	6	8	48
SSS-22 (250 kg)	1	6430	6430
Total sum			9078

Table 5. The expanses for support system’s elements

The cost of one long meter of development working supported by rock bolts and frame support (SSS-22) with the setting increment of rock bolts of 1 meter only for materials:

$$C_{sup} = C_{rf} + C_{b\text{-}u} + C_{ct},\ \text{UAH};$$

где: C_{rf} – the cost of the rock bolts-frame support;

$C_{b\text{-}u}$ – the cost of bolting-up (425 UAH/meter);

C_{ct} – the cost of crown tree (680 UAH);

$$C = 9078+425+680 = 10183 \text{ UAH (343 EUR)};$$

According to the rate of exchange of the National Bank of Ukraine as on the 15th of April, 1 EUR = 29.7 UAH.

Due to the stability of the working, the SSS profile can be reduced from SSS-22 to SSS-19, which will save 780 UAH on the cost of the material per meter of roof support, which will amount to 508560 UAH (more than 17000 EUR) for the length of the development drift 652 meters.

Due to the proposed support system, the working can be reused and costs can be saved on the retimbering of the working and footwalling.

Expenses for the retimbering:

$$R_{ret} = c_1 \cdot (11{,}97 \cdot \ln \Delta \cdot S + 168{,}62 \ln \Delta - 17{,}64 \cdot S - 248{,}5)K, UAH/m$$

where c_1 – a coefficient that takes into account general mine loses (1.17);

S – internal cross-sectional area of the working (11.7 m^2);

Δ – yielding of support (300 mm);

K – a coefficient that taking into account the influence of the support elements;

R_{ret}= 11,7 * (11,97 * ln 300*11,7 + 168,62 * ln 300 – 17,64 * 11,8 – 248,5) * 2,04 = 31119 UAH (1047 EUR),

Expenses for the development drift retimbering:

C_{ret} = 652 * 31119 = 20289588 UAH or 682644 EUR;

where the length of the development drift is 652 meters.

Expenses for the footwalling:

$$R_{fw} = c_1 \left(0{,}34 \cdot h + 0{,}16 \ln(\frac{S}{12}\right) \cdot h)K, UAH/m$$

where h – the depth of the footwalling,

K- is a coefficient that takes into account the influencing factors;

R_{fw}= 11,7 * (0,34 * 500 - 0,04*500) * 4,13 = 7248 UAH/m (244 EUR);

Expenses for the footwalling of the development working:

C_{fw} = 652*7248 = 4725696 UAH (159 120 EUR);

The total savings due to the proposed support system:

C_{total}= $C_{SSS\text{-}19}$+C_{ret}+C_{fw} = 508560 + 20289588 + 4725696 = 25523844 UAH (860000 EUR);

As a consequence, the reviewed support system will allow:

• to reduce industrial injuries caused by rock collapse during mine operations;

• to increase the pace of workings;

• to reduce the expanses for material and labor resources for support workings;

• significantly reduce the cost of retimbering, workings protection and footwalling;

• reduce the metal consumption of the support.

CONCLUSIONS

In the master's thesis, it is carried out the research of stress-strain state of the rock massif and support system, depending on the changing of support elements. In order to solve the problem posed by research of changes in the stress-strain state of the rock massif, several studies were carried out by computer simulations of real mine conditions. It was developed a rational operation of the frame support-rock bolts system that is characterized by a minimum pressure on the frame support in the slightly metamorphosed rock massif. Also, it was:

- investigated the modern method of studying of mechanical properties of rocks by simulation methods to solve the problems of the rock massif stability control;
- set the influence of strength characteristics and deformation characteristics of rocks on the stress-strain state of the massif;
- -performed a substantiation of a simulation model that represents geomechanical behavior of the slightly metamorphosed rock massif;
- performed a substantiation of the support system “frame support-rock bolts” of the rock massif;
- performed a substantiation of the parameters of development drift and its support, that reflect in the most efficient way mining technological conditions for their maintenance;
- method of conducting a simulation experiment is developed, which includes 3 consecutive stages of calculating the SSS of the system and its analysis.

ВИСНОВКИ

В магістерській роботі проведено дослідження НДС гірського масиву та системи кріплення в залежності від зміни елементів кріплення. Для вирішення завдань дослідження коливань напружено-деформованого стану гірського масиву було розроблено декілька комп'ютерних моделювань з реальними шахтними характеристиками. Була розроблена раціональна модель кріплення виробки за допомогою рамо-анкерної системи, яка характеризується мінімальним тиском на рамне кріплення в слабометаморфічному масиві. Також було:

- дослiджено сучасний метод дослідження механічних властивостей порід методами моделювання для вирішення завдань керування стійкістю слабометаморфічним масивом;
- встановлена закономірність впливу характеристик міцності та деформаційних характеристик порід на напружено-деформований стан масиву;
- виконано обґрунтування математичної моделі, яка відображає геомеханіку поводження слабометаморфічного масиву;
- виконано обґрунтування "рамно-анкерної" системи кріплення гірського масиву;
- виконано обґрунтування параметрів пластових підготовчих виробок та їх кріплення, які найбільш повно відображаюсь горно технічні умови для їх підтримання;
- розроблена методика проведення обчислювального експерименту, яка включає 3 послідовних етапи розрахунку НДС системи і його аналіз.

List of references

1.URL: http://www.mbendi.com/indy/ming/coal/as/ua/p0005.htm, accessed on 24th November, 2019.

2. URL: https://energo.dtek.com/business/coal_industry/, official D.TEK website, accessed on 12th of December, 2019.

3. Бондаренко В.І., Ковалевська І.А., Симанович Г.А., Вівчаренко О.В., Малихін О.В., Гусєв О.С. Геомеханіка навантаження і розрахунок параметрів кріпильної й охоронної систем підготовчих виробок шахт Західного Донбасу: монографія/Бондаренко В.І, Ковалевська І.А., Симанович Г.А., Вівчаренко О.В., Малихін О.В., Гусєв О.С. – Дніпропетровськ: ТОВ «ЛізуновПрес», 2014.- 228 с. (російською мовою).

4. Геомеханічні процеси у породних масивах: Монографія/ О.М. Шашенко, Т. Майхерчик, О.О. Сдвижкова. – Дніпропетровськ: Національний гірничий університет (Дніпропетровськ, Україна), 2005.- 319 с.- Рос. мов.

5. Сергеєв Є.М., Голодковська Г.А., Зіангіров Р.С. та ін., Грунтоведення: монографія: видавництво МГУ, 1983.-392с.

6. Society of Civil Engineers, Journal of Soil Mechanics and Foundations Division 97: Serafim JL, Pereira JP: Considerations of the Geomechanics Classification of Bieniawski. In: LNEC (ed), International Symposium on Engineering Geology and Underground Construction, Lisbon, 1983.

7. N. Barton, F. Loset, R. Lien, J. Lunde: Application of Q-system in Design Decisions Concerning Dimensions and Appropriate Support for Underground Installations, International Conference on Sub-Surface Space, Rock Store, Stockholm, 1980, Vol. 2, 553-561.

8. Hoek E., Brown E.T. Practical estimates of rock mass strength. International Journal of Rock Mechanics and Mining Sciences, Volume 34, Issue 8, 1997.- 1165-1186p.

9. https://en.wikipedia.org/wiki/Shear_modulus, accessed on 12th of November, 2019.

10. ISRM (1975): Commission on terminology, symbols and graphic representation, International Society for Rock Mechanics (ISRM).

11. Ільюшин А.А., Ленський В.С. Опір матеріалів: монографія/ Ільюшин А.А., Ленський В.С.-Москва: Фізматліт, 1959. – 373с.

12. Механіка гірських порід: Навч.посібник/ О.М. Шашенко. – Дніпропетровськ: Національна гірнича академія України, 2002. – 302с.

13. http://rudana.in.ua/economic_evaluation_of_developments.htm, accessed on 14th of November, 2019..

14. Farmer, I. W., Kemeny, J. M., & Mcdoniel, C. (1991). Analysis of rock fragmentation in bench blasting using digital image processing. 1037-1042. Paper presented at 7th International Society for Rock Mechanics and Rock Engineering Congress 1991, Aachen, Germany.

15. Goodman R.E., Van T.K., Heuze F.E. (1969). The measurement of rock deformability in boreholes. Proc. 10th U.S. Symp. on Rock Mech., Austin, TX, 523-555.

16. L. Zhang, Engineering Properties of Rocks, 2nd Edition, Butterworth-Heinemann, 2016.-394p.

17. The design process in rock engineering, Z.T. Bieniawski, Rock mechanics and Rock Engineering, 17, 1984. -183-190.

18. Brown E.T., 1993. The nature and fundamentals of rock engineering. In: Hudson J.A. (Ed.), In: Comprehensive Rock Engineering – Principle, Practice and Project, vol. 1. Pergamon, Oxford, UK, pp. 1-23.

19. ASTM D1194-94, Standard Test Method for Bearing Capacity of Soil for Static Load and Spread Footings (Withdrawn 2003), ASTM International, West Conshohocken, PA, 1994, www.astm.org.

20. A.J. Barry, O.B. Nair. In Situ Tests of Bearing Capacity of Roof and Floor in Selected Bituminous Coal Mines. BuMines RI 7406, 1970, 20 pp.

21. R. Hou, K. Zhang, K. Sun, R.P. Gamage, Discussions on Correction of Goodman Jack test, Geotechnical Testing Journal, Vol. 40, No. 2, 2017, pp. 199-209.

22. The complete ISRM Suggested Methods for Rock Characterization, Testing and Monitoring: 1974-2006, Edited by R. Ulusay, J.A. Hudson, (Methods for determining in sity deformability in rock, ISRM, 1979).

23. Danijela Marčić, Anita Cerić & Meho Saša Kovačević (2013) Selection of a Field Testing Method for Karst Rock Mass Deformability by Multi Criteria Decision Analysis, Journal of Civil Engineering and Management, 19:2, 196-205.

24. Suggested Methods for Deformability Determination Using a Large Flat Jack Technique, ISRM. Commission on Testing Methods, Issued: I.S.R.M., 1986., 8 pp.

25. A. Palmstrom, R. Singh.The deformation modulus of rock masses – comparison between in situ tests and indirect estimates. Published in: Tunneling and Underground Space Technology, Vol. 16, No.3, 2001, pp. 115-131.

26. ASTM D4719-00, Standard Test Method for Prebored Pressuremeter Testing in Soils, ASTM International, West Conshohocken, PA, 2000

27. R.J. Mair, D.E. Wood, Pressuremeter testing: Methods and Interpretation, CIRIA Report, 1978, Butterworth, UK.

28. The European Committee for Standardization and Technical Committee ISO/TC 182, Geotechnical investigation and testing, Field testing, Part 3: Standard penetration test, ISO 22476-3, 1st Edition, 2005.

29. ASTM D1586 / D1586M-18, Standard Test Method for Standard Penetration Test (SPT) and Split-Barrel Sampling of Soils, ASTM International, West Conshohocken, PA, 2018.

30. Australian Standards AS 1289.6.3.1 – 2004.

31. Boldyrev G.G.,Testing soil by penetration methods, part 1, Инженерные изыскания, 2010, ноябрь. – 22-30с.

32. Сейсмическое зондирование и определение сорбционной метаноемкости углепородного массива шахты «Краснолиманская» / В.А. Васильковский, В.Г. Гринев, М.М. Довбнич, Я.В. Мендрий // Физико-технические проблемы горного производства: Сб. научн. тр. — 2016. — Вип. 18. — С. 15-18. — Бібліогр.: 18 назв. — рос.

33. G. Sanglerat, The penetrometer and soil exploration, Elsevier, Amsterdam, 1st Edition, 1972. – 464 pp.

34. Гречухин В.В., Геофизические методы исследования угольных скважин, М.: Недра, 1965. – 467с.

35. http://www.solgeo.it/en/cross-hole-e-down-hole-sismici/, SolGeo Geophysical solutions, accessed on 26th of November, 2019.

36. Луазон Р., Фош П., Буайе А, Кокс: Москва: Металлургия, 1975. - 520с.

37. Васильчук Ю.А., Вознесенский Е.А., Голодковская Г.А., Зиангиров Р.С., Королев В.А., Трофимов В.Т., Грунтоведение, Издание 6, МГУ, Москва, 2005. – 1024с.

38. Всероссийский научно-исследовательский институт стандартизации материалов и технологий, ГОСТ 15489.2-2018. Угли каменные. Метод определения коэффициента размолоспособности по Хардгрову..

39. Плачкова С.Г., Плачков И.В., Энергетика. От огня и воды к электричеству, 1 издание, 2012, http://energetika.in.ua/ru/

40. Бондаренко В.І., Ковалевська І.А., Симанович Г.А., Снігур В.Г. Експериментальні дослідження здиманні порід підошви підготовчих виробок на пологих пластах Донбасу: монографія/ Бондаренко В.І., Ковалевська І.А., Симанович Г.А., Снігур В.Г. – Дніпропетровськ: ТОВ «ЛізуновПрес», 2014. – 224с. (російською мовою).

41. M. Kononenko, O. Khomenko, A. Sudakov, S. Drobot, Ts. Lkhagva, Numerical modelling of massif zonal structuring around underground working, Mining of Mineral Deposits, Volume 10, Issue 3, 2016.- pp. 101-106.

42. Fomychov, V., Pochepov, V., Fomychova, L., & Lapko, V. (2017). Computational model for evaluating the state of geomechanical systems during computing experiments. Mining of Mineral Deposits, 11(1), 100-105.

43. M. Toderas, R. Moraru, C. Danciu, Finite element method applied in mine pressure computation within the context of rock massif- support system interaction, Mining of Mineral Deposits, Volume 13, Issue 1, 2019. – pp. 39-48.

44. A. Olovyannyy, V. Chantsev, Simulation of sample testing under compression with the help of finite-element model of rocks being broken, Mining of Mineral Deposits, Volume 12, Issue 3, 2018. – pp. 9-21.

45. D. Sainsbury, Continuum and Distinct element modelling in geomechanics: Proceedings of 2nd international FLAC/DEM Symposium, Melbourne, Australia, 2011.

46. A.P. Gospodarnikov, M.A. Zatsepin, Y.N. Vykhodtsev. Mathematical modeling of applied problems of rock mechanics and rock massifs, Записки Горного института, Т.207, 2014, 217-221с.

47. ASTM D4428 / D4428M-14, Standard Test Methods for Crosshole Seismic Testing, ASTM International, West Conshohocken, PA, 2014.

48. ASTM D 409/ D409M-16, Standard Test Method for Grindability of Coal by the Hardgrove-Machine Method, ASTM International, West Conshohocken, PA, 2016.

49. ASTM D2850-15, Standard Test Method for Unconsolidated-Undrained Triaxial Compression Test on Cohesive Soils, ASTM International, West Conshohocken, PA, 2015.

50. ASTM D4767-11, Standard Test Method for Consolidated Undrained Triaxial Compression Test for Cohesive Soils, ASTM International, West Conshohocken, PA, 2011.

51. The Measurement of Soil Properties in the Triaxial Test, A.W. Bishop, D.J. Henkel, 1962.

52. Brenner R.P., Garga V.K., Blight G.E. (1997). Shear Strength Behavior and the Measurement of Shear Strength in Residual Soils. Mechanics of Residual Soils, ed. G.E. Blight, Chapter 9. A.A. Balkema, Rotterdam.

53. Калиткин, Н.Н. Численные методы [Текст] / Н.Н. Калиткин. – М.: Наука, 1978. – 197 с.

54. Бондаренко В.І., Ковалевська І.А., Симанович Г.А., Мартовицький А.В., Копилов О.П. Методи розрахунку переміщень і зміцнень приконтурних порід гірничих виробок шахт Західного Донбасу**.** Монографія. – Дніпропетровськ: ТОВ «ЛізуновПрес», 2010. – 330 с.

55. Cundall P.A., Strack O.D.L. A discrete numerical model for granular assembles // Geotechnique. 1979. – №1. – с. 47-65.

56. Крауч С., Старфилд А. Методы граничных элементов в механике твердого тела. – М.: Мир, 1987. – 236с.

57. Фадеев А.Б. Метод конечных элементов в геомеханике. –М.: Недра, 1987. – 236с.

58. Алямовский А.А. SolidWorks/COSMOSWorks. Инженерный анализ методом конечных элементов. – М.: ДМК Пресс, 2004. – 432 с.

59. https://www.ansys.com , accessed on 20th of January, 2020.

60. https://www.plm.automation.siemens.com/global/ru/products/nx/, accessed on 20th of January, 2020.

61. https://www.3ds.com/products-services/simulia/products/, accessed on 21st of January, 2020.

62. https://www.solidworks.com, accessed on 21st of January, 2020.

63. Усаченко Б.М. Охрана подготовительных выработок глубоких горизонтов шахт Западного Донбасса [Текст] / Б.М. Усаченко, В.Я. Кириченко, А.В. Шмиголь. – М.: ЦНИЭИуголь, 1992. – 168 с.

64. Зборщик, М.П. Охрана выработок глубоких шахт в зонах разгрузки [Текст] / М.П. Зборщик, В.В. Назимко. – К.: Техника, 1991. – 248 с.

65. Спивак А.И., Попов А.Н. Разрушение горных пород при бурении скважин. - М.: Недра, 1994. - 261с.

66. https://www.itascacg.com/software/flac3d, accessed on 25th of January, 2020.

67. Бондаренко В.І., Чередниченко Ю.Я., Ковалевська І.А., Симанович Г.А., Вівчаренко О.В., Фомичов В.В. Геомеханіка взаємодії анкерного та рамного кріплення гірничих виробок в єдиній вантажонесучій системі. Монографія. – Дніпропетровськ: «ЛізуновПрес», 2010. – 174 с. – Російською мовою.

68. Амусин Б.З., Фадеев А.Б. Метод конечных элементов при решении задач горной геомеханики. – М.: Недра, 1975. – 143 с.

69. Методы и средства решения задач горной геомеханики / Г.Н. Кузнецов, К.А. Ардашев, Н.А. Филатов и др. – М.: Недра, 1987. – 248 с.

70. Бондаренко В.І., Ковалевська І.А., Симанович Г.А., Барабаш М.В., Гусєв О.С. Взаємодія вантажаносучих елементів кріпильної системи виїмкових виробок «масив-рама-анкер»: монографія/ Бондаренко В.І., Ковалевська І.А., Симанович Г.А., Барабаш М.В., Гусєв О.С. – Дніпропетровськ: «Літограф», 2015. – 214 с. (російською мовою).

71. Volodymyr Bondarenko , Hennadii Symanovych , Mykhailo Barabash , Oleksandr Husiev , Ildar Salieiev (2020): Determining patterns of the geomechanical factors influence on the fastening system loading in the preparatory mine workings, Mining of Mineral Deposits, Vol. 14, Issue 1, p.44-50.

72. Yu. Khalymendyk, A. Baryshnikov (2016): Substantiation of cable bolts parameters for supporting mine workings in conditions of laminated rocks, Mining of Mineral Deposits, Vol.10, Issue 1, p. 9-15.

73. Tadolini, S., & McDonnell, J. (2010). Cable bolts an effective primary support system. In Proceedings of the 29th International Conference on Ground Control in Mining, Morgantown, WV (pp. 75-83).

74. Bondarenko, V., Kovalevs'ka, I., & Fomychov, V. (2012). Features of carrying out experiment using finite-element method at multivariate calculation of "mine massifcombined support" system. Geomechanical Processes during Underground Mining: School of Underground Mining 2012, 7-13.

75. Технологія підземної розробки рудних родовищ: підручник / О.Є. Хоменко, М.М. Кононенко, М.В. Савченко; М-во освіти і науки України, Нац. техн. ун-т «Дніпровська політехніка». – Дніпро: НТУ «ДП», 2018. – 450 с.

76.Сборник инструкций к правилам безопасности в угольных шахтах. - Том 2 - Киев 2003. – 480 с.

77. Правила безпеки у вугільних шахтах / НПАОП 10.0-1.01-10 / Редакція журналу «Охорона праці». – Київ, 2010. – 432 с.