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New Techniques and Technologies in Mining

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New Techniques and Technologies in Mining – Bondarenko, Kovalevs'ka & Dychkovs'kyy (eds)
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New Techniques and Technologies in Mining – Bondarenko, Kovalevs'ka & Dychkovs'kyy (eds)
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Preface

Mining is the foremost source of minerals that all countries find essential for maintaining and improving their standards of living. Mined materials are needed to construct roads and hospitals, to build automobiles and houses, to make computers and satellites, to generate electricity, and to provide many other goods and services that consumers enjoy. The high tech industries and even the better known resource industries are all dependent, in some way, on the mining industry.

But exploring, extraction and processing of minerals require big material and labour costs and there is a large number of acute problems to face; such as: environment and water pollution, worsening of mining-geological conditions, depletion of minerals that can be extracted only by conventional methods, rock pressure manifestation, big depths of the deposits and transportation of the minerals on the surface.

In order to find modern solutions there is a large number of scientists and engineers all over the world dedicating their researches to most current problems and inventions of innovative technologies and techniques in mining. Some of the most important results of such researches are presented in this book and cover the following topics: management of strain and stress state of the massif, underground coal gasification, substantiation of rational parameters of various types of support, ventilation in underground openings, design of mine workings and other vital questions.

The primary aim of "School of Underground Mining" is exchange of experience, transfer of new progressive technologies, consolidation of the efforts concerning saving and development of scientific trends, good traditions and worthy status of a mining engineer in society.

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Efficiency of water supply regulation principles

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ABSTRACT: The paper deals with energy efficiency of water supply regulation in different applications: water supply, heating and cooling systems etc. Each case is characterized by a set of requirements to supply and head pressure level. Two major approaches for supply regulation are compared: throttling and pump speed regulation. It is shown that variable speed electric drive application is not always expedient and in some cases can bring no significant cost benefit. Parallel operation of several pumping units is also examined. Recommendations for pump installations' parameters selection, such as regulation range, are given.

1 INTRODUCTION

Energy and resource saving programmes are considered now to be key factors for word's industry due to constant growth of prices for fuels and energy resources. Specific energy intensity of gross domestic product in Ukraine is significantly higher than in countries of Central Europe thus making these problems more urgent. However, the awareness of energy management specialists of modern energy saving techniques and principles in Ukraine is still low.

Electric drive is a major consumer of electric power in industry and municipal engineering with its share up to 65%. Electric installations with continuous duty cycle and varying productivity posses the largest potential for energy saving. The prevailing mechanisms in this class are installations with so called parabolic speed-torque characteristic (centrifugal pump, screw propeller etc). Such installations are as a rule equipped with induction motors with squirrel cages. They are widely used in industry and agriculture and municipal enterprises. The electric drive of these installations is mainly uncontrolled and therefore to regulate productivity throttling and bypassing principles are applied. Unfortunately lowering productivity is not equal to lowering electric energy consumption. So this control principle meets technological demands but does not account energy efficiency of water transportation.

Thereby, to justify the expediency of variable speed drive application it is necessary to estimate the cost benefits of this solution.

2 EXISTING APPROACHES ANALYSIS

Selection of proper production control principle is to be made according to technology requirements and specific performances of equipment installed and its economical efficiency. Incorrect evaluation of economical performances of certain control principle leads to introduction of improper technical solution and thus low production efficiency.

The application of variable speed drive itself cannot provide significant power consumption decrease. Energy saving requires thorough evaluation of technological and technical factors concerning production regulation.

The production level of water supply and utilization facilities tends to vary in wide range during operation. The main controlled parameter at the pumping station is the pressure at the discharge line or at the control point (step-up pumping stations of municipal water supply system). In some cases it is water level or its flow (supply) rate.

It should be taken into account that when using uncontrolled water pumps the excessive pressure can occur under low production rate. Excessive pressure in pipeline causes electric power loss. To minimize such losses it is necessary to achieve maximal efficiency by mutual adjustment of pump's mechanical parameters and the entire system.

Energy losses can also be decreased by proper pipeline processing measures – interior surface processing, elimination, or at least, minimization of elbows and narrowings in the line (Goppe 2008). However, these measures most often cannot be applied for existing water supply facilities. It should be taken into account while designing new systems. Meanwhile, application of variable speed drives which become more and more available (Pivnyak & Volkov 2006) is possible for newly developed as good for existing systems' modernization.

The simplified water supply scheme can be presented as it is shown at Figure 1.

The output of pumping unit is supply (flow) rate

Q_{pump} and head pressure H_{pump} (Geyner, Dulin & Zarya 1991). Because of extra pressure H_{pump} the head pressure in the mainline grows from H_0 up to H_p. Because pressure drops on the sealing and stops valves and the filter on ΔH_1 ,the main pressure decreases down to H_{CV}. The control valve CV determines pressure drop depending on the main parameters H_C and Q_C control principle.

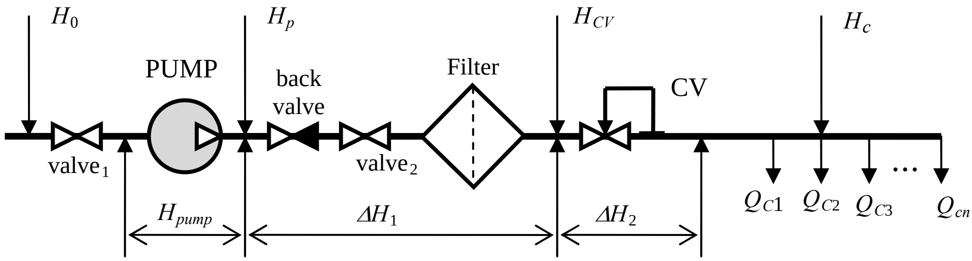

Figure 1. Simplified water supply scheme.

The head pressure H_C and flow consumption Q_C can be controlled according to the following principles (Geyner, Dulin & Zarya 1991):

1) $H_C = const$, $Q_C = var$;
2) $H_C = var$, $Q_C = var$;
3) $H_C = var$, $Q_C = const$.

Let us examine each case from the point of energy efficiency.

2 REGULATION UNDER CONSTANT PRESSURE ($H_C = const,\ Q_C = var$)

Let's consider the first principle, when it is necessary to maintain constant pressure in the hydraulic network under varying water consumption (supply) $Q_C = Q_{C_1} + Q_2 + ... + Q_{C_n}$. This case is common for main pipelines which must provide necessary supply level for each consumer.

The control valve CV maintains constant head pressure H_C at required constant level by varying pressure drop ΔH_2 value. The output power of the pump unit is defined as

$$P_{pump} = k \cdot H_{pump} \cdot Q_C , \tag{1}$$

where k – certain proportional gain.

This power is used to provide necessary consumption and to cover losses $k \cdot (\Delta H_1 + \Delta H_2) \cdot Q_C$ and also to maintain constant pressure level $H_C - H_1$. Thus

$$P_{pump} = k \cdot (H_C - H_0 + \Delta H_1 + \Delta H_2) \cdot Q_C . \tag{2}$$

The electric power rate (consumed from the electric mains) can be determined via efficiency of the pump η:

$$P_{pump}, el = k \cdot (H_C - H_0 + \Delta H_1 + \Delta H_2) \cdot Q_C / \eta . \tag{3}$$

The Q-H curve of the pump is described as follows

$$H = A \cdot \omega^2 + B \cdot \omega \cdot Q + C \cdot Q^2 , \tag{4}$$

where A, B, C – coefficients; ω – angular speed of the pump's wheel.

When pump speed is constant its curve can be written as

$$H = A + B \cdot Q + C \cdot Q^2 , \tag{5}$$

where $A_1 = A \cdot \omega^2$, $B_1 = B \cdot \omega$, $C_1 = C$.

The hydraulic characteristic of the line is described by

$$H = H_0 + R \cdot Q^2 \tag{6}$$

where H_0 – static pressure (back pressure, or uplift pressure), R – line hydraulic pressure.

The operating mode of the pump unit is defined by the intersection point of pump (Figure 2, curve 1) and hydraulic line (Figure 2, curve 2) characteristics. The intersection point “1” is the ideal joint operation calculation point for the pump and the line. At this point the rated supply Q_1 is provided under required pressure $H_1 = H_{pump}$ and maximal efficiency.

When water consumption decreases down to Q_2 level, the operating point moves to the position ‘2’. It is caused by the rise hydraulic resistance because of the closing of consumers’ valves. The head pressure rises up to H_2 value, causing the control valve CV to increases the pressure drop ΔH_2 in order to provide the required line pressure H_C. Meanwhile, the pumping unit keeps operating with pressure H_2. This is an obvious lack of throttling control.

The pressure can also be decreased by pump wheel speed regulation. In this case, the curve of the pump will move to ‘1’ position, and the operating point will move to the ‘3’ position (Figure 2). However, under these conditions, the required supply Q_2 is not provided. The supply will be lower and consumers will open their valves, so decreasing line’s hydraulic resistance. After several iterations, a new operating point ‘4’ will be set.

So, the decrease of the supply in the line to ΔQ value by throttling causes the increase of pressure on ΔH and, accordingly, variation of the pump power to

$$\Delta P_d = k \cdot (H_1 Q_1 - H_2 Q_2). \tag{7}$$

And same decrease of water supply by means of speed regulation leads to the change of power on

$$\Delta P_\omega = k \cdot (H_1 Q_1 - H_1 Q_2). \tag{8}$$

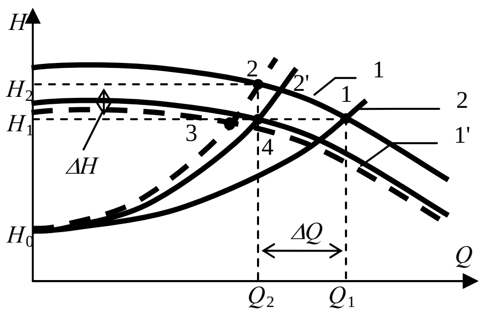

Figure 2. Regulation under constant pressure.

Thus we can asses regulation efficiency comparing power values calculated by (7) and (8) relative to base value $P_1 = k \cdot H_1 Q_1$. So

$$e_d^* = \frac{\Delta P_d}{P_1} = \frac{H_1 Q_1 - (H_1 + \Delta H)(Q_1 - \Delta Q)}{H_1 Q_1} = \\ = \Delta Q^* - \Delta H^* \left(1 - \Delta Q^*\right), \tag{9}$$

$$e_\omega^* = \frac{\Delta P_\omega}{P_1} = \frac{H_1 Q_1 - H_1 (Q_1 - \Delta Q)}{H_1 Q_1} = \Delta Q^*, \tag{10}$$

where $\Delta Q^* = \Delta Q / Q_1$, $\Delta H^* = \Delta H / H_1$.

Formulas (9) and (10) prove that speed regulation is always more efficient than throttling, because under normal operation it is always $\Delta H^* > 0$, $\Delta Q^* < 1$.

Thus

$$\Delta e^* = e_\omega^* - e_d^* = \Delta H^* \left(1 - \Delta Q^*\right) > 0. \tag{11}$$

Lets transform (11) to

$$\Delta e^* = \left(\frac{\Delta H^*}{\Delta Q^*}\right)\left(1 - \Delta Q^*\right)\Delta Q^*. \tag{12}$$

The equation (12) shows that the value of Δe^* which determines relative efficiency of speed regulation principle in compare with throttling depends on ΔQ^* regulation range and $\Delta H^* / \Delta Q^*$ ratio at the new operating point. Let us use (5) to determine $\Delta H^* / \Delta Q^*$. Equation (5) in per units is

$$H^* = A_1^* + B_1^* \cdot Q^* + C_1^* \cdot Q^{*2}, \tag{13}$$

where $H^* = H/H_1$, $Q^* = Q/Q_1$ $A_1^* = A_1/H_1$, $B_1^* = (B_1 Q_1)/H_1$, $C_1^* = \left(C_1 Q_1^2\right)/H_1$.

Taking increments of (13) we obtain

$$\left(\frac{\Delta H^*}{\Delta Q^*}\right) = C_1^* \Delta Q^* - \left(B_1^* + 2C_1^*\right) \tag{14}$$

And substituting (14) in (12) we estimate efficiency

$$\Delta e^* = \left[C_1^* \Delta Q^* - \left(B_1^* + 2C_1^*\right)\right] \cdot \left(1 - \Delta Q^*\right) \cdot \Delta Q^*. \tag{15}$$

Formula (15) shows that speed regulation efficiency relative to throttling Δe^* depends on regulation range relative to the given point ΔQ^* and pump’s Q-H curve B_1^* and C_1^*

The formula estimates useful power saving under speed regulation. Losses in the pump are not taken into account. However, it is known that pumps efficiency significantly depends on supply, taking maximum value at the rated operating point. That is why in order to estimate energy saving it is necessary to introduce $\eta(\Delta Q^*)$ in (15). Figure 3 shows corrected dependency (15) for several pumps (pumps data are given at the Table 1 (Popov 1990)).

Table 1. Pumps characteristics.

#	Pump type	Q-H curve equation $H = A_1 + B_1 \cdot Q + C_1 \cdot Q^2$	Efficiency equation
1	CSP* 38-44-220, 2950 rpm, 38 m^3 / hr, 44 m	$26.8+0.168Q-0.00787Q^2$	$0.04Q-0.00073Q^2+0.000003Q^3$
2	CSP 180-76-880, 2950 rpm, 180 m^3 / hr, 76 m	$75+0.139Q-0.00098Q^2$	$0.0068Q-0.000017Q^2+0.017\cdot10^{-6}Q^3$
3	CSP 180-500-900, 2950 rpm, 180 m^3 / hr, 500 m	$105.5+0.096Q-0.0007Q^2$	$0.0085Q-0.0000276Q^2+0.014\cdot10^{-6}Q^3$
4	CSP 850-240-960, 1450 rpm, 850 m^3 / hr, 240 m	$126.2+0.035Q-0.000049Q^2$	$0.237\cdot10^{-2}-0.024\cdot10^{-4}\cdot Q^2+0.00062\cdot10^{-6}Q^3$

* CSP stands for “centrifugal sectional pump”, correspondent soviet abbreviation is CNS.

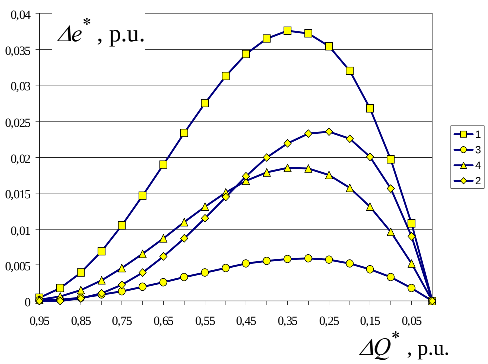

Figure 3. Energy efficiency under regulation by pump speed.

The analysis of this section shows the following.

1. When head pressure is to be maintained, power (energy) benefits from introduction of speed regulation principle relative to throttling significantly depends on pump’s Q-H curve stiffness. The higher stiffness the less efficiency is (look formula (12)). Figure 3 shows that benefit from supply regulation by speed does not exceed 3.7% in the entire regulation range.

2. Maximal efficiency under speed regulation falls within 10…45% supply range relative to operating point. Thus deep regulation (supply less then 45% of the rated value) so and “shallow” (regulation range within 10%) is not expedient. Measures for variable speed drive introduction will not provide cost benefits.

3. When operating at Q-H curve shifts to lower supply zone (to the left), efficiency of the pump will be even less due to higher stiffness of the curve, despite on the fact that efficiency of the motor shifts the same direction under speed regulation.

3 REGULATION UNDER VARIABLE PRESSURE AND SUPPLY

Let us consider supply regulation $H_C = var$, $Q_C = var$ i.e. when there is no requirement for keeping pressure constant. It is peculiar to cases when water supply is stipulated by technology, like refrigerating systems, irrigation and so on.

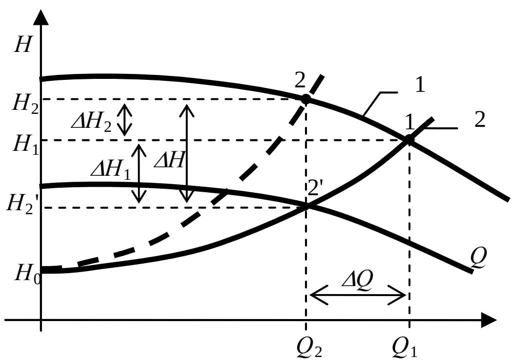

Figure 4. Regulation under variable pressure and supply.

This case the decrease of supply down to Q_2 value by means of speed regulation lowers pump's Q-H curve and shifts operating point from "1" to "2'" position (Figure 4).

Still, throttling shifts the operating point to "2" position.

Energy benefits of speed regulation principle relative to throttling is defined by (11), where the difference is calculated by the following expression

$$\Delta H^* = \Delta H^*_{21} + \Delta H^*_{12}, \tag{16}$$

where $\Delta H^*_{21} = \Delta H_{21} / H_1$ – relative pressure increase in case of throttling; $\Delta H^*_{12} = \Delta H_{12} / H_1$ – relative pressure decrease as a result of pump speed lowering.

It is obvious that ΔH^*_{21} and corresponding efficiency is defined by (14) and (15). The value of ΔH^*_{21} can be estimated by water line's equation in per units

$$H^* = H^*_0 + R^*_1 \cdot Q^*, \tag{17}$$

where $H^*_0 = H_0 / H_1$, $R^*_1 = R_1 Q_1^2 / H_1$.

The expression in incremental form relative to "1" point, considering that $H^*_1 = 1$, $Q^*_1 = 1$:

$$\Delta H^*_{12} = R^*_1 \Delta Q^* \left(2 - \Delta Q^*\right), \tag{18}$$

where $R^*_1 = 1 - H^*_0$.

Substituting (14) and (18) in (16) and find

$$\Delta H^* = \left\{\left[C^*_1 \Delta Q^* - \left(B^*_1 + 2C^*_1\right)\right] + \right.$$
$$\left. + \left[R^*_1\left(2 - \Delta Q^*\right)\right]\right\}\Delta Q^*. \tag{19}$$

From (16) and (19) we determine

$$\Delta e^* = \left[\left(R^*_1 - C^*_1\right)\left(2 - \Delta Q^*\right) - B^*_1\right] \cdot \left(1 - \Delta Q^*\right) \cdot \Delta Q^* \tag{20}$$

In order to examine (20) let us obtain $d\left(\Delta e^*\right) / d\left(\Delta Q^*\right)$ and find its roots:

$$\frac{d\left(\Delta e^*\right)}{d\left(\Delta Q^*\right)} = D^* - 2\left(3R^* - B^*_1\right)\Delta Q^* + 3R^* \Delta Q^{*2} = 0. \tag{21}$$

$$\Delta Q^*_m = \frac{\left(R^* + D^*\right) - \sqrt{R^{*2} - R^* D^* + D^{*2}}}{3R^*}, \tag{22}$$

where $R^* = R^*_1 - C^*_1$, $D^* = 2R^* - B^*_1$, $R^*_1 = 1 - H^*_0$.

Thus under supply level $1 - \Delta Q^*_m$ the speed regulation benefits will be maximal relative to throttling

$$\Delta e^*_{max} = \left[R^*\left(2 - \Delta Q^*_m\right) - B^*_1\right] \cdot \left(1 - \Delta Q^*_m\right) \cdot \Delta Q^*_m. \tag{23}$$

Equations (22) and (23) shows that Δe^*_{max} depends on relative static pressure H^*_0 and coefficients of pump's Q-H curve B^*_1 and C^*_1 for pump's rated speed.

Let us analyze the dependence of ΔQ^*_m and Δe^*_{max} on the mentioned parameters. It is assumed that they do depend on $R^* = R^*_1 - C^*_1$ (not on R^*_1). This assumption affects only factor R^*_1 range. Thus $R^* = 1 - H^*_0$, where H^*_0 variation range is shifted on C^*_1 value. The B^*_1 value represents stiffness of Q-H curve. The higher B^*_1 is, the higher stiffness is and the less efficiency of speed regulation we obtain.

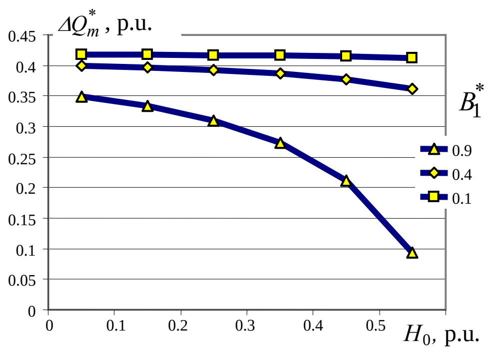

Figure 5. Q-H curves for various stiffness.

In the range for different $B^*_1 = 0.1; 0.4; 0.9$ trends of maximal efficiency were obtained (Figure 5).

Figure 5 shows that maximal efficiency is achieved under maintenance of the pump with no uplift pressure. And the increase of the Q-H curve stiffness dramatically lowers the speed regulation benefits relative to throttling. The figure also proves that maximal efficiency lies within $\Delta Q^*_m = 0.35 \ldots 0.42$ range. Higher ΔQ^*_m corresponds lower values of B_1^* and H_0^*. For B_1^* = 0, H_0^* = 0 we have ΔQ_m^* = 0.423. Substituting this in (23) we obtain $\Delta e_{max}^* = 0.385$.

So, for the water supply regulation under variable pressure, the following conclusions can be made.

1. Theoretical maximal efficiency (energy benefit) of water supply by means of pump's speed regulation relative to throttling is 38.5% of the power consumed at the pump's operating point. This efficiency is obtained for 42.3% regulation depth (range) relative to rated supply.

2. The uplift pressure increase significantly decreases speed regulation efficiency (Figure 6).

4 REGULATION UNDER CONSTANT SUPPLY

In the third case for some technologies it is necessary to maintain constant supply of $Q_C = const$, which is possible only by varying pressure $H_C = var$. The pumping unit can operate with constant power and supply stability can be provided with bypass system. Another, more efficient principle, is maintaining constant power by speed regulation (Figure 6). Supply stabilization, for example, in water heating system, requires the pump speed lowering.

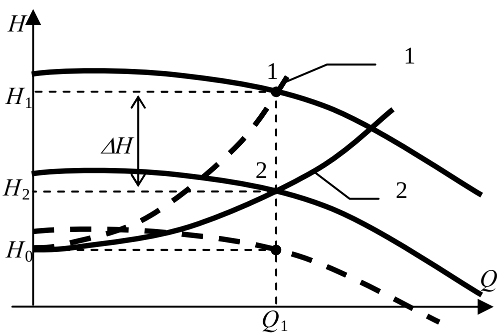

Figure 6. Q-H curves for regulation under various supply.

When consumers open their throttles the main Q-H curve shifts from "1" to the "2" position.

So, the efficiency of speed regulation is defined by

$$\Delta e^* = e_\omega^* - e_d^* = \Delta H^*, \tag{24}$$

where $e_d^* = \frac{\Delta P_d}{H_1 Q_1} = 0$,

$$e_\omega^* = \frac{\Delta P_\omega}{H_1 Q_1} = \frac{H_1 Q_1 - Q_1 (H_1 - \Delta H)}{H_1 Q_1} = \Delta H^*.$$

Formula (24) and Figure 7 shows that maximal benefit of pump speed regulation relative to throttling is limited by uplift pressure

$$\Delta e_{max}^* = 1 - H_0^*. \tag{25}$$

Thus when the pump is operating to the water line with no uplift pressure, the benefit of speed regulation is limited only by its stability in low supply range.

5 PARALLEL PUMPS CONNECTION

In some cases it is expedient to connect several pumping units for parallel operation in order to provide higher supply under given required pressure. Conclusions given above can be transferred for this case. For instance, supply regulation can be provided by simultaneous regulation of all connected units. However, it is not always expedient because this way implies installation of variable speed drives for all installation, which is expensive.

Let us analyze parallel operation of two pumps. Total supply, obviously, is defined as a sum of individual supplies

$$Q^* = Q_I^* + Q_{II}^*, \tag{26}$$

where $Q^* = Q/Q_1$, $Q_I^* = Q_I/Q_1$, $Q_{II}^* = Q_{II}/Q_1$, Q_I – total supply.

Let the relative regulation range be $\Delta Q^* = \Delta Q/Q_1$. Assuming that only one variable speed drive is installed (pump II), while the other one is a fixed speed drive. Then

$$1 - \Delta Q^* = Q_{I,y}^* + \left(Q_{II,y}^* - \Delta Q^*\right), \tag{27}$$

where $Q_{I,y}^*$, $Q_{II,y}^*$ – rated supplies of the pumps.

According to the previous statements, under constant hydraulic pressure it is reasonable to select maximal supply regulation range within λ of its rated supply, i.e. $\Delta Q_{max}^* = \lambda Q_{II,y}^*$. Then under required regulation range ΔQ^* it is necessary to install a pump with rated supply

$$Q_{II,y}^* = \frac{\Delta Q^*}{\lambda}. \tag{28}$$

The supply of the first pump is defined from

$$Q_{I,y}^* = 1 - \frac{\Delta Q^*}{\lambda}. \tag{29}$$

For example, if supply regulation range in the system with constant pressure does not exceed 10% then energy saving (benefit) of speed regulation is only 1.5% relative to throttling (Figure 3). Obviously, it is not wise to install variable speed drive in this case.

Two pumps with one of them equipped with variable speed drive can provide only 3.7% energy saving. And even this small value can only be obtained under wide regulation range – 40…50%. (Figure 7).

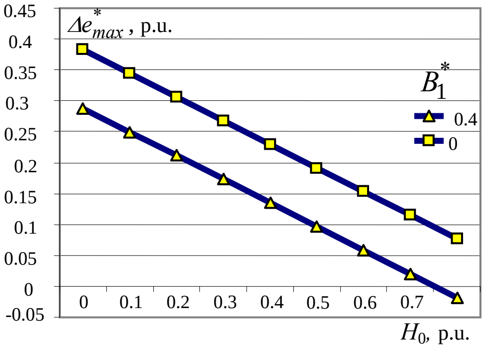

Figure 7. Efficiency under pumps' parallel operation.

These results can be transferred to the case when the pressure is not to be maintained. In this case maximal regulation depth for variable speed pump is to lie within 35…42% range relative to its rated supply (Figure 5). The increase of uplift pressure at the unregulated pump output is also to be taken into account.

For the case $H_C = var$, $Q_C = const$ there is no necessity to apply pump stations with several pumps. One powerful variable speed drive should be installed.

6 CONCLUSIONS

Theoretical maximal energy benefit from pump's speed regulation application relative to throttling principle is 4% under constant pressure and about 40% when pressure can be varied.

When two pumps operates in parallel, supply regulation range of one of then is to be 5…50% of its rated supply.

Energy saving due to variable speed drive application is defined by equivalent supply regulation range.

Installation of variable speed drive in water mains, where constant pressure must be maintained, is not a reasonable solution. Energy saving (benefits) of in this case does not exceed several per cents relative to throttling.

The operation of pumping stations must be organized in a way that each pump would operate at its maximal efficiency under given pressure regulation range. Application of several unregulated drive and one equipped with variable speed drive can be expedient in this case. The regulated pump is to be chosen from required regulation range.

The most beneficial is application of variable speed drive in case of necessity of constant supply.

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New Techniques and Technologies in Mining – Bondarenko, Kovalevs'ka & Dychkovs'kyy (eds)

Substantiation of design and installation technology of tubular rock bolts by explosive method

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ABSTRACT: Mechanism of interaction of a rock massif with rock bolt is developed and simplified tubular rock bolt construction is designed, manufacturing of which is combined with the installation in one process – blast of an explosive linear charge in the internal cavity of tubular shell located in a blast hole. Basic requirements of bolt support are substantiated and positive features of tubular bolt are noted.

1 INTRODUCTION

Experience of application of rock bolts for securing of underground workings stability testifies about considerable economical advantages of the given type of support that allows to drastically decrease material and labour costs, increase working efficiency and safety of works. This is achieved by involvement of rock massif, with help of a rock bolt, into the process focused on resistance against gravity forces, i.e. strength characteristics of rock massif are used (in a varying degree) and it is not necessary anymore to install material-intensive supports for maintenance of mine workings.

Mentioned advantages of a rock bolt gave rise to their wide usage in mining sectors of industries of many countries of the world. So, basic support for chambers' roof support (room-and-pillar mining method) in coal mines of USA is rock bolting with annual installation volume more than 60 million of bolts (Bondarenko 1997), including about 1.5 million of bolts that are being reused. Just on the polymetal mines of USA there are more than 7 million of bolts installed annually; at iron-ore mines of Lorraine there are more than 2 million bolts used annually; at copper mines of Poland there are 85…90% of the openings length is supported by bolts. At coal mines of France 1.5 million of bolts are installed annually, at that, a quarter of this volume is used as an independent support. About 50 thousand of bolts are installed at the mines of Swedish company "Karuna" monthly, 150 thousand of bolts – at mines of RSA. Bolt supports are widely used at mines of PRC (People's Republic of China). Rock bolting is used at more than 30 coal mines of Great Britain, at all mines of Canada, on many mines of Bulgaria, Germany, Italy, Norway and other countries.

Usage of rock bolting also contributes to increase of drivage rate by 30…40% (Kovalevska 2003). High level of mechanization of an opening bolting process (up to 65…85%) should be mentioned as well and also possibility of its partial automation. In addition, usage of bolting significantly decreases number of accidents that leads to increase of mining operations safety. For example, bolting at mines of Germany has allowed to decrease the number of accidents from 86 to 34 among 100 thousand of people, i.e. more than 2.5 times. Thus, given data testify about big effectiveness and availability of bolting use for support of mining openings.

2 CRITERIA OF BOLTS FUNCTIONING

There are more than 600 various designs of bolts known in domestic and foreign practice by the present time. Existing variety of rock bolts, as a rule, has its advantages and disadvantages which are analyzed both from view point of geomechanics of deformation of strengthened near-contour rock massif process and technical-economic indexes of their work. This analysis has a final aim to create a bolt design with most rational regime of interaction with rock walls of a blasthole and rock massif in whole with its relatively little cost (Kovalevska 2001). During research and critical analysis of modern structures of bolts there were formed three basic criteria that characterize application effectiveness of one or another structure of bolts for mine openings support.

From viewpoint of geomechanics of near-contour rock massif strengthening process, the first criterion demands to achieve rational deformation-load characteristics of a rock bolt. Near-contour rock massif strengthened by bolts is considered as a load-bearing structure in modern theories which receives load from the side of deformed rock massif and acts as a support of working. In order to effectively maintain mine openings, deformation-load characteristics of this kind of support must, first of all, correspond to value and nature of rock pressure manifestation, i.e. bearing capacity of strengthened by bolts rock layer must be not less than a load value from the side of rock massif, and its yield – not less than a predicted value of displacement of rock contour with given value of support rebuff reaction. Secondly, deformation-load characteristics of the strengthened rock layer must provide maximum degree of compatibility of separate layers and blocks deformation that compose unstable rocks in order to decrease load. In this connection, it was established by research method (Kovalevska 2006, 1999) that the most rational characteristic is the deformation-load characteristic of a support that reflects its work in mode of constant resistance with value of reaction, that is close to the predicted load on support caused by weight of rocks of unstable balance zone.

Thus, for effective support of an opening, rock bolt must form load-bearing iron-rock structure that would work in a mode of constant resistance (or close to it) with rebuff reaction and yield value equal to or excess predicted load and displacement of rock contour (Kovalevska 2005).

Based upon the above stated material, requirements of the first (geomechanical) evaluation criteria of effectiveness of a rock bolt structure were formed. First of all, in order to form strengthened rock layer of a high bearing capacity, rock bolt must have high reaction of resistance against pulling loads, because the bigger pressure load by bolt of the rock layer the bigger bearing capacity of the most strengthened layer with all the rest conditions being equal. Secondly, during realization of yield of strengthened rock layer, a bolt also must have required yield value, taking into account difference of displacements of an opening rock contour and rock at the deepened tip of the bolt (Kovalevska 2000). Thirdly, in order to achieve functionality of strengthened rock layer in constant resistance mode, the bolt itself also has to have deformation-load characteristic that reflects this mode, i.e. construction of the bolt has to provide its functionality in constant resistance mode or close to this mode.

Second criterion characterizes processability of manufacturing and installation of a bolt and requires the following:

– simplicity of structure;

– processability of installation of joints and components of a bolt;

– reliability of a bolt installation in a blasthole (Bulat 2002);

– little force of inserting the bolt into the blasthole and its anchoring in the blasthole;

– no necessity to design special equipment (if possible) for bolting;

Third criterion characterizes economical effectiveness of application of various bolt designs and states the following requirements:

– relatively low cost of a bolt manufacture;

– high working efficiency during bolts installation.

3 PRELIMINARY RESULTS

Theoretical basis for design development of a tubular bolt is the above considered conceptions of a bolt's work in a rock massif. At this, the main requirement for tubular bolt is the possibility of creation of iron-rock structure (beam, arc) with maximum possible degree of wholeness. Thus, there is a task to maximally decrease process of foliation and fracturing of rock structure during its deformation, liquidate earlier occurred lamination and to limit fracturing. Moreover, requirement for securing of specified installation firmness of a bolt in a blasthole is the secondary requirement compared with the major one – creation of solid iron-rock structure with required bearing capacity.

While functioning, under influence of external load and its own weight there are shear stresses occur in the beams, value of which substantially determines bearing capacity of considered constructions. At the same time, during process of deformation of a rock that contains mine working, intensive fracturing occurs in most of the cases and lamination along planes of weaknesses resulting in significant decrease and disappearance of tangential stresses along these axis, and layers and blocks that form rock structure deform as separate elements. And with that, bearing capacity of rock structure considerably decreases. The aim of a bolting is to prevent lamination and fracturing. Apparently, the biggest effect of rock strata fastening into solid structure is achieved when a rock bolt contacts with rock along its whole length. At every sector of contact due to cohesion forces with rock walls of the blasthole, the bolts prevent lamination of separate rock units and occurrence of fracturing. Moreover, it is important that the bolt material would have deformation characteristics, in particular, elasticity modulus (or deformation modulus) and shear modulus that significantly exceed corresponding characteristics of fastening rocks. Indeed, development of a crack during collateral deformation of a bolt with rock walls of a blasthole, should trigger response in a bolt that is directed to crack closing that exceeds (per unit area) initial forces of lamination as much as deformation moduluses of a bolt material exceed deformation moduluses of the rock. This condition, as matter of fact, is the condition of minimal fracturing and lamination, according to which, the greater rigidity of a bolt material the greater resistance of the iron-ore structure to occurrence of cracks and lamination in fastened rocks, the less number of weak areas in iron-rock structure and the greater its bearing capacity. Big bolt rigidity along axial direction can be realized under condition of creation of significant cohesion force along contact surface with the blasthole walls. Thus, it is necessary that the cohesion forces maximally use strength characteristics of the rock. Simultaneously high strength of the bolt installation in a rock should correspond to the strength capabilities of a bolt material.

Preconditions for limitation of fracturing and lamination during the process of iron-rock structure deformation are considered here. But in some cases even before bolting of a rock massif containing an opening, there are already some cracks and broken contacts among the layers. So it is necessary to minimize softening action on the loading construction. This condition requires maximum degree of mutual deformation of separate elements of an iron-rock structure. For example, if the separate layers in the composite structure do not have sufficient rigid link between each other, when bending they will be moving relatively to each other, that is deforming independently and the bearing capability of the whole beam is determined by the bearing capability of a single layer. It means that the main condition of the mutual deformation of separate rock layers is the absence of movement of the rock layers relatively to each other. If to take into consideration that even when we have dislocations of (5-30) $\times 10^{-3}$ m on each edge of the structure in an opening with width span equal to 5 m, its separate layers deform independently, in this case a bolt has to ensure not only longitudinal (respectively to a bolt) but also a transversal rigid link which would not allow big transversal dislocations. Performance of such link can be executed only by means of direct contact of the blasthole rock walls with the bolt along its whole length when a bolt material, working in shear, will resist any transversal movement of rock layers.

Thus, based upon analysis of modern views about functioning of bolts in rock massif, the conclusion has been made in favour of the bolts with supporting along whole length of the blasthole and basic requirements for their structures were defined. Some of these requirements, first of all, is high rigidity of a bolt material that achieves its highest value in metals. For example, the cheapest construction brands of steel have elasticity and shear modules that, in average, by an order exceed proper brands of concrete (reinforced concrete bolts) and by two orders – synthetic resin (steel-polymer bolts). Therefore, direct contact of metal with rock walls of a blasthole significantly increases forces that prevent lamination and fracturing of rock layers, compared with the case, when there is an intermediate less rigid medium that links metal rod with rock.

The condition of realization of the requirement of the high material rigidity of the bolt, as it was considered earlier, is the provision of high tear strength of the bolt itself and its fastening in the blasthole. Fastening in the blasthole of the majority of considered designs of the bolts is provided either by cohesion forces with the rock (reinforced concrete or steel-polymer bolts) or by forces of friction or shear of rock ledges (screw-in bolts).

These forces cannot use strength characteristics of rocks in full because the rocks, in connection with anisotropy of strength characteristics, have lowered shear resistance. Shear strength for rocks, as a rule, many times and even by one order less than compression strength. If there are some rock disturbances in the rock, for example, micro cracks in the plane of shear, then shear strength becomes even less. The only type of load that the rocks resist most successfully is the compression, and in that connection it is quite effective to use strength compressive characteristics of the rocks for bolts fastening along the full length of the blasthole. Obviously, realization of this idea can be carried out by means of applying of normal stresses on the blasthole surface from the side of the bolt, which provide emergence of tangential stresses during bolt's movement relative to the blasthole. By changing the value of normal stresses, it is possible to achieve maximal use of strength properties of rocks and, consequently, receive high strength of fastening of the bolt. Such method of fastening can be successfully used in loosely-connected rocks, where shear resistance is extremely little and its usage in known structures will not provide with the desired results.

Along with provision of high fastening strength of the bolt in the blasthole it is necessary to also increase its tensile strength. In this respect, a metal that can be strengthened, advantageously differs.

Moreover, it is necessary to consider influence of yield of the bolts on their bearing capacity, as due to only yield of support, destroying concentration stresses can be avoided, which often occur during high rock pressure. Yield of the bolt depends on its character of contact with rock walls of the blasthole. Cohesion forces (reinforced concrete, steel-polymer bolts) cannot provide significant yield, because when bolt dislocates relative to the blasthole, the contact breaks and cohesion forces (except roughnesses that create friction during sliding of the bolt) practically disappear - the bolt loses bearing capacity. Yield of such bolts is provided by means of stretching of their material. The most acceptable, in respect of yield, is the earlier considered contact that provides cohesion of bolt with rock by means of forces of friction with radial pressure along the contact surface. Such contact, under condition of keeping the pressure at the same level, allows to move the bolt respective to the blasthole on any acceptable value without loss of bearing capacity.

Thus, basic disadvantages of the most widespread structures of the bolts were discovered with fastening along the full length of the bolt and basic requirements for effective bolt design development:

– material – steel;

– direct contact of steel with the blasthole surface without any intermediate fastening materials;

– character of the contact – radial pressure along the whole blasthole surface that provides occurrence of tangential stresses of friction;

– strengthening of the bolt material during creation of contact;

These requirements can be performed most successfully in the structure of tubular bolt that is deformed before creation of a definite pressure along the contact surface with the blasthole. The most producible and economical method of expansion of the tube shell in the blasthole is the application of the charge explosion energy of the explosive. As this takes place, possibility of a significant strengthening of the bolt's material occurs simultaneously with pressing of the bolt in the blasthole.

4 RESULTS

The property of carbon steels to deform without breakage of material uniformity by a big value lies in the basis of the tubular bolts design. Rocks possess this property, in average, lower by an order, and this allows to use rock walls of the blasthole as more rigid body, as a matrix for punching and fastening of tubular bolt with needed diameter in it (matrix) by means of explosion. At this, idea of combination of a bolt manufacturing with its installation in one process, and increase of strength characteristics of applied type of steel by means of explosion processing is realized.

Structurally, tubular bolt "TA1" is performed in a shape of a hollow metallic cylinder with constant or alternative lengthwise cross-section i.e. it has smoothbore or corrugated shape.

Workpiece of a tubular bolt is performed in a shape of a piece of metallic tube inside of which, in the central area of the cross-section, there are explosive linear charge and blasting agent (Figure 1).

As an explosive, detonating cord of a type DC-A or DC-V is taken, explosive energy of which provides necessary cohesion at the contact rock-bolt for a wide range of mining-technical and geological conditions, and as the blasting agents – permissible electrical detonator of immediate and delayed action.

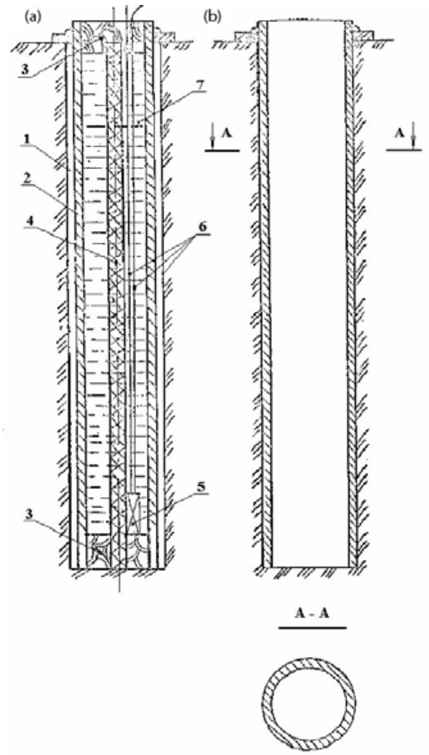

Figure 1. Structure of a shell tube (a) and tubular rock bolt TA1 (b): 1 – blasthole; 2 – shell tube; 3 – clay cap; 4 – detonating cord DCA or DCB; 5 – electrical detonator; 6 – electrical wires; 7 – filling compound (sand, water).

As an inert material, sand or water is taken which serve as a preventive cover for the detonating cord and promotes to even and more intensive influence of a shock wave on the material of the shell tube.

Fixing of the detonating cord in the middle area of the cross-section of the shell tube is performed with help of end caps, for example, from cardboard, clay and other materials, and the fastening of the shell tube in the blasthole – with help of a wooden wedge. In case of tubular bolts usage at mines that are dangerous by gas and dust, the length of the clay caps on the edges of the workpiece should be not less than 0.1 m according to MakSNI (Makeevka research institute).

Inverse initiation of the detonating cord for the purpose of shock wave increase is accepted.

Basic technical data and geometrical dimensions of the tubular bolts have to correspond to calculation data and chart of support of the opening.

Applied materials and bought-in articles have to correspond to current standards and technical conditions.

Following demands are required for the shell tubes for bolts:

– admissible tolerance of the shells length ±10 mm;

– there must not be any metal overlaps caused by gas welder along edges of the shell tube;

– shell tubes that have twists are not acceptable for usage;

Transportation of shell tubes is carried out in mine cars and storage – on the stacks that exclude possibility of their deformation and bending, and also ingress of moisture.

Considered structure of the tubular bolt TA1 has a line of advantages:

– as an energy source for a bolt manufacturing and its installation, the energy of an explosive is accepted, and performance of these various operations is done simultaneously with the blast of an explosive in a workpiece located in the blasthole;

– tube shell is quite simple and has a shape of a piece of a tube of an existing diameter and does not require any preliminary surface treatment and its manufacturing consists of cutting of the needed length tube pieces;

– there is no necessity in manufacturing and installation of such a complex element as a bolt's lock, as the support is manufactured by means of an explosive punching as a solid structure;

– there is no necessity in hammering and screwing of the bolts and, consequently, there is no need in the equipment to mechanize these operations;

– operation time significantly decreases and working productivity during bolt manufacturing and its installation increases due to exclusion of a row of operations that are usually carried out during manufacturing and installation of existing rock bolts;

– reliability of blasting increases because the electricity wires of the detonator are located inside of the internal cavity of the tube shell;

– it is possible to use nonstandard products of the tube production for the rock bolts manufacturing and also used tubes, which significantly increases possibilities of the given method realization and lowers cost of the support;

– usage of explosive as an energy source allows, simultaneously with punching and bolt installation, carry out strengthening of its material that provides a way to decrease consumption of material or to increase the bolt's bearing capacity.

– filling of the internal cavity of the tube shell with inert material increases influence of the shock wave on the workpiece material by 1.5-2 times and ensures safe conditions of detonating cord usage in mines that are dangerous by gas and dust.

The most effective installation of the bolts, in terms of maximum use of strengthening effect of tubular bolts, is their installation during the opening drivage with distance from the face being not more than 2 meters. In this case the bolts come into operation immediately, preventing fracturing and lamination of the near-contour rock massif.

Installation of the explosive in the tube shells is carried out by a shot-firer in the following consequence:

– electric detonator is fixed at the end of the detonating cord by means of electric wires and then it is rolled up into clay cover-cap, the second end of the detonating cord is also fastened by electric wires and DCA is inserted into a tube shell, the end of which is closed by a clay cap with length of which not less than 0.1 m from the side of electric detonator.

– plastic funnel with electric wires is inserted in the open end of the tube shell and then the shell is filled up with either sand or water;

– after the closing of the second end of the tube shell with a cap of 0.1 m length, the workpiece is inserted into the blasthole and, if necessary, fixed in it with help of a wooden wedge, that prevent its falling out of the blasthole under its own gravity or step-by-step performance of blasting.

At a later stage connection of detonators located in the workpieces to the main line and explosion is performed by a shot-firer. Manufacturing and installation of tubular bolts TA1 are finishing at this stage.

In case there are some "misfires", their elimination is done by the following method: on the edge of the workpiece at the blasthole collar parallel to detonating cord, the hole is made in the clay cap by means of breaking, in which the electric detonator is inserted and closed by clay. Repeated blasting is carried out after connection of electric detonator to firing circuit.

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New Techniques and Technologies in Mining – Bondarenko, Kovalevs'ka & Dychkovs'kyy (eds)

Technology in life

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ABSTRACT: Demand on energy in the developing nations is growing rapidly alongside with gradual recovering of the economy and improving the standards of life. But to satisfy the energy requirements in different parts of the world, it is the coal, that remains attractive and important being the only source of energy.

1 INTRODUCTION

The International Energy Agency states: coal comprises 60% of world's reserves of fossil fuel and is spread more evenly and widely than other types of fossil fuel. With today's modern rates of coal consumption in the world, the coal will run out in 250 years, as distinguished from oil reserves, which according to modern estimates, will run out in 40 years and natural gas will run out in approximately 65 years. So in long term perspective, there is no alternative to "black gold-coal". But the situation is getting complicated by the fact that the main coal reserves are concentrated in thin coal seams. It is interesting to note the fact that there is no strict classification in terms of thickness of the coal seam in the world practice of coal mining. For example, in Russian Federation, the thin coal seam are to be the seams which are less or equal to 1.3 m thickness. In Poland, the thin coal seams are considered to be the coal seams which are less than 1.5 m, for example, in mines "Bogdanka" in Lubel and in Katovitse mining company. The same can be attributed to mines in Germany and to mines in American Appalachian mountains. In Ukraine the following qualification is adopted: coal seams which are less than 0.7 m – very thin coal seams; 0.7-1.2 m – thin seams.

According to the opinion of specialists of International School of Underground Mining, the production of coal out of coal seams with thickness of more than 1.2 m decreased nearly two fold, and more than half of coal production falls to coal seams with thickness less than 1 m.

Ukraine is the only country in the world which has gained long term experience, required technologies and equipment for production of coal out of thin coal seams.

The coal seams in Ukraine have the smallest thickness in the world, among industrialized coal mining countries, the coal seams have average thickness of 1.45 m, and in the structure of coal reserves, the seams with thickness of more than 1.2 m have the share of only 20.4%, the thin coal seams (up to 1.2 m) – the share is 74%, including very thin coal seams (less than 0.7 m) – the share is 33.3%.

2 BASIC PART

The mines with thin coal seams have been running for decades in the main industrial Donbass region. In the course of time it is getting harder and harder to mine coal, the considerable amount of workable coal seams are hazardous in terms of mine impacts and mine gas emissions. The depth of each forth mine exceeds 1 km. In Central Donbass (in mines of the towns Gorlovka, Dzerzhinsk, Yenakievo) thin coal seams yield 70% of coal production, and more than 80% of coal is mined in steep coal seams. Under such conditions, it is very complicated to mechanize the technological processes and to ensure the safety of mining.

The main supplier of the equipment which operate in such hazardous conditions is the company "Mining Machines", the main office of which is situated in Donetsk, Ukraine. "Mining Machines" is the leader of Ukrainian mining machine building. The company managers the running of the seven large machine building plants, which are located in the south east of Ukraine and in the south of Russian Federation, and which have more than 100 years' experience of producing the equipment for complicated mining geological conditions. They are: "Druzhkovskiy machine building plant", "Gorlovskiy Mashinostroitel", "Donetskgormash", "Donetskiy Energozavod", "Sverdlovskiy machine building plant", "Kamenskiy machine building plant" and "Ukrtransmash".

Powerful production facilities allow to produce 70% of existing mining equipment in the world.

This fact grants a good opportunity for the customers to order the complete deliveries of the mining equipment on a “turn-key” basis.

The company’s facilities achieved their historical annual production output of mechanized supports – 240 sets, shearers – 1000 units, winding engines – 362 units, fans of the main shaft – 65 units, transformer substations – 2640 units.

Anticipating the trends in the development of mining industry, the facilities of the company have been used as early as in 2001-2003 to start the production of two- prop hydraulic supports of a new generation – 1KDD, 2KDD, DM and DT for thin coal seams. Their life time is three fold more than that of previous types of supports (i.e. up to 8 years without overhaul) and they ensure improved technical performance.

The new supports have the same technical parameters as their foreign analogues and even considerably surpass their operational data.

Having such high technical characteristics, the cost of the company’s products is essentially lower in comparison with the western analogues. This equipment enables to produce coal in semi-automatic and automatic modes and to control the shearer by tracing hypsometry of the coal seam. The above mentioned supports enable to produce coal in coal seams with thickness starting from 0.85 m to 1.6 m. Maximum resistance is 3300 kN .

The shearer UKD 400 (Figure 1) is holding one of the priority positions in the production range of the company. It has a frequency adjusted drive and is designed for cutting coal in the coal seams with thickness of 0.85-1.5 m in the coal faces which are more than 300 m long. Two electric motors with power of 200 kW, enable to cut practically all sorts of hard coal at the feed of up to 8 m per minute.

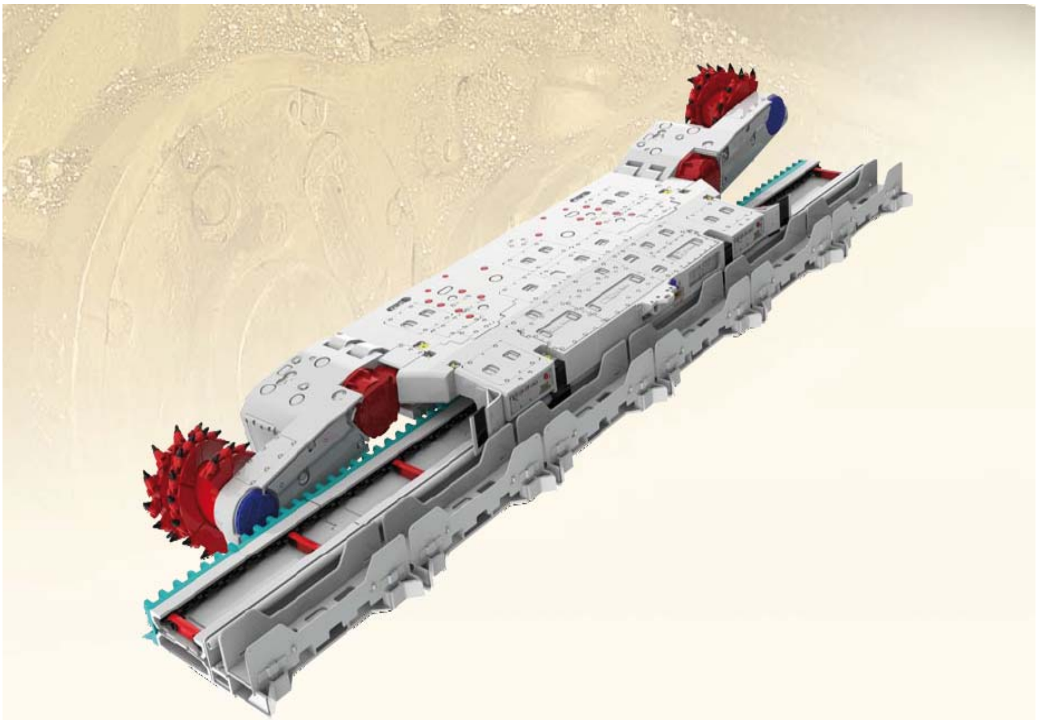

Figure 1. Shearer-loader UKD 400.

Maneuvering speed of the shearer UKD 400 is up to 12 m per min. It is necessary to mention that data of the shearer’s performance: speed, load, the position in the coal face, troubleshooting etc. are displayed on the shearer’s monitor. According to experts’ opinion, the shearer UKD 400 is the embodiment of power multiplied by speed and durable life time.

For the market with average and big thickness coal seams the specialists of the company “Mining Machines” performed the project of creating the shearer KDK 500 (Figure 2). This shearer ensures the output which exceeds 1.5-2 fold the production capacity of its analogues and considerably allows to improve the efficiency of coal cutting with the same costs of coal production.

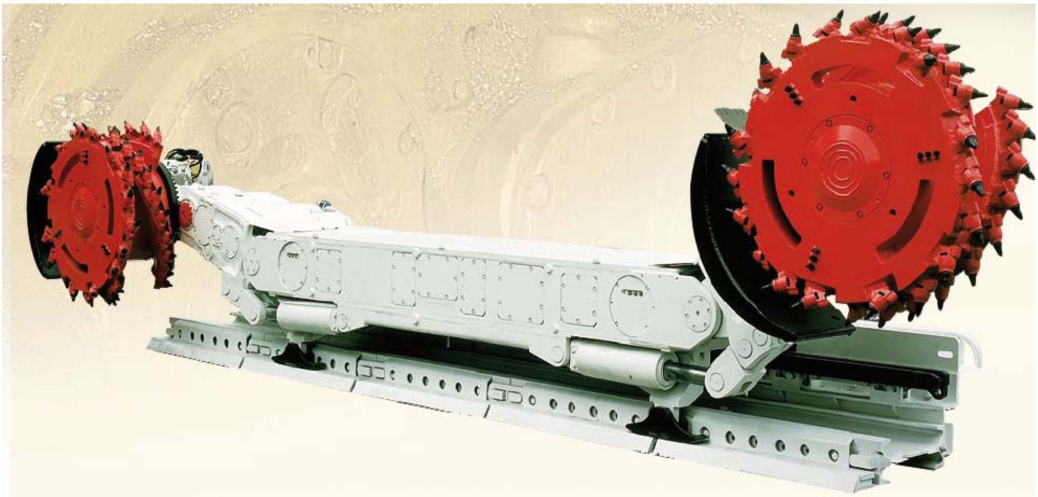

Figure 2. Shearer KDK 500.

The shearer KDK500 proved to advantage while operating in the coal face #17 of Mine Administration “Sadkinskaya” (Belokalitvinskaya district, Rostov region, Russian Federation). The use of the shearer KDK500 enabled the mine to increase average 24 hours capacity up to 3165 tons.

The study of timing observations showed that the main operations of the technological cycle at coal cutting by the shearer reached 79.6% of time duration of the shifts. The shearer KDK500 has produced 1.43 mln. tones of coal for the period of 14 months in the coal faces #17 and #19.

The decision to purchase the shearer KDK500 enabled the mine to ensure the steady increase of production in thick coal seams at JSC “Mine Administration “Sadkinskoye” to achieve the following figures: average monthly production of coal starting from 68.7 thousand tones reached 200 thousand tones (2.9 fold increase); productivity of the worker in coal face ranges from 31 tonnes up to 88 tones / shift (2.8 fold increase).

The shearers KDK500, which operate in the mines “Yuzhnodonbaskaya #3”, the state enterprise “Donetskugol”, named after Bazhanov, the state enterprise “Makeevugol”, “Novodonetskaya”, the state enterprise “Dobropolyeugol”, enabled the mines to achieve maximum production capacity in the coal seams with the thickness of 1.6 m for the coal face of 3200 t / 24 hours at average monthly production of 1700-2400 t / 24 hours. The estimated life time of new equipment in several times exceeds that of replaced analogues and has much longer operational life time ensuring 2-3 fold increase of productivity.

One of the latest innovational machinery which has been put into practice is the flameproof electric battery locomotive with transistor control system AB8T. The locomotive is designed on the basis of battery locomotive AM8D with the main design parameters (overall dimensions, rigid base, clearance). The basic model has undergone some innovations – a removable cabin has been added. A new modern control and slow down systems have been implemented alongside with some additional functions aimed at improving safety of operation. IGBT modules have been used in the function of power elements. Modes of operation are chosen by using software of the central processor. The battery locomotive ensures smooth descending, slow down and makes very low the consumption of the electricity from battery. The power supply is acid battery, type PzS, capacity 560 Ah, which ensures two shift charging cycle at regular operation.

In order to produce modern mining equipment which meets the highest quality standards, the company acquires the best technological equipment from Czech Republic, FRG, Italy , Japan and other countries. In 2006 the company started a complex innovation program of production facilities with more than 20 million dollars of investment. Such amount of currency will be spent for technical re-equipment in 2010.

3 CONCLUSIONS

It is worth mentioning, that in order to improve the efficiency of operation of the produced mining equipment, the company highlights the increasing of complete deliveries of equipment to match the specific geological conditions of the customer and widening the range of equipment due to production of new modifications.

The company considers its priority to interact with the customers by rendering real practical assistance to mines in erection, training of the staff, maintenance of new equipment in the course of operation and regular supply of required spare parts.

Comprehensive scientific-technical strategy enabled the company, alongside with the scientists, design engineers, specialists from the institutes, to master the production of all basic types of transportation equipment, shearers, road headers which meet all modern world requirements in terms of productivity, safety, ergonomics and reliability.

The designing and upgrading of mining equipment is performed by using innovational technologies and completing items which are produced by the leading foreign manufacturers. The powerful stand facilities for conducting performance trials of the produced equipment on the shop floor has been organized at the machine building plants of the company. Particularly, two out of the three trial stands, which are functioning in Europe – STD 2000, which are to be used to trial the produced mechanized supports' sections are located at the plants of the company "Mining Machines": "Druzhkovskiy machine building plant" and "Kamenskiy machine building plant".

In 2010 the company intends to diversify the deliveries of the mining equipment geographically by offering complete deliveries to the end-users of Russian Federation, Iran, Kazakhstan, Vietnam and India. At the moment, the company is developing service-distributional net in the global market. While working with the customers, the company "Mining Machines" follows the innovational principle which highlights the individual approach to each customer by offering the design of the equipment for the specific geological-mining environment conditions.

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Improvement of the methodology of managerial solutions substantiation in coal mining industry

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ABSTRACT: In article questions of methodology of management by coal-mining industry and modern methodological principles of efficiency estimation of managerial solutions concerning functioning of the coal-mining enterprises are considered. Preconditions of searching of the new approaches to a substantiation of managerial solutions in branch and a state level as a whole are determined. Main principles and approaches to forming of the new methodology of management by coal industry are proposed, the basic theoretical positions of this concept are determined. In the work it is also proved the necessity of improving of basic directions of management by resource-saving and management of resource potential is planned.

1 INTRODUCTION

Development of scientific-technical progress, development of society, urbanization and globalization, increasing role of human factor in social life, as well as initiated transition to post-industrial society require deviation from utilitarian understanding and estimation of efficiency of state managerial decisions concerning coal and other branches of economy. Evolution of world economy and economical relations resulted in less part of past and vital work at each enterprise reflecting social expenses for manufacture of production at this enterprise. In addition, with the growing of social significance of humanitarian and ecological factors in human life, visual results of each enterprise and economical branches in their financial and other reported indexes continuously show less accurate efficiency of their operation in national economy and real social usefulness of these results. Everything mentioned requires reviewing former views of sense and content of "usual" categories, as a sample, an efficiency, social expenses, cost, goods for improving of substantiation of managerial decisions.

The aim of this article is to study different approaches to forming of modern methodology of substantiation of managerial decisions in coal industry management.

2 RESULTS

In historical perspective science and management practice always work out the problems of determination of priorities and comparison of separate kinds of effects – economical, social, ecological, political, expenses and results etc. Due to different reasons, a priority was far from being attached to economically efficient decisions.

Because of specific modern economics, progress in integration processes and peculiarities of coal mining industry in general, except Ukrainian one, not all managerial decisions concerning current and perspective functioning of coal mining enterprises is reasonable to explain by the level of economic efficiency of their activity.

From our point of view, the necessity of forming of the new methodological substantiation of managerial decisions has matured. It could consider challenges of globalization and state's role in society's life, present-day condition of economical and social relations, human values as well as transition to open model of economical activity, which accepts values and integrity with environment.

Long-term practice of world's state management is a practice of number of transfers, i.e. free transfer of money, goods, services, tangible property. Last world economical crisis showed that to provide stable functioning of economy, many state decisions are also accepted, which are not aimed to obtain direct current economical effect. In addition, from the state budget, stabilization funds spent multi-billiard costs, which are introduced into keeping rate of currency, providing bank liquidity, cheap credits, stabilization of work of many large enterprises etc. The same situation happens in management for state property. So, M.D. Bilyk observes, that referring to defined role of state property the most preferable tendencies of functioning of state property are the ones outgoing from the regulatory state function, particularly:

– filling economical caves with unfavorable manufacture conditions, to which private capital comes too slowly, but its development is necessary for the whole country's economy;

– sanitation of separate enterprises, or probably whole branches of economy, which temporarily or continuously are revealed unprofitable;

– regulation of social aftereffects of cycloid type of production by widening of scale of social works for overcoming of unemployment;

– providing of realization of active regional policy oriented to leveling of great disproportions in economical development of regions;

– motivation of developing of science intensive, progressive branches, which require enormous initial capital investments and do not provide with quick refund of capital;

– increasing of quantity and quality of so-called "social" goods, i.e. economical, social and administrative areas of the whole society;

– creating and providing of the activity of enterprises in strategically important branches;

– participating in the activities concerning the nature protection, achievement of the ecological balance, improvement of recreation technologies (Bilyk 2006).

Thus, one can affirm that on the present stage of development of world economy within modern social-economic relations in conditions of globalization on the goals indicated and generated state functions then spending of state money according to managerial decisions haven't the aim of obtaining of direct economical effect. These expenses are used mostly for efficient performing of state's functions.

During establishment and development of economical relations, forms and methods of management are changing and a state receives new functions and tasks. Main functions of the state are determined by the tasks arising before the state in a certain moment in all social spheres. In different progress periods the state faces continually new tasks, for solving which there necessary real resources including the managerial ones. There are such situations when the state functions reveal that dependence on permanently changing conditions can not stay invariable. Despite the main state functions which are permanent and are performed systematically, they are changing, developing dependently on those tasks which the state has on the certain historical stage. Subject to tasks and purposes of state are determined economical, political and social conditions.

According to the thoughts of many economists, an economical function is the most important because it creates material foundation for realization of other state's functions.

Economical function usually has anti-crisis orientation and forms stable economical development of the state. General economical function of state consists of many partial economical functions.

From our point of view, starting from the necessity to provide economical safety for national economy, one of the main declared functions of the state is necessary to mark insured and compensative regulation of economy in those social spheres, where even market methods can't provide necessary positive results. It's clear, that market methods don't work in those spheres of economy or others where investments don't give reasonable benefit.

In certain degree insurance function and compensability regulation for state used to be performed and today it is performed through other state functions, but growing and modern understanding of the importance of economical crisis requires its separate investigation.

All managerial decisions, which are accepted in organizations and state control bodies can be divided into two groups – programmed and unprogrammed. Programmed ones are those decisions which operation order is standard and doesn't require searching for alternative options. There is full information about those decisions as well as organization has established actions for their acceptance and realization and all alternatives are known and formalized. For example, tax charge, custom fee etc. Unprogrammed managerial decisions have the following features:

– non-standardized precedence rule with their acceptance and solving;

– scanty information for overall substantiation of decision;

– not established accurate criteria for effective solving;

– probable alternative decisions with not accurate determined results;

– it's impossible to forecast accurately the development of economical situations in national and world economy especially in conditions of widening of crisis phenomena;

– it's impossible to forecast accurately people's behavior as the main element of social-economic system.

Practically all strategic decisions, which are accepted by state control bodies, are unprogrammed, so it's impossible to calculate precisely their introduction of efficiency using famous methods, but in separate cases they try. Explanation of such decisions is possible only basing on natural laws and logic laws.

Long-term continuous unprofitableness of the majority of coal mining enterprises, hot demand of enormous state subventions for their keeping in operation don't allow solving the most important problems of the future of coal industry, and the branch itself was and has been so-called stepchild of Ukrainian economy.

On our mind one of the main reasons of such situation is methodological uncertainty as to solving of the specific problems of the specific and at the same time very important branch of national economy.

That methodological foundation being used now was formed in Soviet Union in other kind of economy. So formation task of new methodological principles is topical for explanation of managerial decisions which can meet the present state of economy.

Design analysis of methodological character shows that nowadays methodological range of problems of certain science branches is studied rather less than generally philosophic and generally scientific, moreover it has unsolved methodological problems. In the same content it concerns an economy of coal mining industry.

First of all it touches the questions of restructuring of coal branch, state support for coal mining enterprises etc. For example, in recent years it was made a lot of attempts at all management levels to solve problems of coal mining industry of Ukraine by content using of generally scientific methods. But, as a rule, branch specificity is not considered as well as influence of forms on processes in country's economy, which occur in coal mining industry.

Necessity of independent scientific approach to accepting the decisions on functioning of coal mining enterprises are mentioned by other economists too (Chilikin 2000).

Formation of methodological base generally and certain science branch separately stipulates collection of data and phenomena, their systematization, classification and generalization. In modern economic literature concerning economics of coal industry facts and phenomena, which are specified by scientists and practitioners are spread in many publications and according to them, nobody made any sufficient modern methodological conclusions. In this connection, development of all level methodology and, first of all, particular-scientific level can be studied as one of the most important conditions of further economical development of coal mining industry of Ukraine.

Economical science comes out from background of rational behavior of management agents, owners and public officers, which accept managerial decisions and perform them. Rational behavior covers affective use of all kinds of economical resources during the process of achieving the set tasks of activity.

So, methodology improvement of substantiation of coal industry development stipulates essence of reveal of rationality of managerial decisions and searching for methods of effective usage of the economical resources.

To accept decisions concerning development of coal mining industry in general and its state sector separately is proposed to use methodology of state rationalism, which has combined character.

Usually rationalism is considered like sign of maturity, civilization, modernity, like significant feature of industrial life style.

Rationalism is usually explained as person's ability to keep intellectual laws, to be logical and sequential; ability to methodological procedure and cognitive attitude to the reality; maximization and optimization of activity results at limited resources and methods, structuredness and analytical transparence of research material, standards, principles, norms and ideals, which are supported by scientific community, behavior quality, which is oriented to realization of personal interest etc.

State rationalism is explained as methodology of accepting of managerial decisions by state control bodies, which is rest upon the priority of common national interests, oriented to economical development and based on the limitedness of society's economical resources.

Principle of state rationalism in economy management: less expenses – bigger results, both in current period and in the perspective. Among relevant peculiarities of rationalism one should notice: sequence, connectivity, harmony, simplicity, structuredness and others. Rationality possesses universal methods. It's one of the general views of industrialized society.

On the authors' opinion, the methodology of state rationalism in economy, first of all must be based on such main principles:

– accounting of natural laws and logical laws;

– reasonability of state control for economical relations, growth and economy structure pace;

– maximization of useful information for substantiation of decisions;

– following the Constitution and laws;

– responsibility before the international legislation;

– declaration of person's life and health as the highest value;

– national security (economical, energetic, etc);

– sound mind – i.e. equivalence between people's personal opinions and environment;

– efficiency – economical, social, political (current and perspective);

– unity with environment;

– system in influencing of state decisions onto national economy and society;

– transparence for society over all stages of preparation and realization of decisions.

It is considered that formation of methodology of state rationalism and its implementation into management practice will allow formation of managerial decisions acceptance process, i.e. creation of opportunities of using formal logic when explaining managerial decisions concerning unprofitable branches, e.g. coal mining.

Over last 15 years mass media generate the opinion in the society oriented to necessity of mines closure, which operation is unprofitable and requires enormous subventions from the state. Over independence period in Ukraine in accordance with developed programs during restructuring process in coal industry more than 100 unprofitable mines were closed. From those 138 coal mining enterprises left in the branch being under control of Ministry of Coal Industry of Ukraine, only 15 of them or 11% are considered as financially balanced, the others are determined as the ones requiring subventions and incapable to work independent (Tulub 2007).

One should notice that acceptance of decisions concerning closure of unviable coal mining enterprises, all enterprises of national economy, regardless to kind of proprietorship, having legal independence, are bound by system economical relations with other national enterprises. For example, coal mining enterprises are bound with supplier of goods which are used during coal production process. In addition, one can note the system ties with the enterprises which produce mining equipment, perform preparation of new longwall.

All enterprises which operate in national economy pay taxes into state local budget, pay charges into fund-in-trust. And if in this system the operation of some coal enterprise is stopped then reducing production capacity at many enterprises will be resulted and then the decrease of all kinds of charges into budget and fund-in-trust. So, more frequently when closing a coal mining enterprise oriented to reduction of state subventions, national economy faces much more losses, which are not accounted by present methods of analysis and estimation. Performed researches proved that closing of many unprofitable mines leads to more losses in economy than total state subventions on their keeping.

Based on methodology of state rationalism and its such principles like national security, efficiency, system influence of state decisions onto national economy and society, coal mining enterprises require further state support. The other matter is the state's ability to provide support, optimization of this support over subjects, kinds, volumes and increasing efficiency of state funds using.

Formation of modern methodology for estimation of efficiency of managerial decisions in coal mining industry is a difficult process which requires efforts of many scientists and practitioners for a long period. But some approaches to formation of such methods, principles are studied by science and they can be offered now.

Among them are:

– coal is a main current and strategic energy source of Ukraine;

– coal is the base for power and economic security of state;

– the majority of time coal industry of Ukraine was unprofitable;

– with deepening of excavations the expenses grow more for coal production;

– increasing of coal cost leads to increasing of cost for goods and services in the country;

– when closing mines the losses are bigger than subventions for supporting their operation and producing coal;

– economical effect from performing of managerial decisions is important, but it is not the only effect from this decision;

– economically unprofitable decision can be useful if promotes achieving the other vital goals;

– significant factor of efficiency of managerial decision is refrained loss;

– in cost price of coal in mines the part of conditional-permanent expenditures is 60-70%, so a mine can not work properly with low level of coal production;

– investments' growth into coal industry results in multiplied profit growth in economy; reduction of investment results in multiplied decline of profits and consumption demand.

Further formation of methodology estimation of efficiency of managerial decisions is an important background for economic progress in coal mining industry. First of all a solution of methodology of state support for coal mining enterprises is worth being developed as well as progress of coal industry with improving of technique of labour remuneration and management of resource potential.

Management of resource potential of coal mining and other enterprises is impossible without use of questions which arose from globalization of scientific-technical progress, development of ownership relations, which require their own accounting in theory and practice of resource using.

At more detailed consideration, resource potential is accepted both like ordinary economical resources and specific resources: intangible assets, enterprise's image, information, time, etc.; involved resources which form the resource potential of enterprise and it can change the type of ownership; resource potential of mining enterprise is limited by time limits that is connected with terms of production of mineral in different conditions; resource potential is greatly influenceced by not only presence of resources but their quality, ability to efficient use.

Resource potential of each coal producing enterprise is one of the constituent parts of coal mining industry of Ukraine and national economy in the whole. It stipulates the importance, urgency and necessity to continue researches and application of principally new approaches of solving the problems of effective formation and more complex use of resource potential of coal mining enterprises accounting their special features.

Social needs are always connected with resources. Oriented to guaranteeing economical supply for society, the first range task of any state is state regulation, to provide efficient use of these resources.

At all stages of society's life there is constant gap between consumption level and production possibilities, because economical resources are always limited. At the same time society always has opportunity for alternative and more rational using of these resources.

Usually resources consumption is in certain space-term limits and with accounting of real or, at least, potential physical accessibility of such resources.

It's necessary to note that opportunities of alternative (more rational) use of economical resources have considerable specifics in different spheres of society and economy. So, it is using of accurate-scientific methodology, which is implemented into research methods and which is pierced by fundamental and common science methodology that will allow to consider branch peculiarities in economical substantiations of branch progress. On the grounds of their scientific methodology, an opportunity to develop and use management methods appears, as well as methods of plan calculations and analysis of activity etc, activity model, algorithms of actions and other managerial tools for economical subjects.

There are some peculiarities of using of economical resources in coal mining branch, which require special methodological approaches for their performing. Particularly, big final "price" of miners' job. This "price" consists of relatively big salary, big social charges and big pension payments; non-renewable nature of the job subject – mineral resources, variable conditions of coal excavation, and non even natural quality of coal; orientation of business activity at the coal mining enterprises onto reducing the losses and decreasing volume of state subventions on covering of production expenses; specific effect of time factor onto the activity of coal producing enterprises, particularly: when deceleration of intensity of second working relatively to time absolute and comparative expenses are growing greatly on keeping breakage heading and preparation of sites as results of rock pressure; termination of performing some production operation for several dozens of hours, for example, dewatering and ventilation, can completely destroy the mine.

So, these peculiarities have cumulative influence on results and operation characteristics of mines and it requires formation of special methodological approach for their accounting in analysis and planning of coal mining enterprise's activity.

3 CONCLUSIONS

In spite of rather deep study of many tasks of state regulation for economy branches, a range of substantiation and rationalization factors in managerial decisions require further formalization. Bu using of proposed methodology of state rationalism one can shape the decisions of important problems in coal mining industry, connected with using of current and development of perspective production-economical potential of coal producing enterprises etc.

From our point of view, development and practical implementation of state rationalism methodology allow to increase the validity of many unprogrammed decisions, which are accepted nowadays on the base of intuition and subjectivity.

And accounting of branch peculiarities of application of economical resources is the main direction of methodology improvement of managerial decisions substantiation with resource saving and progress of coal producing enterprises.

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Review of support systems and methods for prediction of gateroads deformation

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ABSTRACT: The paper describes the support systems applied in gateroads on the basis of selected countries conducting underground mining of hard coal seams. There have been presented the support systems for those workings, being applied in the United States, United Kingdom, Germany and Poland. The methods applied in those countries, allowing to compute gateroad deformations, and to design appropriate supports, for given geological and mining conditions, have also been described.

1 INTRODUCTION

Longwall coal mining has been practiced for many years particularly in European coal mines. In the case, when this method is used, for each longwall panel gateroads and a set-up rooms are developed. In those workings the longwall equipment is placed, containing: shield supports, a shearer or a face conveyor. The European mining industry is working with single entry roadways system, while in the U.S. for one longwall panel from 3 to 5 tailgates and maingates are developed. During longwall mining the gateroads perform many functions that decide on a proper production process. They are the ways that enable output haulage, transport of miners and materials, or are ventilation ducts for the mining area. The aspect of proper ventilation is of vital importance in the case when mining operations are conducted in the conditions of high level of hazards methane explosion or fire. In such situations, considerable deformations of gateroads that give rise to reduction of their cross sections, influence substantially the rise of the level of hazards mentioned, thereby putting the working crew in danger.

Considering the importance of gateroads in the production process, numerous research works have been conducted in many countries aimed, in principal, at ensuring the stability and proper size of those workings. In this paper there are described different kinds of gateroads support which are applied in the: U.S., UK, Germany and Poland. The methods enabling to predict the deformation of those workings, and process of designing the supports reducing this deformation have also been presented.

2 CHARACTERIZATION OF HARD COAL MINING I NDUSTRIES AND SYSTEMS OF GATEROADS SUPPORT IN THE: U.S., UK, GERMANY AND POLAND

With the aim of obtaining a broader view in the subject of support systems applied in gateroads, typical solutions have been described, being applied in the countries conducting underground extraction of hard coal seams. To compare, such countries have been chosen as the: U.S., UK, Germany and Poland. The United States are counted among the biggest producers of hard coal in the world. In the year 2009, a total U.S. production was 973 million tonnes, in that 302 million of hard coal were extracted with underground methods (50% with a longwall system, the other 50% with room and pillar). An average depth of underground mining is 400 m. In the remaining, European countries selected in the paper, the hard coal production level using underground methods, in the year 2009, varied from 13.8 million tonnes in Germany to 77.5 million in Poland. In the UK, 17.9 million tonnes were produced, while nearly 10 million were obtained using surface mining. Figure 1 shows the above presented values of hard coal production in the individual countries, together with the values of average depth of mining, in the case when underground mining methods were used.

In the USA, for each longwall panel, from 2 to 5 tailgates and maingates of rectangular cross section are driven (Barczak 2005 & Peng 2006, 2008). Between those workings chain pillars are left, providing an additional roof support. Rock bolts are the main type of support used in the gateroads, being driven, in general, by means of Continuous Miner. There are used steel bolts of 19 mm diameter and length, most frequently, from 1,8 m to 2,4 m. The fully resin steel bolts are installed with spacing of 1.2 m. (Barczak 2005 & Peng 2006). Figure 2 shows an example of rectangular roof bolted roadway.

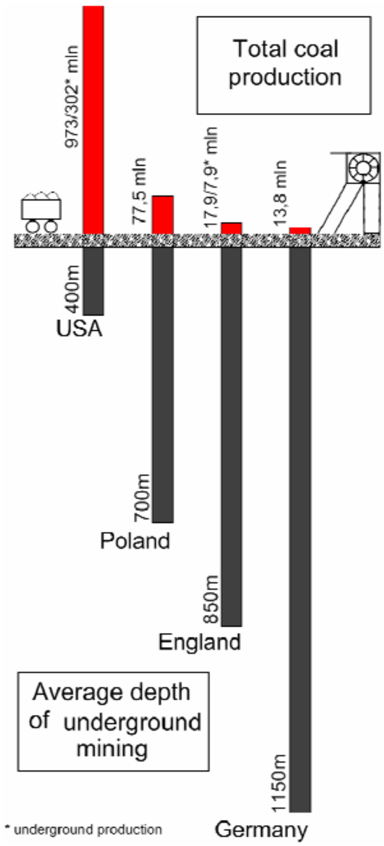


Figure 1. Production of hard coal in the year 2009 and average depth of underground mining in the: U.S., UK, Germany and Poland.

Figure 2. An example of rectangular roof bolted roadway in one of U.S. mines.

In the locations with high values of stress e.g. in abutment pressure areas, or in tailgates located in the one-sided surroundings of the goaf, there are applied additional reinforcements of the main supports, such as cable bolts or various standing supports (Barczak 2005 & Tadolini 2005). The selected types of standing supports, being applied in reinforcement of the main rock bolt supports in the gateroads, are shown in Figure 3.

The Can-type standing support is one of the most frequently used supports. It is composed of a tube filled with material having high mechanical strength parameters. In the U.S. gateroads, there are also used various timber supports, for instance: Cluster Props, Propsetter, or Link-N-Lock cribs (Barczak 2005).

In the UK, after ownership transformation and privatization of the hard coal mining industry, the rectangular cross section and rock bolts as the main support became to be a standard in the gateroads (Bigby, Altounyan & Cassie 2006). Such a system of supporting the entries, introduced in a large scale in 1987, results mainly from continual striving for lowering the production costs, and from the ability to obtain higher rates of development works (Altounyan & Hurt 1998). To protect the gateroads, as the primary support fully resin steel bolts are used and additionally flexible or cable bolts, including pretensioned bolts. The spacing between the rows of bolts varies, depending on geological conditions, 0.6 m to 1.0m. In the roadways, there is also used rib bolting using either steel bolts or GRP bolts (on face rib side). The reinforcement being frequently used, in particular in tailgates outby the face, are the Link-N-Lock cribs (Bigby, Altounyan & Cassie 2006). Figure 4a presents a detailed scheme of the rock bolt in a gateroad in which fully resin steel support and, additionally, flexible bolts were installed in the roof. In the face rib sides, GRP bolts were applied. Figure 4, b shows a view of the rectangular rock bolted gateroad in one of the UK mines.

In the UK mines, with the aim of ensuring acceptable size of the rock bolted gateroads, coal pillars are being left between longwall panels. The width of those pillars is variable, for instance, at a depth of mining of 800 m, the width of pillars ranges from ca. 60 m to 120 m (Cassie, Altounyan & Cartwright 1999).

In the German mining, the principal support applied in gateroads is the steel arch yielding support. The single frame of support consists, most frequently, of five-elements of section with mass from 34 to 40 kg / m. The gateroad cross section value varies from 25 m^2 to 36 m^2, on the average being 30 m^2. The frame distance ranges from 0.6 m to 1.2 m, most 0.8 m. Currently in Germany, nearly 70% of gateroads are developed in combined support i.e. bolt and arch support with concrete backfill, called type A (Eikhoff 2010). This type of support was developed at Ibbenbüren mine in 1985, and relies on applying in workings at the beginning rock bolts, and then, at a distance of up to 50 m, the arch support (Junker et al 2006). Figure 5 presents working development in combined support – type A.

(a) (b) (c)

Figure 3. Selected types of standing support being applied in gateroads in the U.S.; Can support (a), Link-N-Lock crib (b), Cluster Props support (c).

(a) (b)

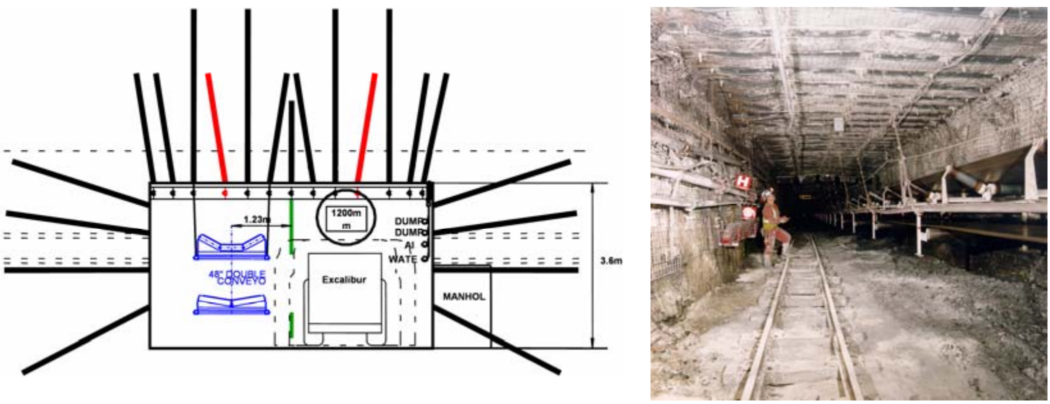


Figure 4. Rock bolted gateroad in one of the UK mines; (a) detailed scheme of rock bolting, (b) view of rectangular rock bolted gateroad.

In German mines most rock bolts the 2.5 m long and 25 mm diameter are used, and having the tensile strength of 340 kN. In the workings, concrete backfilling is used between the breach of working and the supports, using the materials with high mechanical strength. During retreat longwall mining very often the one of roadway after the first passage of the face is being used. In such cases, roadside packs are being made in the roadways providing reinforcement of steel supports, and additionally isolating the goaf debris (Junker et al 2006).

The main support type used nearly in all hard coal mine roadways in Poland, including gateroads, is the steel arch yielding support. The frame is the main element of this support, in general arch-shaped. It contains three or four elements made of V section, and most often having a unit mass in the range 25-32 kg / m (Prusek 2010). Over the last years, in the

Polish mining industry, the frames have been used with cross sections from 13 m^2 to 18 m^2. Most often, in above 50% of workings, the support with the cross section of ca. 15 m^2 is used. Depending on the local geological and mining conditions, the frame distance adopted is from 0.5 m to 1.0 m, while most frequently the spacing used is either 0.75 m or 0.8 m.

Figure 5. Working development in combined support, type A (Eikhoff 2009).

In the Polish mining gateroads, combined support is more and more often used, with the bolts playing a role of additional supports. Many mines perform bolting of cross-bar arches of steel supports, which apart from strengthening the rock mass, eliminates the necessity of installing steel props in the area of longwall T-junction. This bolting is generally made outby the face by means of fully resin steel bolts, of the length from 2.5 m to 2.7 m (Figure 6a). There is also applied bolting between steel frames or fastening with bolts steel cross bars or stringers to the arches. This is made by using flexible bolts of the length ranging, in general, from 4.5 m to 8.0 m. Apart from bolts, the most frequently applied in strengthening steel yielding arches in gateroads, there are used: stringers, frictional steel props and timber props. In the case of repeated use gateroads after first face passage, are often made reinforcements of steel supports by means of wood cribs, or there are made roadside packs along the goaf line, using various chemical materials (Figure 6b) (Prusek 2008, 2010)

(a) (b)

Figure 6. Reinforcement of steel yielding arches in gateroads by means of: (a) bolting cross bar arches with fully resin steel bolts, (b) roadside pack of mineral binding material.

With the aim of improved utilization of support parameters, and obtaining a momentary contact of the support with the roof, the Polish mines undertake more and more often the trials to apply backfilling with the use of chemical materials. The backfilling of this type is commonly used at Bogdanka mine. There, it has become a standard supporting element, being one of the factors that ensure obtaining of very good rates of roadway drivage (Chmielewski, Masiakiewicz & Kozek 2010). The mine working with the arches backfilled with binding material at Bogdanka mine is shown in Figure 7.

Figure 7. Roadway at Bogdanka mine with arches backfilled with binding material.

In Poland, the number of mines that use the supports made of new sorts of steel with enhanced mechanical parameters has increased over the last years. Apart from enhanced mechanical parameters, the new steel is also more resistant to corrosion processes which may substantially lower the load-bearing capacity of the supports, and finally result in the loss of their stability. The use of steel with enhanced parameters positively affects the process of roadway maintenance, and results in reduced costs of drivage through increasing the pitch of the frame or using lighter section (Prusek, Kowalski & Skrzyński 2006).

3 METHODS OF PREDICTING GATEROADS DEFORMATION

The investigations on the phenomena occurring in the gateroads, and in particular the rock mass deformations caused by abutment pressure have been the matter of interest of scientists from many countries, in which underground mining of hard coal seams has been conducted. As a result of research work conducted, a number of methods have been developed, enabling to predict deformations of the gateroads, and optimize the supports being subjected to abutment loads. Among those methods, one can distinguish empirical methods and numerical modelling methods commonly used over the recent years.

In the U.S. mining industry, when planning underground mining process using the longwall method, in relation to gateroads, mainly the computations are performed in respect of designing chain coal pillars to be left between the entries, and of analysing roadway supports. The size of pillars, which can be either stiff or yielding ones, are being computed by applying various analytical and numerical methods, as well as empirical relationships (Mark 1990, 2006). During designing gateroad supports, the U.S. approach takes as a base the conception of the Ground Reaction Curve (GRC). This curve presents the dependence of abutment load and convergence which may occur in given gateroads (Mucho et al., 1999; Mark & Barczak 2000; Esterhuizen & Barczak 2006). The concept of selecting the supports based on the determined ground reaction curve is presented in Figure 8.

When selecting gateroad support based on the knowledge of the GRC, one takes mainly into account: load-bearing characteristics of the support, time of its installation in the working and convergence of working. Figure 8 presents an example of applying too soft support, when the line of its load-bearing characteristics intersects the GRC behind the inflexion point, in consequence the convergence is no longer acceptable and puts in danger of rock fall. It should be noted that in the case of using high stiff support in the gateroad, the support will be subjected to high load values, being able to cause its premature damage. For this reason, in every case, individual computations and analyses should be made to select the suport relative to a given course of roadaway convergence (Mucho et al. 1999; Esterhuizen & Barczak 2006; Barczak et al., 2005).

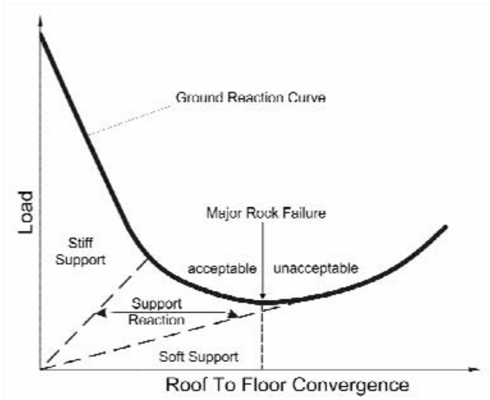


Figure 8. Concept of selecting working support based on the GRC (Mucho et al., 1999).

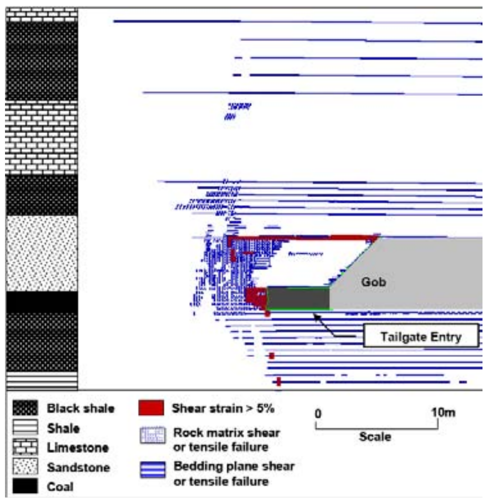


Figure 9. Numerical modeling of the rock mass deformation around the tailgate by means of FLAC program (Esterhuizen, Barczak 2006).

With the aim of computing the course of gateroad deformation, such programs as LaModel and FLAC are commonly used (Barczak et al., 2005). The LaModel program utilizes the boundary element metod, and allows to compute the convergence of roof strata consisting of laminated elastic rock mass, and assessment of interaction of: support, roof, floor and pillar yielding (Heasley & Salomon 1996, Barczak et al., 2005). In the case of inelastic rock and failure of the roof or floor, the FLAC program is utilized in numerical modeling (Barczak et al., 2005). An example of applying this program for numerical modeling of the rock mass around the tailgate is presented in Figure 9.

When selecting the gateroad support using the GRC conception, of vital importance is the knowledge of the support load characteristic. The majority of standing supports used in gateroads (50 different types) had been tested in mine roof simulator in the NIOSH (National Institute for Occupational Safety and Health). The results of the tests were included in the program named STOP (*Support Technology Optimization Program*). The program allows to compare the support characteristics with the ground reaction curves and to choose an optimal support relative to occurring convergence (Barczak 2000).

Referring to the problem of gateroad deformation, the U.S. experience points out that those roadways are more stable in the case of their smaller width, and appropriate orientation of the workings with the direction of maximal horizontal stress (Mark & Barczak 2000).

The subject of appropriate orientation of workings relative to the direction of maximal horizontal stress is devoted a lot of attention in the UK. The first measurements of the direction of action of maximal horizontal stress were conducted in underground conditions in mid-eighties of the last century at Selby mine. Since that time, such measurements in the British mines have been conducted currently, being one of the most important factors in the process of designing mine working supports (Bigby, Altounyan, Cassie 2006). Wide experience obtained points out that improper orientation of the gateroads relative to the direction of action of maximal horizontal stress can give intensity of their deformation.

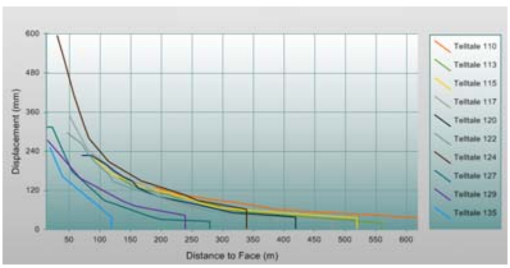


Figure 10. Horizontal convergence in the gateroad depending on the distance from face (Bowler, Betts & Altounyan 2008).

In the UK, when selecting gateroad support, numerical modeling is being applied. Also, there are performed underground measurements of the gateroad deformation for optimization of numerical models, and at selection of an optimal support scheme, ensuring minimization of deformation. An example of underground measurements of gateroad convergence at Daw Mill Colliery is presented in Figure 10.

One can observe from Fig.10, that as the longwall face comes nearer, the rise in horizontal convergence of the roadway takes place, and at a distance of 50 m outby the face, the value of convergence exceded even 600 mm. Taking into account substantial difficulties related to side wall deformation in the gateroad, a number of actions were taken with the aim of limiting this negative phenomenon. New support types were introduced and longer and larger diameter rock bolts were applied for bolting the side walls. To evaluate the effect of changed rock bolting parameters on deformation of the gateroad, numerical modeling was applied (Figure 11).

The investigations on the course of gateroad deformation have extensively been conducted in the German mining industry. Based on the results of measurements and underground observations, as well as physical modeling, a number of empirical relationships have been worked out, enabling to predict the gateroad convergence (Götze & Kammer 1976; Jacobi 1976; Kammer 1980). To calculate the principal convergence (K_0) of the gateroads, the relationship has been given, and then modified throughout the years. The last of the published forms of the relationship is that as shown below (Junker et al., 2006):

$$K_0 = -78 + 0.066\,T + 4{,}3\,SV \cdot M + \\ + 24.3 \cdot \sqrt{-4 \cdot ln\left(\frac{\beta_{Dmin} + 34}{168}\right)}\,, \qquad (1)$$

where: K_0 – convergence of roadway as percentage ratio of initial height; T – depth of mining, m; M – thickness of seam, m; SV – number depending on the type of roadway protection at the edge of goaf; β_{Dmin} – minimal compressive strength of rock in the roof or floor of roadway, MPa.

Relationship (1) allows to calculate principal vertical convergence of the roadway developed during the advancing longwall with caving, being outside the past mining activities, and with arch supports applied with stone lining. In other cases, the roadway convergence (K) is being calculated taking into account the principal convergence (K_0) and proper correction factors (Kammer 1980; Junker et al., 2006).

(a) (b)

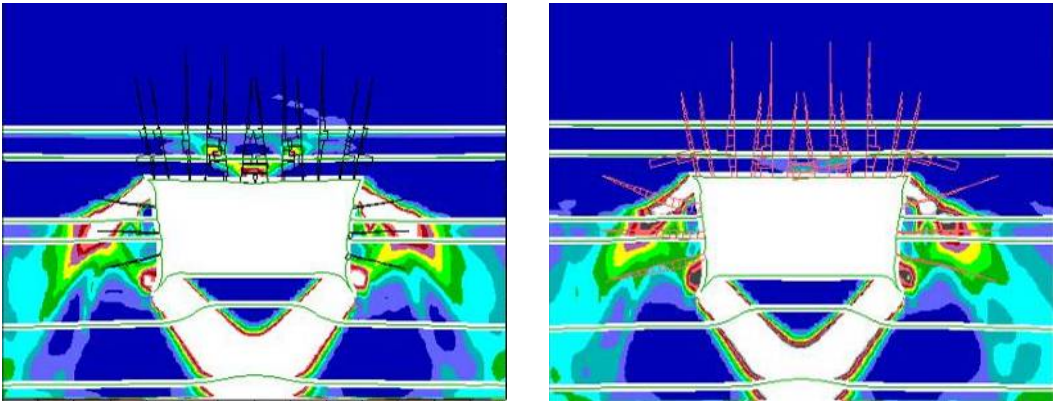


Figure 11. Numerical modeling of gateroad deformation at Daw Mill Colliery; (a) conventional scheme of bolting; (b) scheme of bolting using side wall rock bolts with increased length and diameter (Bowler, Robinson & Altounyan 2009).

(a) (b)

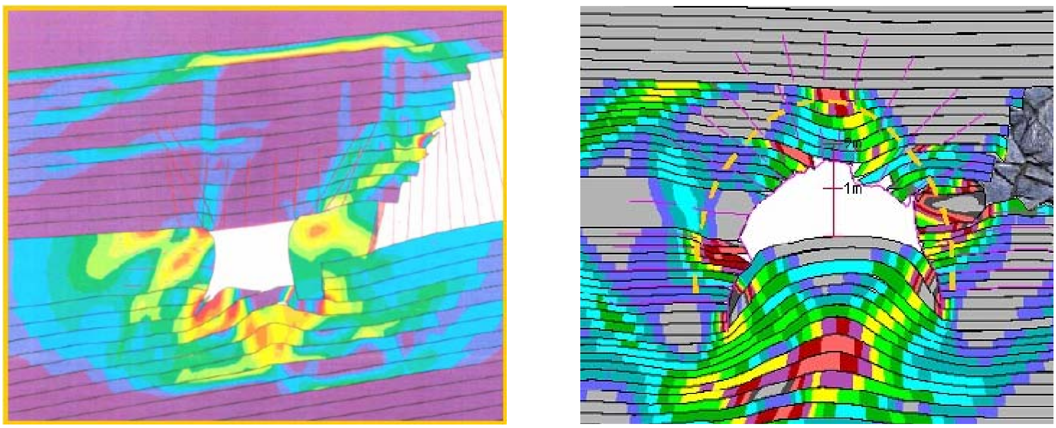


Figure 12. Numerical modeling of gateroad deformation; (a) rectangular bolted gateroad (Langhanki 2001); (b) arch-shaped bolted gateroad (Ruppel & Scior 2008).

In Germany, apart from empirical methods, to predict gateroads deformation numerical modeling is also used (Hucke et al., 2006; Langhanki 2001; Ruppel & Scior 2008). Computation examples of convergence of gateroads with rectangular and arch-shaped cross section, with rock bolting are shown in Figure 12.

The numerical computations are verified, and the rock mass models created are calibrated based on: laboratory testing of rock and supports, physical modeling, underground measurements of: properties of rocks, convergence and stress. The information required for calibration of numerical models is shown in Figure 13.

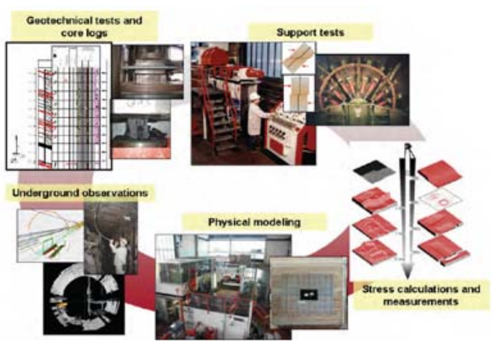


Figure 13. Information required for calibration of the numerical model (Studeny & Scior 2009).

Also in the Polish mining industry for many years underground investigations of gateroads deformation have been conducted. On the basis of measurements a number of empirical relationships have been worked out (Prusek 2008). From among the methods that base on empirical relationships, the most extensive is the method worked out at the Central Mining Institute (GIG). This method allowing to predict the course of vertical convergence of gateroads, taking into account a large number of geological and mining parameters (Biliński 1989). The relationships applied in this method underwent many modifications, based on currently obtained results of underground tests. The prognoses of convergence based on the method discussed are being made by means of elaborated computer programs, speeding up and improving the process of computations. An example of the computation result of predicted deformations of the gateroad is show in Figure 14.

In Poland, apart from empirical methods, in making assessment of gateroad deformations, numerical modeling is being often applied. Figure 15a presents the results of numerical modeling of gateroad maintained inby the face using support members to model the caving zone (Prusek & Masny 2007). Figure 15b presents the results of computations of rock displacement into the gateroad at the distance 100 m inby the face. Calculations were carried out by means of the Phase[2] software and modified Hoek-Brown's criterion (Prusek 2008a).

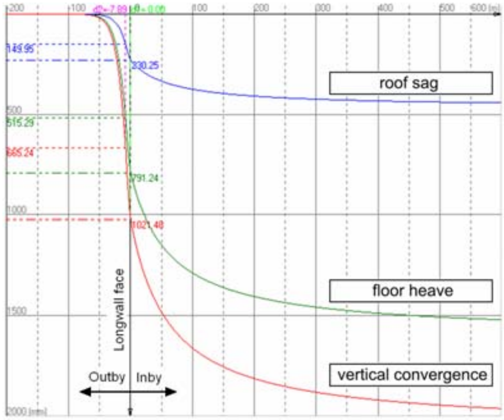
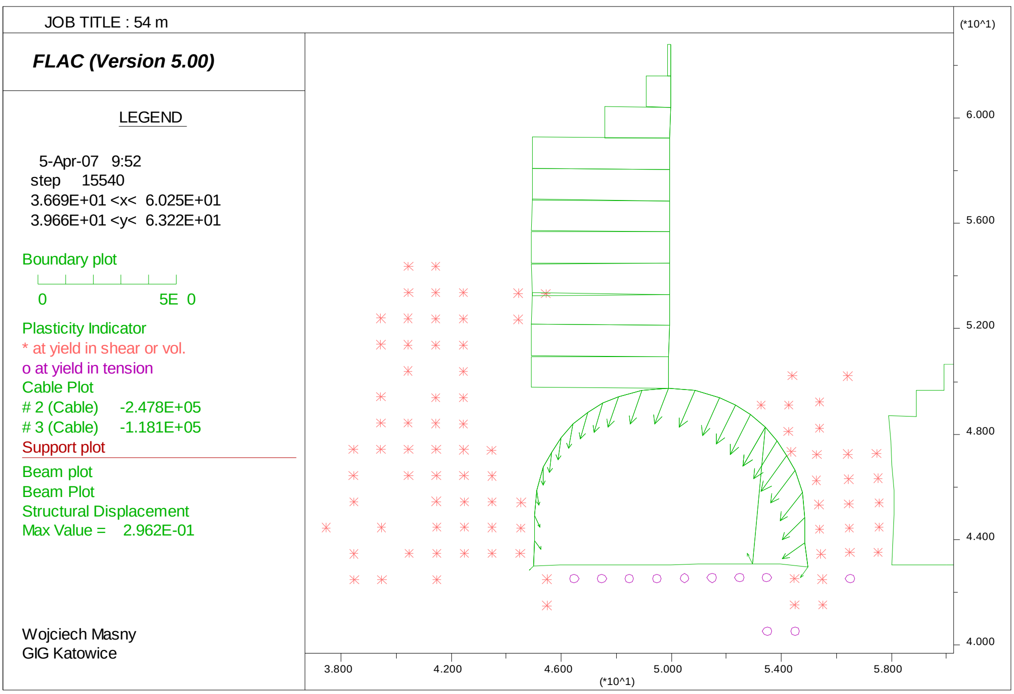


Figure 14. Results of computations of the course of gateroad convergance outby and inby the face based on empirical method worked out at GIG (Prusek 2007).

(a)



(b)

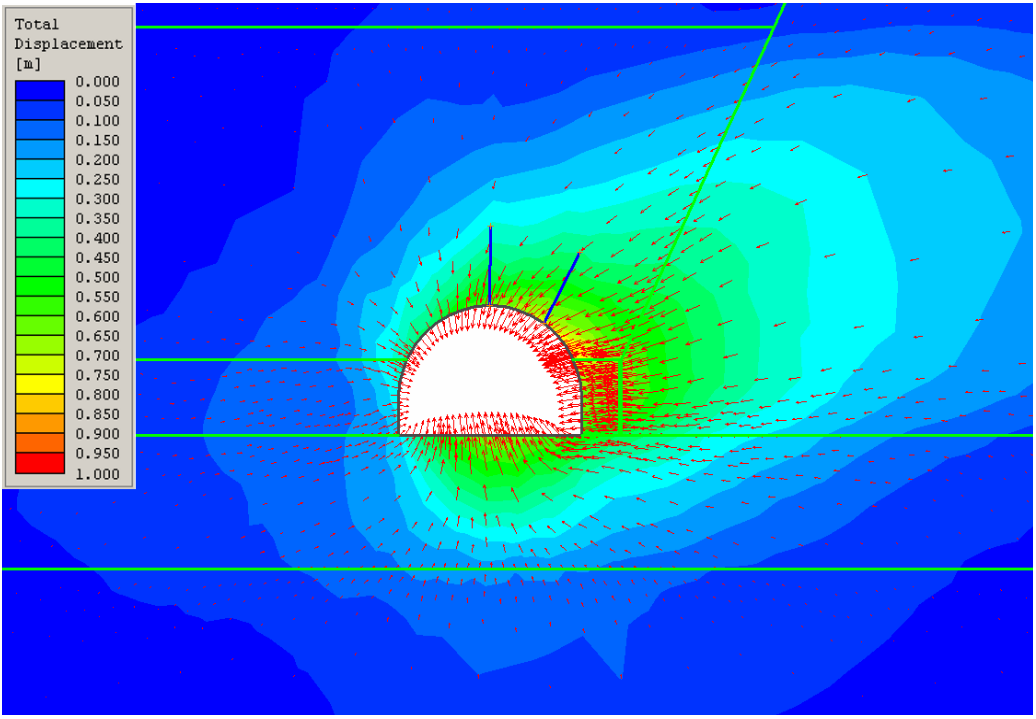


Figure 15. An example of numerical modeling of the rock mass around gateroads; (a) zone of rock mass damage in the surroundings of the gateroad, axial forces and vectors of support displacements at a distance of 54 m inby the face (Prusek & Masny 2007), (b) displacements of rock strata into the gateroad at a distance of 100 m inby the face (Prusek 2008c).

Both in the case of applying empirical methods and numerical modeling, the results are verified based on currently performed measurements of gateroads deformation. Such measurements are performed both manually, by means of simple measuring devices, and using special research equipment that enables continuous measurement of roadway deformations. An example of such equipment is a monitoring system developed by GIG specialists within the framework of MONSUPPORT project, which was financed by the Research Fund for Coal and Steel. Within the framework of research work, a number of sensors and measuring devices were developed, enabling to measure loads and deformations of steel arch yielding supports and bolts, together with the system of data transmission to the surface (Prusek 2008b). A scheme of the monitoring system developed is shown in Figure 16.

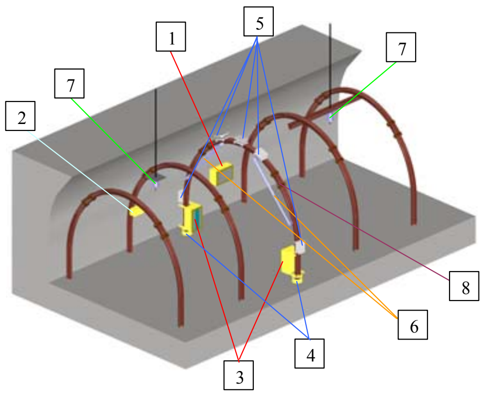


Figure 16. Scheme of monitoring system of roadway support: underground computer (1), module for support deformation measurement by means of the photogrammetric method (2), laser module (3), hydraulic dynamometers (4 and 5), displacement sensors to measure the slide in support friction joints (6), tensometric sensors of bolt load (7), tensometric sensors to measure the force acting at the support shackles' bolts (8). – (Prusek 2008b).

4 CONCLUSIONS

The overview of the support systems applied in the gateroads point at its relatively high diversity. In general, one can say that outside Europe, e.g. in such countries as the United States, the principal support type is rock bolting, while a characteristic feature for the U.S. mining industry is making several gateroads with pillars for one longwall panel. In the case of European countries, the situation is similar in the UK, where the mining industry was privatized, and continual striving for reducing costs of mining resulted in wide application of rock bolting. However, it should be noted that in the UK single gateroads are driven for each longwall panels and to maintain their stability, coal pillars are left between panels of the width being often more than 100m. In Germany, where an average depth of mining now approaches 1200 m, steel arch yielding supports are used as principal type of supporting gateroads. Recently nearly 70% of gateroads are developed in combined support (type A bolts and arches support). The use of concrete backfill on support arches is a standard in all roadways. In most cases, one of gateroads is maintained inby the face and repeated use for the next longwall panel. In such situations roadside packs are made along the goaf line. In Poland, similarly as in Germany, the principal type of gateroad support is steel arch yielding support. Over the last years, there has been observed a wider use in gateroads of additional rock bolting, most often for fastening cross-bar arches of steel support. More and more frequently, there are also used flexible bolts to provide additional reinforcement of arch support. It is predicted that due to continually worsening geological and mining conditions, and connected with this fact substantial convergence of gateroads, the combined support will be more and more often used, in particular with applying rock bolting. The steel arch frames will be made of larger size, using steel with enhanced mechanical parameters. Owing to a number of advantages, the mechanical lining on steel arches will also be more extensively used.

With the aim of proper designing the gateroad supports, the knowledge of convergence occurring in given conditions will be needed. For this reason, many methods have been worked out allowing to predict the convergence occurring in the gateroads, including the influence of longwall abutment load. In the last century, based on extensive underground measurements, and laboratory testing, a number of empirical methods have been worked out, and being applied in German or Polish mining industries. Over the last years, owing to development of computer technologies, numerical modelling has been widely used in computation of convergence in gateroads, or assessment of changes that take place in the rock mass surrounding those workings. Specialist computer softwares currently available allow to perform various types of computations and analyses, being necessary in selection of the gateroads support. However, when using numerical methods, it is necessary to keep in mind the possibility of making errors, for instance resulting from improperly assumed rock mass parameters in the computations. For this reason, it is considered indispensable performing periodical verification of numerical computation results, on the basis of underground measurements. Such an approach allows to calibrate numerical models and to reduce the prediction errors.

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Natural and mining factors that define quality of black coal for heat power stations

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ABSTRACT: Demand for quality of anthracite coal, its basic formation and provision of necessary level of quality, taking into account mining-and-geological and mining conditions, demands of market and existent standards are considered.

1 INTRODUCTION

Production and economical indices of power stations depend on quality of the fuel. Unsubstantiated quality level of coal in modern conditions can lead to economical failure of the enterprises-manufacturers and consumers.

Conception of the quality is complex, multiple-factor and can be described by only one index. Integral characteristics of coal quality for power engineering can be calorific value of working fuel. It is the most important consumer feature of the fuel for its usage in power engineering.

Most important characteristics of coals are calorific value, grindability and reactivity which are connected with petrographic compound and metamorphism level of coals.

2 BASIC INFORMATION

There is higher calorific value of coals, i.e. combustion heat of organic mass of coal (without admixtures) and lower calorific value of working fuel, i.e. calorific value of coal taking into account its ash and moisture content.

Higher calorific value is connected with elementary compound of coals and their metamorphism level.

Lower calorific value of working fuel is determined together with highest calorific value of coal and its ash and moisture content, composition of mineral admixtures and other elementary indices of quality.

In practice of energy calculations, the following formula for determination of the lowest calorific value of working fuel is used (by means of recalculation of the highest calorific value, taking into account the amount of ballast admixtures):

$$Q_p^n = Q^v \frac{100 - A^d - W}{100} - 0.025W \text{ , MJ / Kg,}$$

where Q^v – highest calorific value of coal; A^d – factual ash content of coal in air-dry state, %; W – moisture content, %.

Requirements demanded for coal quality are determined by technologies of its use. The most widespread method in coal energetics is method of burning of preliminary dried and grinded to 0.1 mm fuel in flash furnaces. Burning of coal-dust flame is performed in the flow in which high efficiency of oxidation is reached that is conditioned by developed surface for the reaction of grinded fuel with oxygen.

In general, all types of coal can be used for dust flaming. Their ash content varies in range from 20 to 45%.

Expenditures for technological material (fuel) rise and the consumption of fuel for power generation rises too, also its cost, tariffs for transportation, volumes of its processing in the warehouses and in technological process itself.

Decrease of fuel consumption, except for increase of economical indices, leads to positive integral effect.

It is known, that with increase of ash content, it is required to buy and process larger amounts of natural fuel with high content of ash.

Consequently, the power station processes more slags, captured ash with corresponding expenses of means and labour. But ash brings many harmful components that trigger premature erosive and corrosional wearing of the equipment.

During designing of heat power stations, fuel resources of the region is taken into account, and also composition of coal and their ash content. Thus, each of them can have economical operation mode with fuel quality corresponding to the project values.

With decrease of coal quality, in order to keep thermal capacity of the boiler unit and provision of required parameters of burning it is required to additionally burn more high-calorie fuel: natural gas or fuel oil that causes additional costs.

For stimulation of fuel quality increase for power stations in Ukraine in 2002, the state standard for hard fuel is approved (Technical conditions 2002), according to which, the coal for dust burning on thermal power stations is divided into 4 categories based upon conformity of basic indices of quality to project requirements of active coal-dust boiler units.

Coefficients of energy value of a unit of lowest calorific value are introduced for each quality category, correspondingly: 1.00; 0.75; 0.50; 0.30. These coefficients should consider costs for coal enrichment and additional expenses of thermal power stations for fuel burning with quality lower than the project one and can be used for price formation (Pilova 2004 & Sinyakovich 2004).

In the first quality category there are coals quality of which provides stable conditions of dust burning without addition of natural gas or fuel oil. Lowest calorific value of these coals must be not less than 20.097 MJ / kg.

In the second quality category there are coals that have lowest calorific value equal to limits from 17.585 to 20.097 MJ / kg. These coals are also good for dust burning but only with addition of natural gas or fuel oil. The third quality category contains intermediate products with lowest calorific value not less than 16.747 MJ / kg, the fourth category – slags with lowest calorific value not less than 12.560 MJ / kg. With that, general moisture of the coals of the first, second and third categories should be not more than 12-14%, intermediate products – 9% and slags – up to 18%.

Analysis of these data and their comparison with ash content of raw coal ash shows that only a little part of them can be attributed to the second category without enrichment and practically all extracted coal does not correspond to the first quality category.

Thus, from point of view of fuel cost decrease during power generation, one should look up to use of first category coals. Thus, all extracted coal for energy purposes should be enriched.

Problem of quality and effectiveness of black coal and anthracite usage is defined by both natural conditions (mining-geological, seams thickness, their ash content, sulphur content) and used extraction technologies.

Fossil coal presents solid system of organic matters and inorganic components (minerals). Therefore, composition and amount of mineral part are important characteristics of coals. They define coal usage in industry and also characteristics of gained slags and ash losses during burning (Eryomin & Bronoviec 1994.).

Mineral components of coals are conditionally divided into two groups: macro- and micro components (correspondingly with their content in mineral matter more or less than 1%). Macro elements are S_i, Al, Fe, Ca, Mg, S, and sometimes N, Ka, Ti. Mineral components are deposited as rock layers, lens, burs, organomineral compounds. Mineral formations in coals can be presented also by mineral fragments and rocks as mudstone, siltstone, sandstone and limestone.

Mineral matters, just like moisture, is a ballast during transportation of coals. In addition, they represent the source of an ash formation, quantity and composition of which can significantly influence possibility of coals use in production.

High coal ash content worsens working conditions of energy units. Ash leads to decrease of coke quality and, consequently, indices of blast furnaces: consumption of coke increases per ton of produced cast iron. So, with increase of coke ash content by 1%, coke's consumption increases approximately by 2% and, correspondingly, productivity of a blast furnace decreases.

Properties of ash and slag depend on mineral components containing in coal, ash-forming components, and also rocks contaminating coal during an extraction. Composition of ash considerably differs from mineral component of coals that undergoes significant changes during oxidation and thermal decomposition during burning.

Basic components of coal ash are oxides: SiO_2, Al_2O_3, Fe_2O_3, MgO, CaO, Ka_2O, Na_2O, TiO_2, SO_3. Besides, there are rare and dispersed elements in ash: nonferrous metals, noble and radioactive metals.

For many coal deposits there is a correlation between ash content and ash composition. During burning process on large power stations, 60% of inorganic components transfer into ash and slags. Mineral component is removed from coals at enrichment plants. At this, around 40% of inorganic components are directed to wastes as a rock.

Sulphur is enclosed in fossil coals and coal rocks as sulphides, sulphates, organic compounds and elementary sulphur.

Sulphide sulphur is enclosed in coals basically as pyrite and marcasite. Sulphite sulphur basically occurs as $CaSO_4$ and $Fe_2(SO_4)_3$ in a small amount (0.1-0.5%) and makes up 10-12% of general sulphur. Its content significantly increases due to weathering and strong coal oxidation. Elementary sulphur has 0.03-0.20% in coal of Donbas and is basically located in fine-dispersed form.

Possibility of coals usage depends on ash content in coals. During energy usage of coal all types of sulphur except of sulphite one transfer into SO_2 and is removed together with combustion gases and that causes wearing of the chimneys, boilers and equipment, and also leads to negative influence on the environment.

Extraction of black coal has a line of peculiarities and they are determined by a row of factors: spatial change of mining operations, dependence of results of mining production on characteristics of a coal seam and mining-geological and mining-technical conditions, low accuracy of information about object of mining production.

Listed factors influence the change of extracted coal quality.

Quality of gained coal is formed under aggregate influence of a big number of factors that are combined in groups: natural, economical and technological factors.

Each of the factors except natural can be attributed to manageable. Manageable factors are those that can be influenced to change coal quality and its stability relative to the desired level. Unmanageable factors cannot be influenced by any manageable actions. They contain group of natural and economical factors. They are connected with the consumers in given kind of coal production, with its amount on the market and also with prices fluctuations and other economical changes.

Economical factors determine, in general, effectiveness of coal deposit development, rationality of acceptance of this or that technical, technological and organizational decision. But, in its turn, economical indices of extraction and process of coals depend a lot on natural and technological factors and on the market conditions. So, economical factors can be determinative and secondary during management of quality of production.

Natural and economical conditions are dominating for coal quality determination. In addition, level of mining operations technology significantly influences economical results. Development of geotechnologies creates preconditions for decrease of natural factors influence level on the coal quality (selective extraction, improvement of underground transport schemes and so on).

Except of mentioned natural, technological and economical factors, there are organizational, social, political and other factors that have an influence on the quality of coal industry. Quite often, a coal with lower quality is economically reasonably to extract under conditions of deficit for fuel in order to provide energy independence and safety of the government in extreme conditions, either because of the imperfect mechanism of the prices adjustment.

Character feature of the coal industry production is its single and multipurpose use. In connection with this, one should differentiate such categories of coal quality as theoretical, consumer and integral.

Theoretical quality of fossil coals are defined by aggregate of their objective properties such as petrographic composition, chemical composition, heat value, physical-mechanical and technological characteristics.

Consumer quality evaluates properties of coal industry one-sidedly, considering it only from a position of concrete consumer, based on the level of technology and economics. Each enterprise-consumer is interested in usage of high quality coals as its own production costs reduce.

Unlike consumer quality, integral quality is defined on the basis of accounting of indices of productions that take part in creation of the end product. For example, mining, processing (enrichment), metallurgical, energy and others. The basis of the integral approach of assessment of the production quality should be aspiration to receiving the summarized effect from the whole chain of adjacent productions, providing thereby minimum of labour and material costs for creation of the final product and also rational use of the earth resources.

Optimal quality defines such aggregate of consumer features of coal that provides the most profitable indices of the end product taking into account expenses for extraction, enrichment and processing of the end product.

Efficiency of coal usage is defined by not only its composition and properties of organic matters but also by amount and composition of accompanying inorganic components that are considered as ballast and harmful admixtures. They are also a reason of environment pollution.

Coal after being extracted represents by itself a mechanical mixture of coal and rock pieces of different sizes and also pieces of intermediate fractions.

Pollution of coal by fractions is conditioned by technology of its extraction and structure of a coal seam. Ash content of fractions is different – for coal it is equal to an internal ash content substantiated by mineralization of a coal matter. Otherwise, this ash content is called mother ash content and ranges from 3 to 15%. Ash content of intermediate fractions makes up 25-45% and is determined by level of coal saturation with admixtures of host rocks and products of their destruction. Ash content of a coal rock ranges in quite wide limits: from 70 to 92% and is conditioned by level of its saturation with organic matters.

Presence of mineral admixtures in coals substantiates not only increase of their ash content but also their density. Coal ash content, i.e. mass or weight fraction of incombustible remains after burning of a coal matter is conditioned by mineral admixtures in coals.

Mineral admixtures have different origin. So, depending on this, they are divided into internal (internal ash) and external (external ash). The source of external ash is different salts, coverings of mineral matters that took place during accumulation of plant residues in a process of coal creation. The sources of internal ash are interstratified rocks in a coal seam and also roof and bottom rocks that is substantiated by applied technologies of coal extraction.

Coal fractions are those that have density not less than 1500 kg / m^3, intermediate fractions have density of 1500-1800 kg / m^3 and natural fractions – more than 1800 kg / m^3.

Separation of rock fractions from coal ones is implemented according to their density difference (Reference book for coals enrichment). In practice of coal enrichment, gravitational methods of separation are applied for these purposes based on the difference of interaction of separated materials pieces with liquid medium that have diverse density and size. Dense-media separation is mostly used for big classes of coal that is carried out in magnetite suspensions. For small classes – heavy-media hydrocyclones, spiral separators. For sludges (particles less than 0.5 mm) – flotation is used or hydraulic classification.

Peculiarity of black coal deposits in Ukraine is a big stratification depth, low thickness of coal seams and their complex structure. These factors predetermine respectively high prime cost and big ash content of a rock mass. Depending on these conditions and also taking into account fuel deficit and social-economical problems of mine regions, factual ash content of a rock mass ranges in limits of 35-55%. So, based on the mentioned demands of the consumer to the quality of coal industry, virtually all gained coal has to be enriched. As the concentrate yield defines volume of finished products, so its value considerably influences economical indices of coal output. With other equal conditions, concentrate output decreases with increase of ash content of enriching coals.

3 CONCLUSIONS

So, based upon the executed analysis, the following conclusions can be made:

1. Quality of extracted black coals is conditioned by natural properties of fossil coals, mining-geological conditions and mining-technological factors of extraction.

2. Black coals of Ukraine's deposits are not worse than other world's deposits by their quality defining indices, except increased sulphur content.

3 One of the most important factors that define calorific value and technological suitability of black coals, is their ash content consisting of internal and external ash. Internal ash is conditioned by natural factors that are considered as uncontrollable. External – to mining-technological and economical that are controllable and partially controllable.

4. Decrease of external ash content is possible by means of proper extraction technologies and separation of ballast admixtures from coal at processing plants. Coal industry of Ukraine has processing plants with total power increasing modern level of production output.

5. Enriched coal has higher consumer properties and even with existing economical conditions and formed prices, Ukrainian coals can be competitive. But quality level should be economically substantiated for each kind of coal production.

6. Rational quality is reached with full separation of polluting rock fractions with concentrates receiving and ash content being equal to rock mass ash content that corresponds to the world's practice (Pilov 2001, 2002 & Pilova 2002). But due to high dilapidation of Ukrainian power stations, application of such fuel is not rational without their reconstruction.

7. Prevailing factor of coal extraction economical indices increase is ash content of raw coals, rational level of which has to be substantiated taking into account factors that determine prime cost of coal production and demands of its consumers. Thus, rating of raw coals ash content has to be based on new principles considering not only mining-geological conditions of extraction but also their grade composition, following technologies of enrichment, demands of coal production market.

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Influence of advance rate of high-output longwalls on methane abundance of mine workings

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ABSTRACT: Research results focused on increase of reliability and authenticity of output prognosis in highly productive long faces based upon laws of methane discharge into longwalls with outputs more than 3000 t / day are presented.

1 INTRODUCTION

Experience of designing and mining of longwalls that have working load of more than 3000 t / day under conditions of seams thickness equal to 1.8 m and stratification depth of more than 800 m, has shown that the results of calculation of expected methane debit and its distribution along sources of emission based on the natural methane content, does not correspond to actual data. It can be explained in the way that calculation methods, mentioned in existing regulatory documents, to the great extent, are based on empirical dependences received in longwalls that advance with rate of 6 m / day. Usage of methane abundance prognosis method, based on such dependences, at high speed of advance can lead to significant errors.

Provision of high output of a longwall is one of the main trends of efficiency increase of mining operations. Achievement of high technical and economic indexes using combined mining method, including extraction panel development and retreat system of mining of longwall with direct-flow ventilation scheme.

2 CONDUCTION OF RESEARCHES

A fresh air stream is directed into the longwall along two workings, meanwhile one of the workings is maintained behind the longwall and the other one is filled after its advancement. During development of the extraction panel it is best to drive one conveyor drift, provide its protection and reuse it during mining of lower longwall to the dip. Such mining method satisfies modern requirements of mining operations safety.

In recent years the author has executed experimental works dedicated to the comparison of actual methane debits in longwalls with computational data. It is taken into consideration that while choosing research object, in high-output longwalls, ventilation schemes with additional fresh air stream mixed with outgoing air stream are used as a rule, and also effective methods of complex degasification of adjacent seams and worked out areas are used as well. Methane debit in longwall is the basic factor that restricts extraction of coal.

Information about actual methane debits in workings is received based upon gas surveys performed at extraction areas of mine "Krasnoarmeyskaya-Zapadnaya #1". This mine is the biggest mine of Ukraine and it develops single gaseous seam d_4. Daily average output of the longwall exceeds 3000 tons, taking into account that several longwall's output reaches 5000-7000 t / day. The mine is considered to be extremely dangerous according to methane factor, dangerous by coal dust explosions, coal, rock and gas outbursts. During mining operations and increase of the operational depth, absolute gas abundance of mine is increasing from 193 m^3 / min in 1999 up to 296 m^3 / min in 2008.

Predicted gas abundance reaches 350 m^3 / min in the end of 2010. With coal's methane content being 16-20 m^3 / t of dry ash-free mass, methane emission at extraction areas exceeds 45 m^3 / min. Meanwhile, consumption of gaseous mixture and methane content ware checked in workings and degassing pipelines with interval of 30 minutes, using standard methods and devices.

Survey points were located in longwalls 10-15 meters from ventilation working and in outgoing air streams of extraction areas, daily average coal output was being registered during survey time. Data of methane content and air consumption registered by automatic control devices were added to the results analysis.

General average debit of methane at the area ($\bar{I}_{area}$) was determined as the sum of average debits during observing time of methane debit in outgoing ventilation stream of an area ($\bar{I}_{area}$) and in degassing gas pipeline ($\bar{I}_{deg}$). Methane debit that discharges from worked-out area is calculated according to the next formula:

$$\bar{I}_{w.a} = \bar{I}_{area} - \bar{I}_{st} \text{ , m}^3 \text{ / min} \quad (1)$$

Average methane debit was compared with debit-calculated according to natural gas content taking into account seam's ash-content and moisture content. Initial data presented by geological service of the mine were used for calculations.

Analysis of the results has shown (Table 1, Figure 1) that actual debit of methane in a stope can significantly differ from the computational one. This difference increases with rising of rate of coal production. With production of up to 500 t / day, the debits virtually coincide. With increase of output up to 4000 t / day and more, actual debit makes up about 30% from the calculation debit.

Table 1. Gas balance of extraction.

Names of stopes	output	Methane debit, m^3 / min			calculation debit in a stope	Ratio of actual debit and calculation debit
		At the area	In a stope	In worked-out area		
Southern of block #6	470	14.0	4.2	9.8	4.2	1.0
1 southern of block #5	1100	17.9	5.1	12.8	6.2	0.82
6 northern of block #6	1087	18.8	5.3	13.5	6.5	0.82
4 southern of block #3	1400	19.5	4.0	15.5	7.1	0.56
1northern of block # 8	2000	26.0	5.2	25.8	10.9	0.48
2 northern of block #5	2130	29.6	5.3	21.5	11.9	0.45
4 northern of block #5	2500	32.2	4.8	27.4	12.6	0.38
1 stope of block #8	2540	26.6	4.0	23.7	12.8	0.31
2 southern of block #5	3000	40.1	4.8	35.3	13.4	0.36
4 northern of block #2	3000	34.3	4.1	30.2	12.7	0.32
2 southern of block #2	3480	35.5	4.2	31.3	13.8	0.30
3 southern block #3	3500	33.9	4.0	29.9	13.6	0.29
3 northern of block #5	3970	40.2	4.1	36.1	14.0	0.29

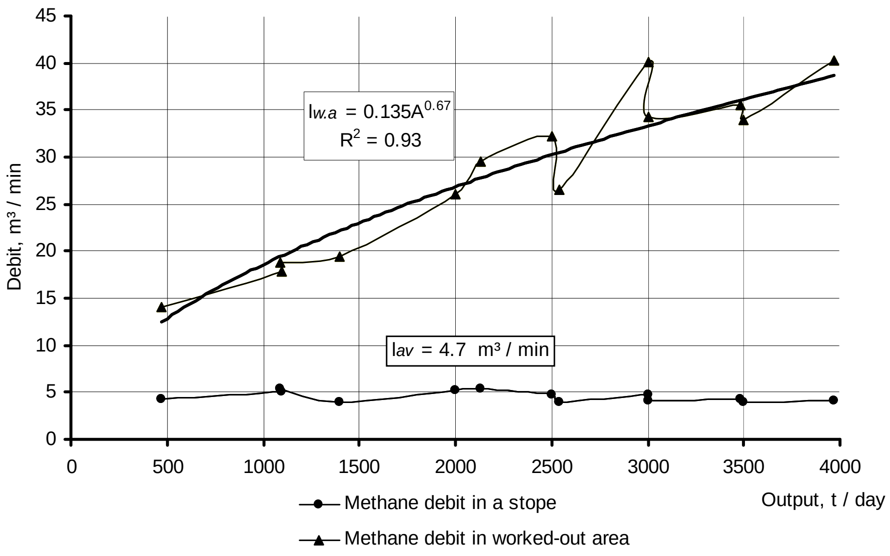

Figure 1. Dependence of methane debit in a stope on coal production rate.

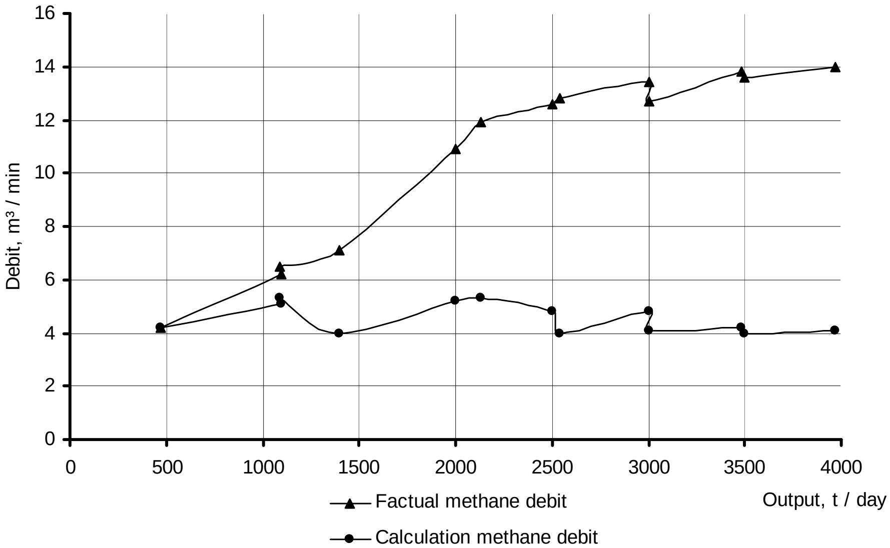

Figure 2. Dependence of methane distribution in workings on coal production rate.

Increase of coal production rate is followed by methane emission rise from adjacent seams and gas-bearing rocks as a power function, while absolute methane debit in a stope virtually does not change, remaining at level of 4.7 m^3/min (Figure 2).With that relative debit decreases from 13.5 down to 1.7 m^3/t.

Residual methane content of coal that is transferred outside of the stope is calculated according to the following formula:

$$X_0 = \frac{100 - W - A_{ash}}{100} \cdot 18.3 \left(V^{daf^{-0,6}} \right), \tag{2}$$

where $W = 3\,\%$ – moisture content of coal; $A_{ash} = 15\,\%$ – ash content of coal; $V^{daf} = 30\,\%$ – volatile matter content

$$X_0 = \frac{100 - 3 - 15}{100} \cdot 18.3 \cdot 30^{-0,6} = 2.2 \text{ m}^3/\text{t}$$

Having natural methane content of coal equal to $X = 17\,\text{m}^3/\text{t}$ in boundaries of the working area, the following amount of methane should discharge from an extracted seam:

$$q_{seam} = (1 + k_{ex}) \cdot (X - X_0), \tag{3}$$

where $k_{ex} = 0.05$ – coefficient of exploitation losses that take into account part of methane discharged from coal in worked-out area

$$q_{seam} = (1 + 0.05) \cdot (17 - 2.2) = 15.5 \text{ m}^3/\text{t}$$

That 1.15-2.1 times greater than actual number at output of 500 and 4000 t / day correspondingly. First value is within the limits of error prognosis. The second value is an evidence of an obvious error in either determination of initial gas content of coal in its zone of extraction or in residual gas content outside of the working area. An error in both parameters is not excluded though.

Decrease of initial gas content can be explained by flowing of liberated methane from off-loaded face area of the seam into worked-out area through cracks in roof. The researches performed by the scientists of various countries and approved by the experience of mining operations, have allowed to determine that roof movement essentially decreases during increase of stope advance speed down to 4-5 m/day, its further increase virtually does not influence the movements. Fast longwall advance leads to more intensive movement of roof but with that its subsidence decreases, fracturing in rocks develops slower, rocks deform less, due to that fact sustainability of exposed roof increases and conditions of its management improve.

Increasing of longwall advancement from 4-5 to 10-12 m / day virtually does not influence roof's subsidence.

In case of a high advance speed of the longwall (more than 200 m / month) the roof rocks do not break in face area. Increase in face advance rate substantiates decrease of fracturing, face slip in face area of the seam and decrease of roof movements, increase of step of primary and secondary roof cavings, but it can contribute to accumulation of potential energy of reversible deformation in host rocks. With all that, accumulated energy can manifest itself during the rocks breakage with dynamic effect.

Presence of such phenomenon is proved experimentally. Radioactive gas (crypton-85) injected into a seam on a distance of 35 meters ahead of the longwall, was found in check boreholes within 10 meters along the strike of the seam at the 5th meter above the gas injection place. When the face advances by the distance of 18 meters from the injection place, radioactive gas used to be found in air leaks through worked-out area. Flow of methane through cracks in roof rocks above the longwall was found as well by means of boreholes drilling from this working that cross only immediate roof that has thickness 6 meters. Methane was discharged along the boreholes by means of natural pressure with debit of 0.27 l / min which increased up to 5.1 l / min after the shearer's passing.

Process of methane discharge needs to be carefully explored. Decrease in methane discharge from a mined seam depends not only on face advance rate, seam thickness and mechanical features of roof rocks, but also on the effectiveness of roof's gas drainage by boreholes. Exploration of this process is important because an error in methane distribution that discharges from mined seam between the stope and worked-out area does not only decrease admissible coal production rates depending on gas factor but also decreases requirements for worked-out area's gas drainage.

3 CONCLUSIONS

Increase of output in the stopes reduces staying time of the broken-down coal in the boundaries of stope and that is why residual methane content determination method needs to be specified as well.

Performed researches allow to conclude that an existing method of methane abundance prognosis is overstated and this does not allow to use it with the output more than 1000 t / day. Under such conditions it is reasonably to determine expected methane debit in designed longwalls based upon actual debit during stoping in comparable mining-geological and mining-technical conditions.

Because of this, it is logical to correct working standards using which it is prescribed to determine actual methane debit in a longwall "on a distance of 15-20 meters from a longwall", because methane flows to this area not only from a longwall but also from a part of a worked-out area. Debit of methane that flows into a longwall, is suggested to measure in this working on a distance of 10-15 meters from a ventilation working. This will allow to increase safety and reliability of methane abundance prognosis, and to optimize complex of actions that prevents gas danger, that, in general, will promote increase of effectiveness and safety of mining operations.

New Techniques and Technologies in Mining – Bondarenko, Kovalevs'ka & Dychkovs'kyy (eds)

Strength calculations of block elements of room-and-pillar mining under permafrost conditions

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ABSTRACT: The paper covers the results of rheological properties of gypsum under natural conditions of permafrost and stress-and-strain conditions of the rocks of room's roof in gypsum quarries depending upon the thickness of the roof. The recommendations concerning room and pillar parameters are given.

1 INTRODUCTION

The research of both strength and rheological characteristics of rocks under specific mining and geological conditions helps to determine optimum dimensions of rooms and pillars for long-time strength.

Rheological properties of rocks are one of the most important factors that determine stress-and-strain conditions of rock massif to develop underground workings. Mining pressure in underground constructions, time effect on underground workings roof stability, displacement of rock mass while mining, etc. are the results of changing of mechanical properties of rocks connected with their rheological properties.

Rocks of Olekminsk deposit (Yakutia, Russian Federation) are situated in permafrost zone. Rheological properties of the gypsum deposits have not been studied yet. The most important is the research of rheological properties of frozen gypsum. It is known that ice is very creepy. One can suggest that frozen gypsum has the same properties under sufficient ice content.

Thus, the research of rheological properties of frozen gypsum is rather important to substantiate parameters of rooms and pillars.

2 STUDYING OF THE RHEOLOGICAL PROPERTIES OF GYPSUM

As the effect of positive temperatures changes, rheological properties of rocks testings were performed under natural conditions of quarry. In order to do that, one of the quarry rooms was used as a research laboratory. The temperature in the room was three degrees below zero Centigrade. The laboratory was located in such a way in order to avoid the effect of mining process on the research results.

Cores were selected from monolith. Cores were prepared manually. Tests were performed on spring-controlled stands. Samples were loaded using hydraulic jack. Jack pressure was measured with the help of manometer. Deformation was determined by means of clock indicators.

Before creep tests the strength of gypsum as for uniaxial compression was determined. The information is required for preliminary choice of loads on the sample (Erzhanov 1964). Fracture load was 110 MPa. 15 samples were taken for rheological tests. Three similar samples were under each load. Value of the loads was 0.3; 0.4; 0.5; 0.7; 0.85; 0.95 of fracture value. Rock creep tests lasted for 150 days. Time diagrams of deformation under constant load that is creep curves were developed on the basis of the data (Figure 1).

Creep curves have three specific areas: initial curvilinear area belonging to non-stationary creep; rectilinear area called stationary creep, and terminal area of curves belonging to progressive creep which results in fracture.

Rheological tests of frozen gypsum show that it has noticeable creep under loads being more than 0.4 of fracture one. If load was 0.3 of fracture the deformation would stop after 20 days and creep would not be available. The results give the ability to make the conclusion. Strength margin equal to three may be taken if durable strength of construction and complete elimination of gypsum creep are required.

If loads are 0.5 to 0.85 of fracture one, then creep rate is not important and irreversible deformations of pillars grow slowly. If it is required to keep the construction stability during several decades then strength margin can be decreased up to two. In this case the pillars will deform themselves. Their height will be decreased as a result of plastic deformation but fracture will not take place. Pillars will keep their carrying capability.

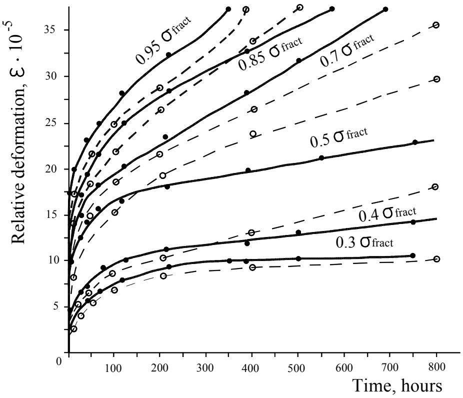

Figure 1. Curves of Gypsum Creep
— Experimental Curves;
– – Design Curves.

To determine absolute deformation of pillars in a long time the experimental data concerning gypsum creep are shown as a functional relation.

For sedimentary rocks congenital linear creep with sufficient degree of accuracy is described by an equation where creep memory function is expressed as exponential function which reflects initial stage of the rock creep process completely (Erzhanov 1964):

$$\varepsilon(t)=\varepsilon(\sigma_0)\left(1+\frac{\delta}{1-\alpha}t^{1-\alpha}\right) \qquad (1)$$

where $\varepsilon(\sigma_0)$ is the sample of deformation under rapid load up to stress value; α and δ are the equation parameters (rheological characteristics of rock).

To determine α and δ parameters calculation method from (Guidelines 1972) was used. Initial data are:

- the value of initial deformation $\varepsilon(\sigma_0)$ for $t=0$;
- deformation period;
- those corresponding to chosen deformation periods.

Creep curves for the loads under which experimental curves were obtained according to calculations of relative axial deformations were developed. The results are shown in Figure1.

Comparison of design data and results of natural research demonstrate good coincidence of the results. The deformation value coincides rather accurately if load is 0.3 σ_{destr} and if loads are more than 0.85 σ_{destr}. If loads are within 0.4 σ_{destr} and 0.7 σ_{destr} calculated deformation value is some higher. As it increases strength margin, one may say that the calculation method reflects gypsum creep adequately.

Obtained values of relative deformations were used to determine absolute deformation of pillars with given height after 50 and 100 years. The results are in the Table 1.

Table 1. Relative and absolute deformations of pillars under durable loads in gypsum quarry.

Load in parts of fracture one	50 Years		100 Years	
	Relative deformation, $E \cdot 10^{-5}$	Absolute deformation, mm	Relative deformation, $E \cdot 10^{-5}$	Absolute deformation, mm
0.3	80.5	6.4	99.4	8.0
0.5	244.7	19.6	328.3	26.4
0.7	517.8	41.4	697.5	56.0

3 EFFECT OF ROOF THICKNESS ON STRESS IN IT

It is not difficult to determine dimensions of interchamber pillars while substantiating room parameters. Results of numerous research show that simple calculation methods are expedient for gypsum deposits (Melnikov 1964 & Methods 1962).

It is much more difficult to determine maximum allowable room distance. The issues are highlighted completely in construction standards and regulations where room roof is considered as a slab with equally distributed load. Gypsum quarries don't have "false roof". Roof is rock layers with different strength and elastic properties. There is some kind of tie between layers that is rocks transit into each other gradually. Listed peculiarities should be taken into account to calculate strength of room distance.

If roof rocks are weaker than gypsum then the gypsum ceiling is left in the roof, if it is necessary to test strength of the ceiling.

Finite element method (Zienkiewicz & Taylor 2000) is used to calculate ceiling stresses if room distance in gypsum quarry is constant and roof thickness is 1, 2 and 3 m. The calculations are required to determine effect of roof thickness on stress in it.

The results show that if roof thickness increases, the stresses in it increase too.

Calculation method of Privarnikov-Philippov (Philippov 1979) was used for similar conditions. Calculation results on two methods are in the Table 2.

Stress values in the ceiling calculated according to the two methods may be considered as equal ones. The data are used to obtain the dependence of tension stresses in ceiling depending upon its thickness variation (Figure 2).

Table 2. Results of ceiling stress calculations.

Method	Tension stresses (MPa) in room roof is ceiling thickness is		
	1 m	2 m	3 m
FEM	5.91	8.90	9.10
Privarnikov-Philippov	6.17	7.05	8.23

Following conclusions are possible. Usually, the ceiling is considered as a support beam or a slab loaded with equally distributed load. In this case stresses decrease if thickness increases. Actually stratified medium with layers tied is available in room roof. The analogy with support beam is not expedient. Besides if roof thickness increases, the height of room decreases, and it results in stress increase in roof.

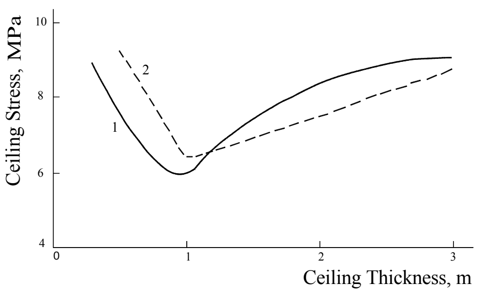

Figure 2. Dependence of tension stresses in room roof on ceiling thickness: 1 – on FEM; 2 – on Privarnikov-Philippov method.

4 SUBSTANTIATING DIMENSIONS OF ROOM AND PILLARS

The results were used to calculate strength of interchamber pillars and room distance of gypsum deposit by the finite element method.

Design values of compressive stresses within pillars and tension stresses within distance center were compared with allowable stresses as for compression and tension for rock. Elastic characteristics of rock taken for calculations are in the Table 3.

Finite element method was used to calculate stresses in the construction. The strength margin equal to three is determined to correspond to 4m pillar width, 8 m room distance, and 1 m thickness of gypsum ceiling.

Table 3. Elastic characteristics of gypsum and rocks of Olekminsk deposit.

Rocks	Elastic modulus, $E \cdot 10^4$ MPa	Poisson ratio
Gypsum	38	0.7
Dolomite	50	0.8
Argillite	11	0.30
Siltstone	20	0.13

Finite element method helped to take into consideration the effect of rock stratification on stressed state of pillars and room roof. Besides, the method allows determining stresses around rooms having complicated forms (Figure 3).

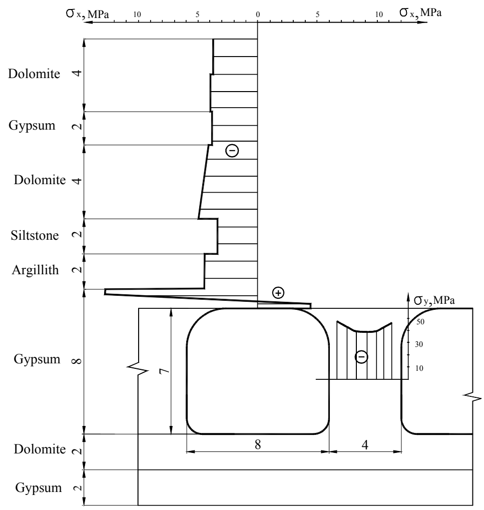

Figure 3. Design diagram and stress epures in roof and pillar rooms.

5 CONCLUSIONS

The research helps to come to the conclusions. The research of rheological properties of frozen gypsum shows that under 0.35-0.85 loads of fracture one the rate of stable creep is not important. In 50 years the absolute pillar deformation will be 41 mm if load is 0.7 of fracture one. Creep stops in 15-20 days if loads are less than 0.35 of fracture one. In 100 years absolute deformation of pillars will be 26 mm as a result of stable creep under load equal to 0.5 of fracture one. Strength margin can be equal to three if it is required to have durable strength of construction and to avoid gypsum creep completely. Thus recommended parameters of rooms are: width of pillars is 4m, room distance is 8m, and thickness of gypsum ceiling within the roof is 1m. Using the parameters of rooms for Olekminsk deposit helped to decrease gypsum losses by 30%.

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Dependence of effectiveness of development of mining operations on processibility of coal seams deposits with thickness of 1.2 m

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ABSTRACT: Questions of effective application of high-efficient longwall equipment in difficult mining-geological conditions of Donbass on seams with thickness up to 1.2 m depending on processability of development of deposits in mine field are considered and criteria of estimation by means of coefficients of processability and use of means of coal extraction mechanization are proposed.

1 INTRODUCTION

Donetsk coal basin of Ukraine is characterized by difficult mining-geological conditions: low thickness of coal seams, big mining depth, high gas abundance, propensity of many seams to sudden outbursts of coal and gas, weak country rocks. Mining operations are conducted at 25 mines (11.5%) at depths of 1000-1300 meters. Mining conditions at the reached depths are rather complex. Combination of mining-geological negative factors demands improvement of exploration, development and extraction technologies.

Thickness of mined coal seams is 0.65-2.39 m, seams with thickness of 1.0-1.5 m are prevailing. Increase of average thickness of mined seams is caused by increase in volumes of wall rocks cutting on seams with thickness less than 1.0 m, and, as a consequence, by increase of ash content of extracted coal. Coal refers to the class of humite and sometimes contains sapropelic-humus interlayers.

Mudstones are deposited in a roof, and sometimes siltstones, sandstones, rarely limestone are as well. The bottom of coal seams is presented by mudstones and siltstones. Coefficient of working coal-presence of a productive strata, in average, makes up 0.77.

At the present time, achieved high face outputs are related to mining of coal seams under favorable mining-geological conditions on mines of Donbass and in other coal basins of the world. An average level of mechanisation of stopings makes up about 65.8%, reaching 100% at some mines. High face output is reached on mines "Krasnoarmejskaja-Western", "Krasnolimanskaya", "of Zasyad'ko", "the Red Partisan" where daily average output from one face reaches 3-5 thousand tons, and daily advancing equal to 8-12 m.

2 PROBLEMS

Majority of the Donbass underground mines equipped with mechanized complexes, operate in hard mining and geological conditions. It leads to big delays and loss of working time. Expenses for purchases of complexes turn out to be not reasonable because of the incompatibility of mining-geological conditions to technical requirements of the given mechanisms. It is hard to achieve high results as the coal reserves do not correspond to the technology which is used by the mechanized complexes.

The task of stable and high efficient output is polyfactorial and cannot be solved by means of taking separate actions focused on liquidation of consequences of conditions changing especially change of face equipment in one extraction panel. A question of coal seams suitability for extraction by high efficient mechanized complexes is emerging and there is also a question of whether these seams are ready to be extracted.

3 ANALYSIS OF RESEARCHES AND PUBLICATIONS

In the work, the authors divide reserves of an extraction panel into categories and groups by stratification conditions of coal seams that judging by their thickness can be applied in conditions of Donbass, but judging by the conditions of geological phenomena and readiness of the reserves for the beginning of the extraction, such assessment is quite acceptable.

First of all, it is necessary to consider structural changes in mother rocks that can be encountered in mining extracted area during stoping. Geologic phenomena are related to such structural changes that need correction of the technology or performance of actions to bring the massif in accordance with technical conditions of implementation of mechanization means. Questions concerning readiness of the reserves for the stoping contain a whole complex of mining operations. Among them: drivage of extraction drifts, face entry and assembly chambers, installation of the equipment and transport.

These are the important components of a common system of a longwall functionality. But in order to provide high face output it is necessary to consider rock massif condition, its properties and process of formation of loads on a face support and support of extraction openings. Also possibility of immediate roof to collapse during exposure, inclination of the seams to geodynamical phenomena and coal to self-ignition.

For rational disposition of development openings it is desired to know the stress state of the tectonic block and orientation of horizontal stresses. At parallel disposition of extraction openings with respect to horizontal component of stresses, the costs for opening's maintenance decrease by 1.5-2.0 times (Baranov 2006 & Zabrodin 1981).

Given approach to planning of mining operations development is acceptable at the initial mining of a part of a mine field under conditions of already accepted project solutions, the awareness of the stress field location allows to choose density of support installation and predict costs for the extraction openings maintenance.

Analysis of structural composition of the host rocks massif and its change during the mining of the extraction panel is the basis when choosing set of face equipment based upon the criterion of the minimal resistance of a section of a mechanized support. Given criterion is established according to the weight coefficient of the roof taking into account areas with changes of structure of the host rocks and advance rate of the face. It is established that the face advance rate and structure of the host rocks have a significant influence on the rocks dislocation mechanism and on the character of load formation on the support elements.

There is a known method of maximum longwall output in which stability of the roof in the zone of the shearer's cutting depends on its width of cut (Baranov 2006). Depending on the readiness degree of the deposits for mining, there are four groups which they are divided into: highly producible, producible, low producible and non-producible. Application of highly productive face mechanized complexes for the last group of the deposits is not rational because it is unreal to gain high longwall output in this case. Presence of zonality in structure of the host rocks of the Donetsk basin indicates on the third group of the deposits where it is necessary to carry out measures directed to provision of the deposits produceability.

Taking into account possible variants of zonality combination in structure of the host rocks, performed measures should provide acceptable mining conditions for effective application of the face equipment during the whole process of the extraction panel mining.

4 RESULTS OF THE RESEARCHES

Effective application of the stoping technology intensification under mining-geological conditions on the big stratification depths is constricted by natural and anthropogenic formations of the host rocks structures. Classification of the structural massif changes zones is proposed for the performed measures focused on stabilization of the rocks in longwalls (Figure 1).

The basic formations of the natural character are the low-amplitude geological phenomena, zones of intensive fracturing and dynamic rocks displacements, lithologic differences and their thickness changes, and also zones of deep faults with possible water rupture into mine workings. The anthropogenic formations are the zones of lithological differences layering, dynamic rocks displacements, uncontrolled rocks cavings and heaving of the rocks.

Influence of these factors is displayed as a bottom heaving and deformation of the openings, rocks cavings into the working area of the face, difficulties in supporting the longwall's abutment with the stopes, load increase on the support's elements.

Effectiveness of the technology of highly-mechanized stopes is determined by the machine time coefficient, maintenance of the stopes without repairs and by the presence of local control of the host rocks massif condition with timely implementation of planned preventive measures. Considering an extraction panel as a solid object, there are four zones that can be distinguished: substitution of lithologic differences, liquidation of the water ruptures in fractured massifs and deep faults, and also sudden loads on the support during hanging of thick and hard rock layers.

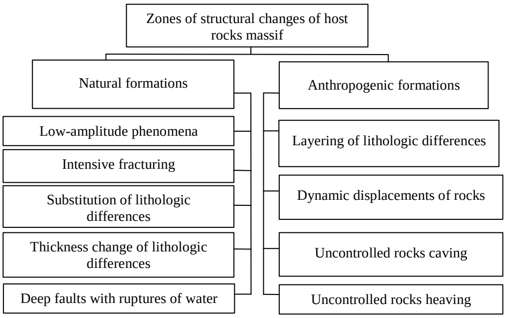

Figure 1. Classification of the zones of structural changes of host rocks massif.

Each of these zones has its peculiarities and requires extra resources and equipment for its successful overcoming. Zones with various structure of the host rocks massif differ by the character of rock pressure manifestation. Given feature becomes obvious during coal extraction by the highly mechanized face complexes which can move along the massif with quite high speed of the daily advance.

In order to provide stable face output and effective coal extraction it is necessary to establish produceability of the deposits in extraction panel and their correspondence to technical conditions of mechanization means. Such approach will allow to preliminarily define list of measures focused on increase of effectiveness and rationality of their implementation in concrete zones of structural changes. Length of the zones and their possible distribution area in massif, type of lithologic differences and their thickness are determined during the stage of designing of mining-geological chart of the extraction panel. Physical-mechanical properties of the host rocks are defined, stability of the immediate roof and bottom, possible water inflows and gas discharge into mine workings.

Disposition of cracks in a massif and their orientation according to the plane of a face are determined. During study of given factors it is necessary to consider all stratigraphy of rock massif in order to take into account mechanism of rock pressure formation. Level of credibility of the received initial information is sufficient for forecast of the possible rock pressure manifestation as when using pillar mining method the deposits are contoured by mine workings. Qualitative and quantitative indices are defined in order to describe condition of the coal.

Considering geomechanical task as the defining effectiveness of the mechanized face equipment application, it is necessary to provide produceability of the deposits by means of measures implementation in zones and their distribution along an extraction panel taking into account the time for their realization.

Design diagram for determination of deposits produceability in extraction panel is shown on Figure 2.

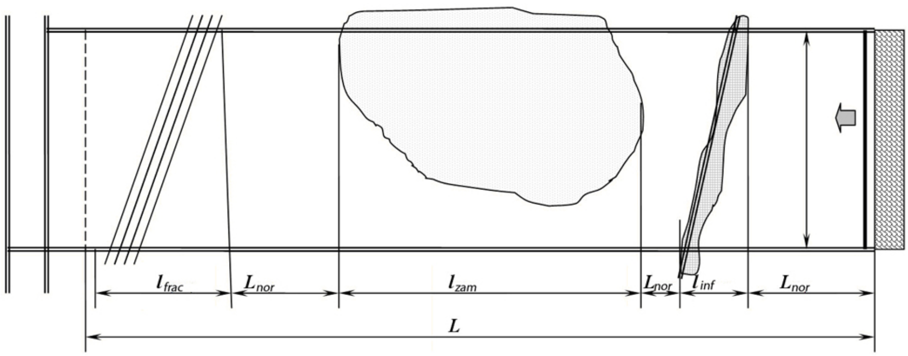

Figure 2. Design diagram for determination of deposits produceability in extraction panel.

Each of the zones is characterized by area in which it is necessary to carry out measures that are differ in technology and in duration time of the influence on the massif. Taking constant longwall length as an evaluation parameter of the influence on technological processes, we can establish length of the zone along direction of the face advance. This is a quite acceptable admission as even small area at a separate section of the longwall leads to general delay in the face advance.

Quantity of additional measures depends on the complexity of the geomechanical characteristics change and on the technical conditions of the mechanized complexes application.

For influence evaluation of the complicating factors on the application effectiveness of mechanized complexes it is suggested to introduce coefficient of deposits produceability (k_{prod}) which shows the volume ratio of the deposits that are deposited under mining-geological conditions out of limits of a complex technical conditions to general quantity of coal reserves in extraction field.

Taking into account consequence of passing of zones with different production readiness of deposits, produceability coefficient can be presented as a product of coefficients of various produceability

$$k_{tech} = \prod_{i}^{n} k_i \, , \tag{1}$$

where k_i – coefficient of technological complexity of i zone, in which it is necessary to implement the measures; n – number of zones in extraction panel in which it is necessary to implement the measures.

Coefficient of technological complexity of i zone expresses ratio of general costs for implementation of measures directed to improvement of geomechanical characteristics of the host rocks to the level of produceability of the deposits, to general costs for the coal mining, which meet technical conditions of the complex based on the length of the extraction panel.

$$k_i = \frac{З_i l_i}{З_{tech} \sum l_{tech}} \, , \tag{2}$$

where $З_i$ – costs per one meter of the longwall advance for implementation of the measures directed to improvement of geomechanical characteristics of the host rocks to the technological conditions of the complex, UAH; l_i – length of the zone in which the measures should be implemented, m; $З_{tech}$ – costs per one meter of longwall advance during coal extraction under technological conditions, UAH; l_{tech} – length of the extraction panel with mining-geological conditions which meet the technological conditions of the complex, m.

Evaluation of the coefficient of deposits produceability of coal in an extraction panel is not enough to provide effectiveness of high performance mechanized complexes application. The coefficient does not express the time for the mining of deposits and the degree of the usage of resource by the mining equipment and its correspondence to the anthropogenic environment of mine workings.

Zonality of mining-geological conditions and physical-mechanical properties of a rock massif necessitates the correction of technological processes and consumption of different volume of the shearer's resources, conveyors, pump station and so on. Thus, time of the deposits extraction in different zones without implementation of measures directed to improvement of deposits to technological conditions will differ from a rated time.

So the effectiveness of usage of the mechanized complex resource is proposed to evaluate also with coefficient of service life usage during extraction of coal in the mine field (panel). With zonal structure of the rock massif the coefficient of service life can be expressed as a ratio of deposits mining time to general time of expenditure of service lives in separate components of the complex (shearer, face conveyor, pump station and so on), it has the following formula

$$k_t = \frac{\sum_{i}^{n} t_{t.d.}}{\sum_{j}^{m} t_m} \, , \tag{3}$$

where $t_{t.d.}$ – deposits mining time in an extraction field of i zone of host rocks structure with n number of zones, hour; t_m – time of service life of j's component of mechanized complex with general quantity m, hour.

The expenditure of the service life of mechanized complex is influenced not only by the length of extraction panel, but the length of a longwall as well, i.e. some area index that defined productivity of a stope in unit time. Therefore, with zonal structure of host rocks these two indexes should be used for objective assessment of effectiveness of high performance complex application. In the aggregate they characterize rightness of making a technological decision and selection of mechanization means in extraction field. Technological measures of improvement of host rocks to the level of corresponding technical conditions of the mechanization means functioning are considered, also their consequence and time of implementation.

On the basis of above stated information, coal deposits in an extraction field are proposed to be considered producible under condition when coefficient of produceability $k_{prod.} > 0.85$ and coefficient of service life usage of mechanized complex $k_m > 0.90$.

Given admissions are accepted based on the data about probable error of initial information concerning mining-geological conditions of mining process.

When defining the coefficient of service life usage of the mechanized complex it is necessary to consider various value of service life for coal shearers, conveyors and mechanized support. Probably to equalize service life of the components, for example, sections of mechanized support, it will be necessary to change a face conveyor several times or a coal shearer in which service life depletes much earlier. Thus, expenses for packaging of the face complex will be increasing to all reserves but it will have discount cost for the left reserves.

Therefore, to determine produceability of an extraction panel reserves it is necessary to develop classification of technological principles of rock massif state management during its structure changing.

It is known, that technological parameters of stoping significantly influence the stress-strain state of the rocks, formation of load on the support elements and manifestation of rock pressure in mine workings. Length of longwall, its rate of advance in unit time define condition of stopes and also character of rock cavings in worked-out area. This instrument of regulation of rock massif condition can be effectively applied at substitution of lithologic differences and their thicknesses change because the influence area embraces a considerable value of host rocks.

Presence of zones of intensive fracturing and low-amplitude faults in an extraction panel is a constricting factor during intensive extraction of coal seam, although their stress state can be regulated by technological parameters under definite circumstances: shearer’s width of cut, density of support installation and its yield.

Zone of rocks crashing which is squeezed between immovable tectonic blocks has sufficient cohesion force in order to not fall with a small exposure area of immediate roof rocks. At the same time a long period of face presence in this zone leads to redistribution of stresses around the stope and can lead to rocks fall and blockage of working area. It is essentially especially for abutment of longwall with stopes where the increased concentration of stresses takes place. Quite different character of rock pressure manifestation takes place with presence of humidity that leads to plastic state of massif in the zone of intensive fracturing. With presence of collector connection with water-bearing levels in considered zone, water comes into mine workings as drops or water jets. Depending on place and amount of water income into mine workings, there can be applied both measures designed for water drainage from a working zone and controlled parameters of stope disposition.

5 CONCLUSIONS

Effectiveness of mining operations development depends on the orientation of development workings with respect to force field of stresses of the Earth at a concrete area of a mine field and on the produceability of reserves for stoping.

With zonal structure of host rocks, it is necessary to introduce produceability coefficient of extraction panel and coefficient of service life usage of face mechanized complex in order to objectively evaluate effectiveness of high performance complex application. In total they characterize rightness of technological decision acceptance and selection of means of mechanization in an extraction field.

Coal reserves in an extraction field are considered to be producible when coefficient of produceability is $k_{prod.} > 0.85$ and the coefficient of service life usage of face mechanized complex is $k_t > 0.90$. At this, technological measures designed for improvement of host rocks to technical conditions of functioning of mechanization means, their consequence and time of implementation are taken into account.

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New Techniques and Technologies in Mining – Bondarenko, Kovalevs'ka & Dychkovs'kyy (eds)

Safe electric power transmission for explosive underground mines

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ABSTRACT: New system of electric power transmission for explosive underground mines is considered. The system includes equipotential current-carrying paths which advanced disabling in the case of emergency state of the line is provided. Processes in the proposed power transmission system caused by open faults, short circuits and current leakage occurrence are described. Obtained relationships show possibility and ways of sparking-proof power transmission enhancement. Investigations made it possible to develop specialized units for monitoring of processes faults occurence. Reserves of the transmitted power rise in the suggested system were revealed and that was confirmed by testing. The obtained results indicate possibility to implement the system in powerful DC and AC networks for increase of their safety in explosive environment.

1 INTRODUCTION

Earlier research and experiments in relation to spark-proof properties of parallel paths of trolley locomotive current collecting devices showed possibility of safe breakage of the shunting power transmission lines feeding load of quite high power (Ivanov 1995). This idea had been taken as a basis of new devices providing sparking suppression at current collection of trolley mine locomotives (Ivanov 2000). The flame- and sparking-proof properties of the device are provided by means of determination of the current in the parallel paths distribution and disabling the power consumers when this distribution becomes uneven. The uneven distribution serves as a criterion of the state preceding one more path breakage. As a result of the investigations, the general current distribution behavior was determined and the devices effectiveness was substantiated.

The above mentioned method provides use of electricity in enhanced explosive conditions may be applied for development of new electric power transmission systems of increased power for such conditions including spark-proof systems of electric power supply.

2 ENERGY LIBERATED AT CONDUCTOR BREAK

As compared to the devices providing sparking suppression at current collection of trolley mine locomotives, in this case, much greater length of the parallel circuits and their mutual coupling should be taken into consideration. This coupling can be of greater importance because of greater length and smaller distance between the current carrying conductors.

Taking into account that breaking of one of the parallel branches does not stipulate changes in the paths, being at different potential, processes arising at the line break may be analyzed using an equivalent circuit for unipotential branches (Figure 1). The capacitor voltages caused by some unbalance of the line actual parameters are negligible. The line longitudinal per unit capacity is rather small. Ignoring the indicated parameters the equivalent circuit becomes as it is shown in Figure 2. Maximum energy liberated in electric discharge that develops in the place of the branch breakage is assessed by maximum voltage across the capacitor C after the switch of S break. Providing that the load current I does not vary at one of the branches break, i.e. $I = const$. This is explained by the fact that breakage of a parallel path in the considered circuit in the presence of another operating path, the line resistance variation is very small and does not affect the load current.

A set of equations describing processes in the line:

$$i_1 + i_2 = I$$

$$R_{i_1} + L\frac{d_{i_1}}{dt} + M\frac{d_{i_2}}{dt} = R_{i_2} + + L\frac{d_{i_2}}{dt} + M\frac{d_{i_1}}{dt} + u_c , \qquad (1)$$

where L and R – inductance and resistance of one branch, M – mutual inductance of current carrying line conductors.

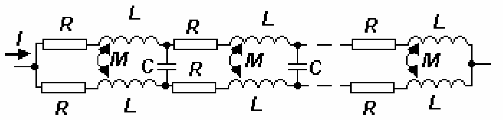

Figure 1. Equivalent circuit for unipotential parallel branches.

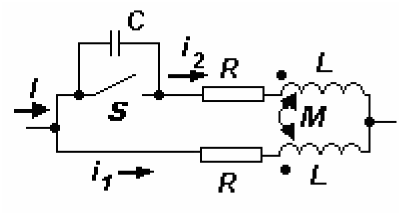

Figure 2. Equivalent circuit for assessment of energy liberated in the discharge.

With account of initial conditions $i_1(0)=i_2(0)=I/2$, the system solution for i_2 and u_c is

$$i_2 = I/2\, exp\{-R \cdot t/[2(L-M)]\}\{cos\, \varpi t + \\ + R/[2\varpi(L-M)]\}sin\, \varpi t\} \tag{2}$$

$$u_c = IR\{1-exp(-\delta t)\times \\ \times[cos\, \varpi t + sin\, \varpi t(\delta - 1/2RC)/\varpi]\}, \tag{3}$$

where

$$\delta = \frac{R}{2(L-M)},$$

$$\varpi = \sqrt{\frac{1}{2C(L-M)} - \frac{R^2}{4(L-M)}}\,.$$

Ignoring the damping, we find that the maximum voltage across the capacitor is

$$U_{c_{max}} = I\sqrt{\frac{L-M}{2C}}\,. \tag{4}$$

At that the energy stored in the capacitor is equal to the maximum energy that can be released in the electric discharge comes to

$$A = \frac{CU_{c_{max}}^2}{2} = \frac{L-M}{4}I^2\,. \tag{5}$$

The energy dissipating in the discharge can be considerably less than the determined capacitance maximum energy, as far as zero, if the protection comes into action previously the full break of one of the conductors. This is explained by the fact that the full break is preceded with redistribution of the current between the parallel branches.

It follows from equation (5) that the more coefficient of mutual inductance of the line parallel conductors the less is the released energy that dissipates in the discharge at breaking of one of the line conductors.

Therefore, to reduce the released energy it is necessary to use cables that have greater mutual-inductance coupling of the conductors. The effect can be promoted with shielding of the unipotential parallel conductor groups.

3 CLOSING THE CONDUCTORS

Analyzing processes at closing in the circuits of the transmission line with parallel branches the cases of closing the conductors of unipotential paths, closing the conductors that have different potentials and the line output terminals should be considered.

With respect to danger of explosive sparking, closing of the unipotential conductors is safe because of very small voltage between the points to be connected. This voltage can appear only due to insignificant unbalance of these conductors. But, following these conductors connection, the line break in any point between the point of closing and the load does not bring to variation of current distribution in the circuit part between the power source and this point and can not be detected from the initial point of the line. On this reason, the line should be disabled when closing of unipotential conductors in any point occurs.

Such possibility will be provided if closing the parallel conductors at any point causes their current distribution variation.

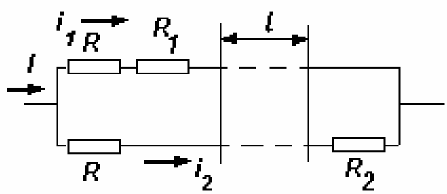

Figure 3. Circuit providing current redistribution under conductors connection.

The transmission line circuit meeting the requirements is shown in Figure 3. Its particular feature is availability of identical resistors R_1 and R_2, being of small value, additionally introduced into the branches, the first at the beginning and the second at the end of the line. In this case, connection of the parallel conductors in any intermediate point along the section l causes the currents redistribution that can be revealed by comparison the voltage drops in the resistors R.

The described circuit has high sensitivity relatively to approaching between unipotential conductors but is ineffective at short circuits between the line conductors of opposite polarity and in the case of small but unsafe leakage currents in the line.

4 LEAKAGE AND SHORT CIRCUIT DETECTION

The leakage currents and short circuits occurring in the line can be detected if resistors R, which in particular may be of equal magnitude, are inserted in each line branch at its ends.

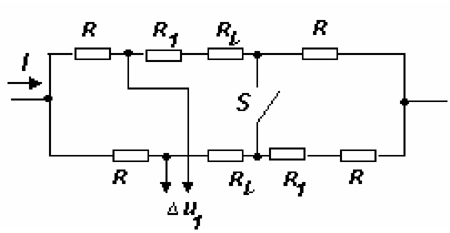

Figure 4. Equivalent circuit for sensitivity assessment.

For assessment of sensitivity relative the short circuit, use the voltage increment Δu_1. Consider the worst conditions when the short circuit is located at the far end of the line (Figure 4). Assume that before the short circuit the branch currents are equal, then

$$\Delta u_1 = I \frac{RR_1}{2R + 2R_l + R_1}, \tag{6}$$

where R_l – resistance of a branch.

It follows from the equation (6) that the circuit sensitivity depends on the load current. It should be taken into account when devices for sparking protection are developed. The parameters must be selected so that reliable power source would switch off at the short circuit occurrence at minimum load current.

Assessment the circuit sensitivity for leakage currents in the same conditions (Figure 5).

It is seen from equation (7) that magnitude of the signal is directly proportional to the leakage current between the line conductors having different electric potentials. The sparking protection sensitivity must guarantee its reliable operation when the leakage current reaches unsafe value.

$$\Delta u_2 = i_{lk} \frac{R^2}{2R + R_1 + R_l} \ . \tag{7}$$

Protective cutting off for the case of short circuit at load end can be provided by common methods. The described method of electric power transmission makes possible switching off a power source at the very early stage of sparking developing and mainly before the line conductor break. This permits to increase maximum permissible current of a line.

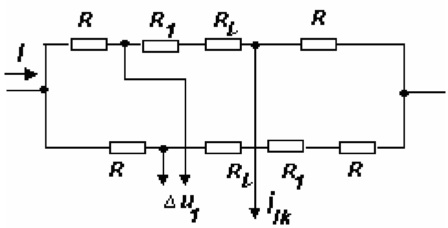

Figure 5. Leakage sensitivity assessment.

Sufficiently high sensitivity to leakage currents at use of fast acting switching device permits to increase the line voltage. Devices of sparking suppression based on the described method provide considerable increase of the spark-safe power transmitted to a load through electric line.

Preliminary test results confirm applicability of the described method for safety improvement of powerful DC and AC lines employed in explosive dangerous conditions.

5 CONCLUSIONS

New method of power transmission safety improvement for explosive dangerous environment, based on advanced line disabling and measurement current distribution in parallel current carrying paths can find application in explosive underground mines for feeding of automation, telemetry, communication means as well as for power lines supplying electric consumers of technological equipment.

Analysis of processes caused by a conductors break in the transmission line with parallel current carrying paths indicates to possibility of considerable increase of its spark-safe transmitted power.

Use of the transmission line having parallel conducing paths provides better conditions for safety improvement at occurrence short circuits and unsafe leakage in the line.

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Analysis of hydrogeodynamics in a mining region during exploitation till closure of coal mines

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ABSTRACT: The paper presents the results of studies concerning changes of hydrogeodynamics in the region of Central Donbass (Ukraine). The active mines are expected to be closed within the next twenty years, which requires predictive estimations for hydrogeodynamic trends in mined out strata. The studies were performed by numerical modeling of regional ground water flow. The model allowed reproducing paleo-hydrogeological conditions during pre-mining time, which is highly important to assess the impacts of the exploitation period. The predictive parametrical estimations were analyzed for different scenarios of mine closure. The proposed technical schemes include local draining and grouping of mine drainage; they aim eventually at self-draining of the post-mining region.

1 INTRODUCTION

The Donetsk basin (Donbas) is the main coal region in Ukraine having more than 200 years history of mining, with the number of active mines exceeding one thousand. The Central region as one of the Donbas key areas belongs to heavily built and highly developed mining industry agglomerations, with total population approaching to one million people. Such industrial cities as Gorlivka, Yenakievo, Dzerzhins'k and Vuglegors'k are located here. Twenty eight coal mines have been operated in Central Donbas till the year 2000; they exploited from 19 to 26 coal seams.

Long-time mining has changed hydrogeological conditions in Central Donbas on the area larger than 410 square kilometers due to increase in rock permeability, total exhaustion of water resources and further complication of underground runoff into streams. The situation was aggravated by restructuring of the coal industry, which disturbed the equilibrium in the system "ground waters-bedrocks-day surface". Closure of coal mines has led to ground water rise and flooding large areas, this initiated dangerous subsidence and firedamp emission.

These phenomena were studied by systematization of monitoring data, estimation of hydrodynamic impacts on bedrocks, evaluation of geotechnical conditions in mining affected zones, mathematical modeling of ground and mine water flow, calibration of key parameters for modeling of hydrodynamics in mined out rocks, comparative analysis of engineering solutions and technical measures being developed to regulate ground water head on mine lands.

The hydrographic network in Central Donbas consists of small rivers Kryvyi Torets, Balmutka, Rosokhovata, Zalizna, Krynka and their inflows Bulavin, Korsun', and Sadki. The absolute levels of water in rivers varies from 85 to 252 m, absolute elevation of watersheds reaches 310 m. The river network density ranges from 0.2 to 0.5 km / km^2. Meandering river beds have significant slope; their width narrows to 5-10 m during the lowest water level in summer. The average value of underground runoff for the coalmining area in Donbas is estimated at 0.8 $l / (km^2\ s)$. Ground and surface waters are well connected hydraulically.

The aquifers in Central Donbas are located in modern Quaternary, Permian, and Carbon sediments (Hydrogeology and engineering geology of deep horizons in Donbas 1974, Hydrogeology of the USSR 1971). The aquifer in modern sediments extends along rivers as a narrow strip or localized on watersheds. The aquifer in Carbon sediments often reaching the day surface is spread throughout. Permeable sandy and clayey soils in Carbon sediments form the complex of confined aquifers. Its top part lies as a rule in highly fractured rocks of 40-100 m thickness. Ground waters here are replenished by infiltration and discharge from the dense river and gully network in Donetsk Ridge.

One of the main factors influencing environmental conditions around the mines being closed is the ground water head distribution over area and in depth. Ground water rise considered as flooding complicates industrial and agricultural activities on mine lands and leads to undesirable hydrogeological transformations (Norvatov 1988, Zilberg 1983). The available standards define flooding as a complex process caused by man-made and natural impacts and accompanied by changing of the water balance up to critical values, which requires taking protective engineering measures.

2 METHOD OF MODELING

Prediction of complicated ground water conditions on mining affected areas was performed by mathematical modeling that is used now as the advanced tool for development of engineering solutions (Bachmat et al. 1980, Gavich 1980). Its first stage consists in taking assumptions and acceptable hydrogeological simplifications as well as determining parameters and site characteristics, which allows formulating the proper flow scheme. The next stage results in the calculation scheme and joining its modules followed by identification of the model. Its sensitivity to variations of flow parameters and boundary conditions is estimated by solving a series of inverse problems, which enables calibrating and validating the model. The third stage of modeling is aimed at making hydrodynamic predictions, calculating additional characteristics, improving the flow scheme, numerical analyzing proposed engineering solutions to quantify their impacts.

The software MIF developed at the Laboratory for hydrogeological studies of the Ukrainian State Geological Prospecting Institute (Dnipropetrovsk) and adopted for mining hydrogeology conditions was used to create and identify the flow model of Central Donbas. The program is applicable to numerical solution of differential equations governing 2D and 3D flow and transport in heterogeneous layered rocks. The model accounts for ground and surface water interaction, seepage through confining layers, precipitation, infiltration and artificial recharge, evaporation depending on ground water depth, discharge on the day surface, mine water drainage by stationary devices, time-dependent boundary conditions and flow parameters, transition between free and confined flow mode, hydraulic connection among aquifers, nonlinearity of flow in free aquifers, and rock anisotropy.

The unsteady flow in hydraulically connected permeable layers is governed by the known system of differential equations for free and confined aquifers (Bear et al. 1968, Hydrological Forecasting 1985, Zhernov & Pavlovets 1976). The number of equations is determined by the aquifer number, head values and layer elevations.

Depending on hydrodynamic conditions three types of boundaries are simulated. They are described by time-dependent head, discharge, and the relation between head and discharge:

Seepage through low-permeable confining layers is simulated according to Miatiev-Girinski’s assumption about neglecting horizontal flow in aquitards and vertical flow in aquifers (Shestakov 1979). The flow domain is divided by volume elements of horizontal size Δx and Δy formed as a result of crossing vertical orthogonal planes. The saturated rock properties are averaged for each cell and relate to its center. The computational scheme is based on the inflow-outflow balance between adjacent grid blocks.

The spatial derivatives of the flow equation are replaced with the differences along *Ox* and *Oy* axes (Lomakin et al. 1988, Shestakov & Luckner 1976)

$$F\frac{\partial}{\partial x}\left(T\frac{\partial H}{\partial x}\right)_{ij}=\frac{H_{i,j-1}-H_{ij}}{\frac{1}{\Delta y_i}\left(\frac{\Delta x_{j-1}}{2T_{i,j-1}}+\frac{\Delta x_j}{2T_{ij}}\right)}+ \\ +\frac{H_{i,j+1}-H_{ij}}{\frac{1}{\Delta y_i}\left(\frac{\Delta x_{j+1}}{2T_{i,j+1}}+\frac{\Delta x_j}{2T_{ij}}\right)}, \tag{1}$$

$$F\frac{\partial}{\partial y}\left(T\frac{\partial H}{\partial y}\right)_{ij}=\frac{H_{i-1,j}-H_{ij}}{\frac{1}{\Delta x_j}\left(\frac{\Delta y_{i-1}}{2T_{i-1,j}}+\frac{\Delta y_i}{2T_{ij}}\right)}+ \\ +\frac{H_{i+1,j}-H_{ij}}{\frac{1}{\Delta x_j}\left(\frac{\Delta y_{i+1}}{2T_{i+1,j}}+\frac{\Delta y_i}{2T_{ij}}\right)}, \tag{2}$$

where $F = \Delta x_j \Delta y_i$, F – horizontal area of a cell containing the node (i, j), $H_{i\pm1,j\pm1}$ – average levels in neighbor cells; $T_{i\pm1,j\pm1}$ – average transmissivities in neighbor cells.

Difference approximation of the term describing flow between adjacent aquifers is written as

$$Q_{aq,ij}=\frac{k_{0,ij}\Delta x_j \Delta y_i}{m_{0,ij}}\left(H_{1,ij}-H_{2,ij}\right), \tag{3}$$

where $k_{0,ij}$ – conductivity, $m_{0,ij}$ – confining layer thickness, $H_{1,ij}$ and $H_{2,ij}$ – heads in two aquifers.

The recharge *w* distributed over the block area is formally concentrated in the node (i, j) so that

$$Q_{r,ij} = w(x_j, y_i)\Delta x_j \Delta y_i \,. \tag{4}$$

The boundary condition governing ground and surface water contact is written as follows

$$Q_{gsw,ij} = \chi_{ij}\left(H_{s,ij} - H_{ij}\right), \tag{5}$$

where χ_j – leakage coefficient quantifying flow resistance of boundaries between ground and surface waters, and openings,

$$\chi_{ij} = \frac{T_{ij} N_{ij}}{L_{ij} + \Delta L_{ij}} \,, \tag{6}$$

where $H_{s,ij}$ – average water head in the stream, L_{ij} – distance between the block center to the stream, ΔL_{ij} – the parameter of bed penetration into the aquifer, N_{ij} – stream length in the block.

Mine drainage is governed by the relation

$$Q_{md,ij} = T_{w,ij}\Delta x_j \Delta y_i\left(H_{w,ij} - H_{ij}\right), \tag{7}$$

where $H_{w,ij}$ – water head hold owing to mine drainage, $T_{w,ij}$ – additional transmissivity accounting for hydraulic connection in cells containing mine drainage.

The condition given along the pond contours of small hydraulic resistance can be written as (Livshits & Belokopytova 1987)

$$Q_{p,ij} = T_{p,ij}\left(H_{ij} - H_{p,ij}\right)\Delta x_j \Delta y_i \,, \tag{8}$$

where $T_{p,ij}$ – pond bottom transmissivity, $H_{p,ij}$ – water level in the pond.

The time derivative in the flow equation is replaced with the difference relation corresponding to the implicit calculation scheme.

The number of layers in the flow model is determined by practical reasons. The more complicated is the hydrogeological structure, the more layers should be introduced to describe it. Greater inclination of strata and tectonic breaks as well as the increasing range of conductivity in depth requires more layers for calculations. After mining due to rock crumbling may arise a larger aquifer hydraulically united through underground workings. In such case the number of layers can be reduced, especially for closing thin coal seams. The inflow values are of most importance for description of remote permeable strata. Flow parameters in each layer are schematized with regard to horizontal and vertical heterogeneity; their average values depend on elevation, aquitard permeability etc. (Livshits & Belokopytova 1987, Rudakov & Sadovenko 2006).

The vertical components of conductivity and transmissivity between blocks of adjacent layers is calculated according to formulas for piecewise-homogeneous medium

$$K_{z,l+1/2} = \frac{\Delta z_l + \Delta z_{l+1}}{\dfrac{\Delta z_l}{K_{z,l}} + \dfrac{\Delta z_{l+1}}{K_{z,l+1}}} \,, \tag{9}$$

$$T_{z,l+1/2} = \frac{\Delta z_l + \Delta z_{l+1}}{\dfrac{\Delta z_l}{K_{z,l}} + \dfrac{\Delta z_{l+1}}{K_{z,l+1}}} \frac{1}{0{,}5\left(\Delta z_l + \Delta z_{l+1}\right)} \,, \tag{10}$$

where $K_{z,l}$ and $K_{z,l+1}$ – conductivities of layers indexed with "l" and "l+1".

The model is identified by comparison with well testing data that are sensitive to the interval of the borehole used for measurements. The narrower this interval becomes, the more exact head values can be obtained. The averaged value of ground water head in the measuring working interval within the block (i, j) on the layer "b" is estimated as (Bilokopytova et al. 2003)

$$h_w = \frac{\sum\limits_{b=m}^{n} \dfrac{T_{b,ij} \cdot h_{b,ij}}{ln\left(r_{ab}/r_w\right)}}{\sum\limits_{b=m}^{n} \dfrac{T_{b,ij}}{ln\left(r_{ab}/r_w\right)}} \,, \tag{11}$$

where $T_{b,ij}$ – transmissivity, $h_{b,ij}$ – ground water head, r_w – well radius, r_{ab} – radial distance from the block center to the circumference where head is assumed to be equal $h_{b,ij}$, r_{ab} equals to the radius of sinking $S = h_{b,ij} - h_w$ in the observation well.

3 MODEL CREATION

The calculation scheme was compiled after analyzing mining area conditions and interpreting the test problem solutions aimed at evaluation of the optimal number of layers. The aquifer in fractured rocks was reproduced by two layers on watersheds and one layer on elevations lower than 170 m. Ground water flow in deeper strata was simulated using four layers, which enabled proper reproducing vertical and horizontal heterogeneity, mine water inflow to drains on different depth. Owing to the model regional scale the flow in the locally spread Quaternary aquifer was simulated together with the flow in Carbon sediments. Thicknesses of the first and second layer reach 130 and 90 m respectively. The relatively permeable third and fourth layers of thickness 200 and 300 m contain watered tectonic breaks; the fifth and sixth layers are located in the deeper zone of slow water exchange.

The flow area was contoured regarding to the boundaries of hydrogeological basins and watersheds in order to maximally diminish the effects of outside inflows on the total water balance.

The following data were used for model creation: surface elevation, hydrogeological sections, vertical profiles of conductivity along deeply penetrated wells and shafts, absolute heads of ground water in zones of high fracturing and mining, water levels in streams and ponds, mining work schedules, inflows to different mining horizons etc. The area covered with the flow model of Central Donbas occupies 1151 km^2, grid step equals 0,5 km, the number of grid blocks amounts 27139 (Figure 1).

The weighted average conductivities in blocks along the *Ox* and *Oy* axes differ due to varying velocity directions and rock anisotropy. The layer initial conductivities along the *Ox* axis estimated by aquifer testing were corrected while inverse modeling (Figure 2). The calculated anisotropy relation k_y/k_x ranging from 0.05 to 2.0 accounts for difference in longitudinal and transversal conductivities caused by rock heterogeneity and tectonic breaks.

River and ponds were simulated as third-type flow boundaries, with using absolute water stage, bottom elevations, and conductance of bottom sediments. The parameter of interaction between surface and ground waters below rivers and ponds being wider than the block size is calculated as

$$DL = \frac{k_0}{m_0} F_b ,$$

where k_0 – river mud conductivity (on the average $k_0 = 2{\cdot}10^{-4}$ m / day), m_0 – river mud thickness (usually m_0 – 0.25 m), F_b – water body area in the block. The parameter *DL* for Volyntsevs'ke and Gorlivs'ke water accumulating ponds reaches 200 m^2 / day.

Piecewise distribution of the infiltration rate is formed by dividing the area into zones with regard to geomorphology and ground water depth. Before mining the natural recharge was ranging from 18 to 25 mm / year. The total infiltration rate during mining includes also leakage from accumulating ponds, which was estimated by inverse modeling.

To describe mining horizons in the numerical model the schemes of their location on the area of neighboring mines were compiled (see an example on Figure 3).

Inner boundaries such as surface of underground workings were simulated with account for additional flow resistance on openings, and the difference between the mining area and the block size. The flow resistance parameter was estimated by inverse modeling with regard to ground water head elevation over mining horizons and inflow into workings.

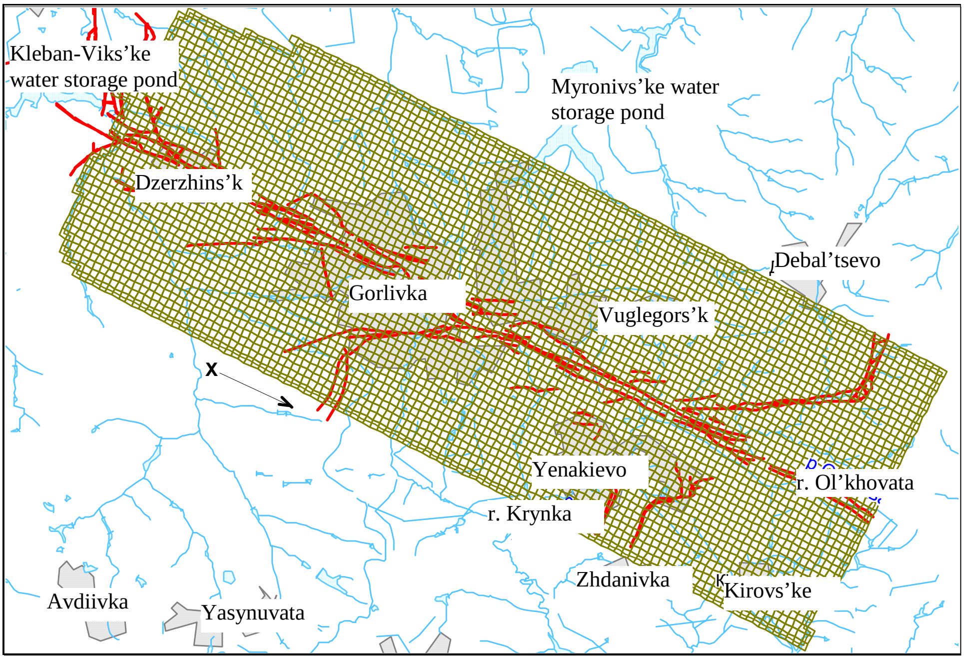

Figure 1. Calculation grid of the flow domain.

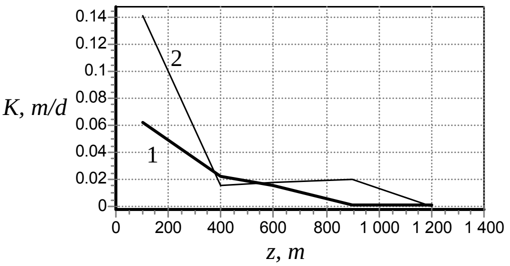

Figure 2. Change of weighted averaged conductivity *K* in depth *z*: 1 – modeling, 2 – aquifer testing.

The data for water intake schedule were input from the State ground water inventory related to the Donetsk region in years 1981, 1991, 1992, 2000. The simulated area does not contain wells of essential intake except single ones; the systematic data about pumping rates miss. The most of wells were not equipped with measuring units so water discharge was evaluated by pump rates and their working time. Eventually, water intake was simulated as the distributed negative parameter to be added to infiltration. The absolute values of intake within Central Donbas was ranging from 2.46 to 3.7 mm / year.

Geological structure of Central Donbass changed by mining has led to the complicated ground water head distribution in depth that had to be taken into account at model identification. This required a special approach to compare simulated and observed heads and inflows correctly.

All wells penetrating several layers were distributed among grid blocks; the data about ground water head were systemized regarding to locations of layers and intervals of their crossing by boreholes. If a borehole penetrates several layers, the head averaged over these layers should be used to compare the modeling results with observations.

4 MODELING RESULTS

The model was identified through successive solving of inverse problems. Firstly steady ground water flow was calculated for pre-mining time; then unsteady flow was simulated from 1980 till 2004 with regard to changes in hydrogeological trends caused by flooding of mines having started since 1996.

Special attention was paid to the factors affecting ground water head. The decisive influence on its distribution under natural conditions was shown to make increased conductivity and infiltration. The obtained conclusions about parameters and boundary conditions changing ground water head were used for model calibration starting from the steady flow problem.

The layer conductivities along the *Ox* axis estimated by modeling were proven to be close to the values obtained by aquifer testing. Calculated ground water head was ranging from 84 to 300 m before mining. Flow on watersheds was found to be directed downwards; on the contrary, ground water in confined aquifers flows from valleys upwards. The differences in heads reach ±10 m. Ground water depth in the zone of intensive fracturing does not exceed 5 m on the major part of the simulated area, locally reaching 20-40 m (Figure 4).

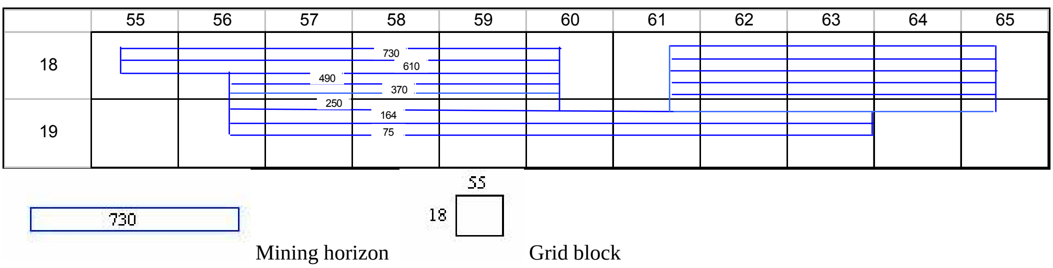

Figure 3. The mine works schedule on the mine named after Rumiantsev over the seam "Yulievs'ky" according to gridding.

Before mining the ground water balance was formed mostly by infiltration (88.3% of inflow), discharge to surface water bodies (50.4% of outflow), and evaporation (39.8% of outflow). The computational analysis of all inflow and outflow components has confirmed their complete equivalence in the ground water balance.

Depending on the mine work schedule and availability of the monitoring data the following stages of ground water flow during mining were simulated: (1) from 1980 till 1990, (2) from 1990 till 1993, and (3) from 1993 till 1996.

The calculated distribution of ground water head at 1980 was given as the initial conditions for the next period; the same flow parameters were used. Deepening mines was simulated by changing absolute elevation of mining horizons and varying resistance to water inflow to workings. The model reproduced changes of the day surface under mining impacts. Calculations related to 1980-1990 were compared with the available data about total mine outflow. The difference between simulated and observed inflows did not exceed 1.05%. According to the calculated balance in 1990 (Figure 5) mine inflow was provided mostly by infiltration, stored in rocks and surface waters.

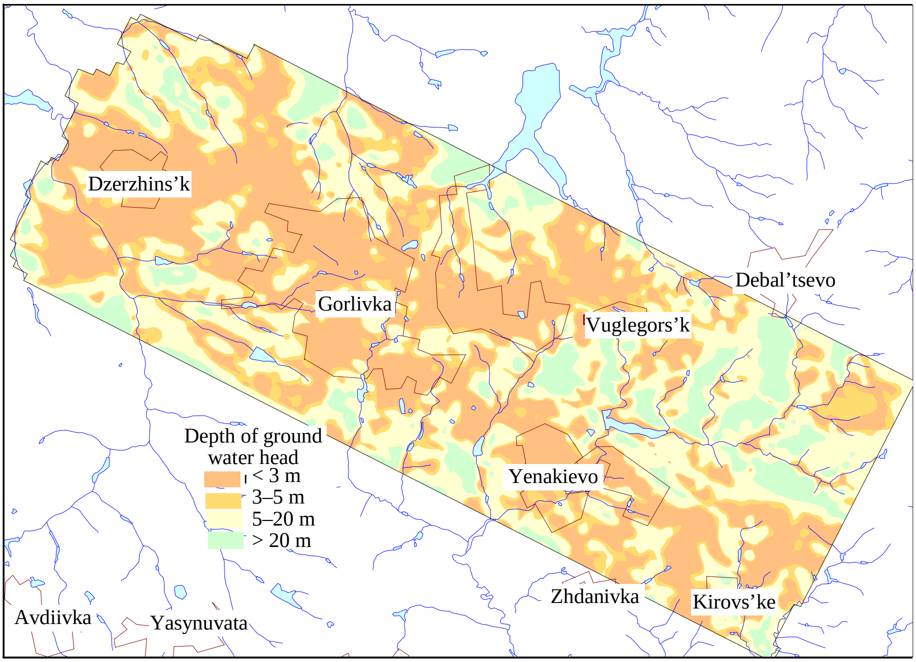

Figure 4. Depth of ground water head before mining calculated by modeling.

Flow modeling under further deepening of mines during 1990-1993 was verified by head elevations measured in 1993. The conductance parameter quantifying the boundary condition on workings was estimated within the range from 0.01 to 1.2 m^2 / day.

Figure 6 demonstrates modeling correctness on the example of two layers. The water balance during 1993-1996 remained almost stable, only the storage share decreased slightly.

During simultaneous flooding and exploitation of mines in 1996-2004 underground hydrodynamics was formed by flows in artificial voids like workings and ruptured zones with increased fracturing and natural voids like pores and fractures in undisturbed rocks.

Artificial voids are considered as a hydraulically united system that determines boundary conditions for saturated rocks during flooding. For this time underground workings and shafts were simulated as connected voids governed by the parameter of gravitational storage capacity in mined out rocks. Conductivity of affected rocks was given several times higher than undisturbed rock conductivity.

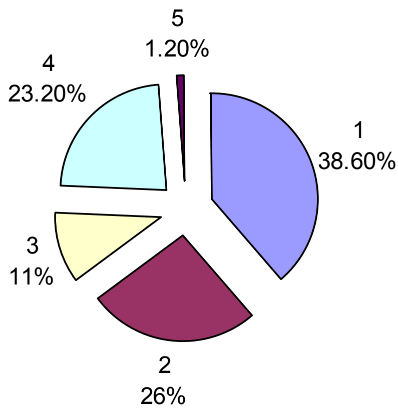

Figure 5. Calculated balance components of water inflow to mine in 1990: 1 – recharge, 2 – storage, 3 – shortening of ground water runoff, 4 – surface water inflow, 5 – shortening of infiltration.

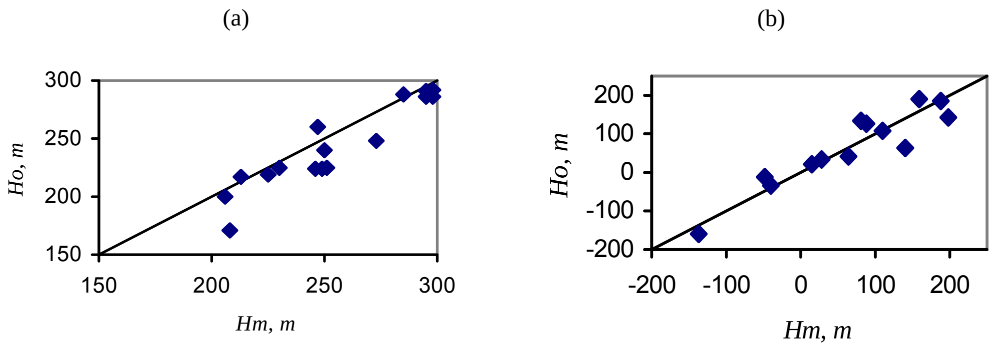

Figure 6. Comparison of modeled (*Hm*) and observed (*Ho*) ground water heads: (a) second layer, (b) fourth layer.

Inverse modeling of mine flooding was performed in two stages: (1) from 1994 till 2000, and (2) from 2000 till 2004. Calculations were compared with measured inflows to mines and mining horizons as well as to water heads in observation wells and flooded shafts in 2000 and 2003. Additional inflows through connecting workings from flooded mines to those being exploited were also taken into account (Sadovenko & Rudakov 2005). Figure 7 demonstrates accepted agreement between observed and simulated heads in a mine during its flooding.

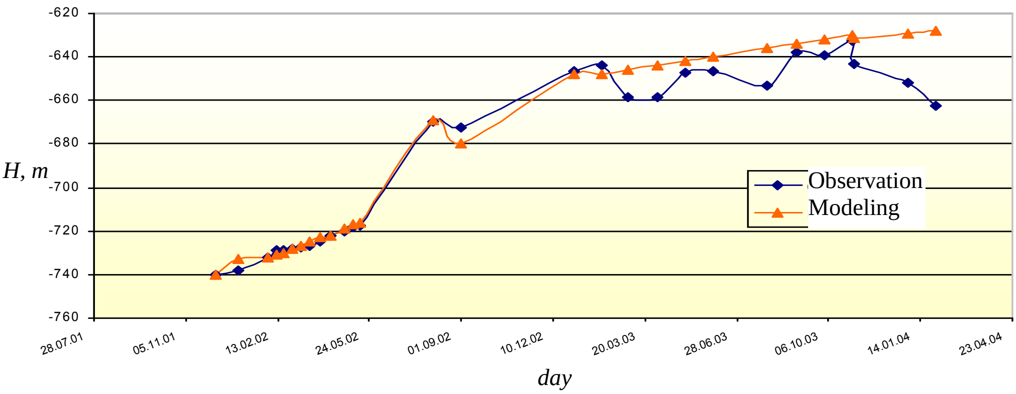

Figure 7. Rise of ground water head *H* in coaly rocks during flooding of the mine "Chervony Profintern".

The verified model allows building predictive hydrogeological map (Figure 8) showing ground water head distribution after flooding of all mines in accordance with the temporal schedule.

The identified predictive flow model of Central Donbas enabled estimating the efficiency of mine areas protection against flooding by grouping drainage on major mines, which was quantified by the relation of the drained area to the drainage rate. This approach was proven to be effective for two mines only. It can be explained by long-time exploitation having formed the man-made water balance with dominating surface layer components. The developed model allows optimizing different engineering solutions suited for these conditions including horizontal and radial drains as well as deepening natural river beds.

5 CONCLUSIONS

The regional flow model of Central Donbas identified on the scale of 1:50000 enabled adequate reproducing of hydrogeological conditions for pre-mining time, analyzing regional trends in ground water head re-distribution on area and along key profiles during exploitation and flooding of mines.

The changes of principal factors influencing ground water head in mined out rocks were estimated. Particularly, rock conductivity increased 10-

100 times, gravitational storage grew 7-14 times, infiltration became 2.8-3.5 times more comparing to the pre-mining period.

The generalized parameter quantifying ground water inflow to workings takes into proper account their additional flow resistance. Depending on the mining stage and rock heterogeneity it is ranging from 0.01 to 1.2 m^2 / day.

Simultaneous flooding of all mines in Central Donbas from 2005 till 2030 was shown to result in significant ground water rise. The area of ground water depth less than 5 m is expected to enlarge from 264.1 to 647.2 square kilometers and reach 56% of the total region area. This requires differentiating mine closure in time and taking proper preventive measures.

Zoning of mine areas by hydrogeological and geotechnical conditions and determining the mine flow intensity enabled contouring the areas to be protected effectively with drainage. Different schemes to mitigate mine closure impacts were developed. Deepening river beds coupled with horizontal drainage in valleys and radial drainage on built-up territories will further to transformation of Central Donbas into the sustainable geotechnical system.

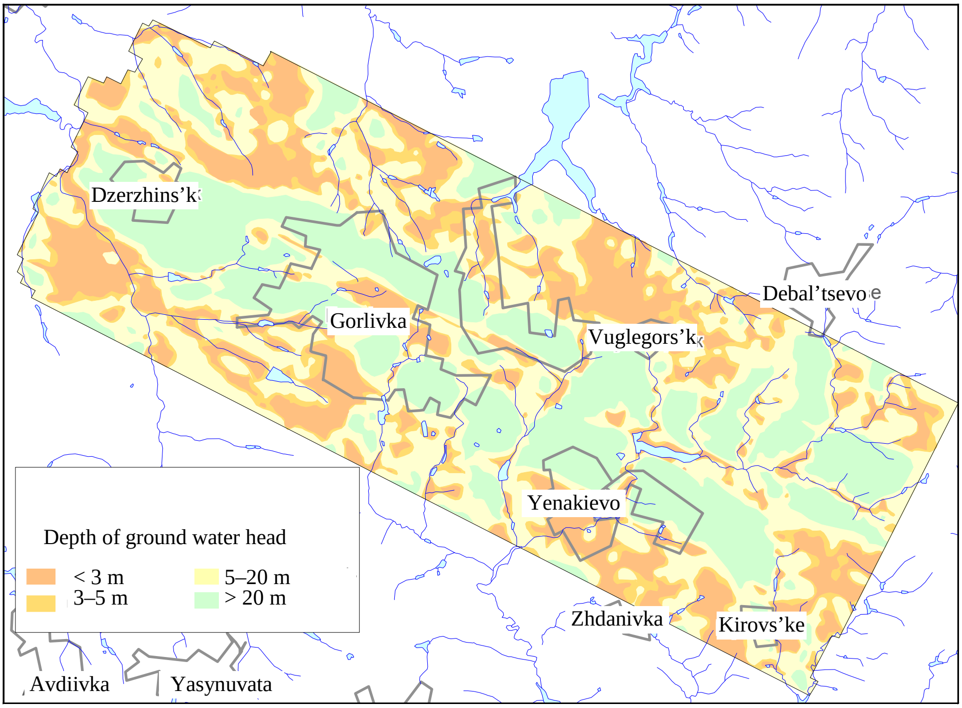

Figure 8. Predicted ground water depth in Central Donbas after mine closure by 2030.

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New Techniques and Technologies in Mining – Bondarenko, Kovalevs'ka & Dychkovs'kyy (eds)

Bifurcational model of rock bottom heaving in mine workings

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ABSTRACT: The process of rock floor heaving in workings of coal mines in Ukraine is researched. Analysis of "in-situ" measurements and numerical simulation is made. The behavior of a geomechanical system "working-rock mass" is established. The fact is proven that the process of a sill is linked to the electoplastic stability loss of rock mass around working. The bifurcational model of rock pressure phenomenon is offered. Distribution of equivalent stress around workings before and after the moment of bifurcation point appearance is obtained

1 INTRODUCTION

On the coal mines of Donbass rock floor heaving is one of the most widespread sign of mining pressure in works. Expenses on liquidation of rock floor heaving are very large. From data (Koshelev, Ignatovich & Poltavec 1991), an extent of the deformed working from the rock floor heaving is 45% from a general extent.

For adequate description and prognosis of conduct of rock mass around of working on deep horizons of mines, it is necessary to perfect theoretical models taking into account more heavy terms.

A complex index while estimating state of working are deformations of the rock contour.

All rock pressure phenomena can be divided into two groups on magnitude of working contour deformation and velocity of its increase. 1st group – geomechanical phenomena which show in a working as displacement of a rocky contour with the maximum magnitude up to 20-30 cm and which are implementation of plastic deformation. These phenomena can be adequately circumscribed on the basis of strict analytical solutions engaging continuous medium models. 2nd group – phenomena with bigger plastic deformations. Depending on physical and mechanical rock properties, number of geomechanical and technical factors, the large moving can be realized both by instant dynamic processes (mining shots, sudden raising of soil) and during long period of time (rock floor heaving) as a static processes. Mathematical descriptions of both one and another phenomena from positions of the generally accepted elastoplastic or elasto-viscidly-plastic models of environment difficultly and for their research of the special approaches and models are required.

2 "IN-SITU" RESEARCHES OF CONFORMITIES TO THE LAW OF ROCK MASS DEFORMATION IN THE VICINITY OF A WORKING

Shashenko (1988) offered new approach to the study of rock floor heaving problem in workings. His approach based on a hypothesis about the elastoplastic stability loss of rock mass in the vicinity of the single working. It is a summary of many results of "in-situ" researches of working contour displacement. These researches allowed to select the presence of four areas around of working. Every area is characterized by different intensity of mechanical processes flowing:

I – area of elastic deformations;
II – area of plastic deformations;
III – area of rock floor heaving;
IV – area of stabilizing (2) or unstabilizing (1) of rock floor heaving process.

On a Figure 1 the lines of displacements and speeds of working contour displacements are resulted for the mine "Belozerskaya" of "Dobropol'eugol". In workings of this mine ground rock floor heaving is the characteristic display of rock pressure.

The area of rock floor heaving (III) is distinctly visible on the line of displacements speeds on Figure 1b. It is characterized by the rapid flowing of process and sharp increase of displacements after achievementt critical limit.

A fourth area completes the process of active deformation of rock mass near contour. Intensity of deformations falls sharply. They either stabilize (line 2) or slowly proceed in time (line 1).

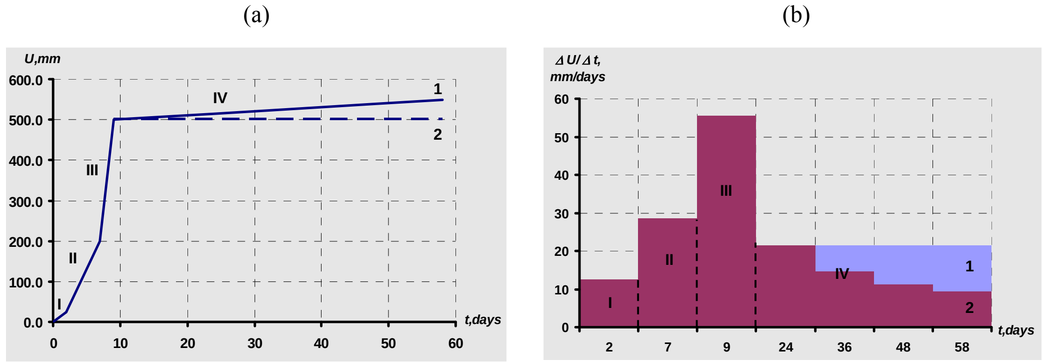


Figure 1. Displacements (a) and speeds of displacements (b) of contour with in meters of "Belozerskaya" of "Dobropol'eugol" (1 – undamping process and 2 – damping process).

3 LABORATORY RESEARCHES OF CONFORMITIES TO THE LAW OF ROCK MASS DEFORMATION AROUND WORKING

Researches were executed on the special flat stand for workings with the arched form of section, located in a homogeneous rock mass.

As equivalent material mixture of sand, technical vaseline, paraffin and graphite was used. Compositions of mixtures correspondent to the properties of rock mass in "Dobropol'eugol'".

On the model of rock mass, containing working, weight was put with an interval in 1 kg. Rising deformations of mass registered on every stage by camera. By shots the high-quality picture of mass behaviour was built. The process of model deformation was studied by the special markers which were inflicted on the side surface of the model.

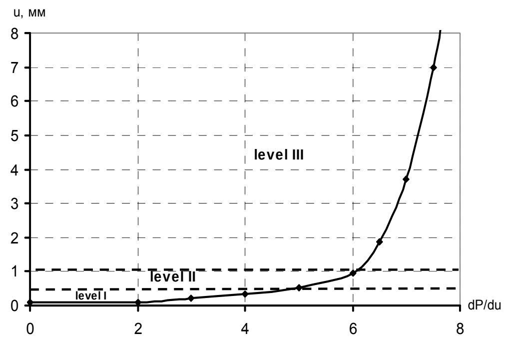


Figure 2. Dependence of floor heaving on the intensity of an exterior loading.

It was set that the mechanism of geomechanical processes corresponds the physical model of elastoplastic stability loss. It is evidently characterized by dependences from Figure 2. We can highlight three levels of deformations there: level I – resilient deformation of model; level II – formation of unresilient deformations zone around working; level III – a loss of elastoplastic stability of rock mass – ground rock floor heaving. The third level of deformations is characterized by that the considerable moving of floor in working u corresponds a small increment of the external loading dP / du.

Such mechanism of rock floor heaving comports with the vectorial picture of markers displacements on the model (Figure 3). Results discovered that during rock heaving basic deformations take place in floor of working and affect deep layers.

Executed "in-situ" and laboratory researches, well show us the picture of rock floor heaving development. It also shows that a general physical model is the basis of this phenomenon.

4 DETERMINATION OF AN ANALYTICAL TASK ABOUT THE ELASTOPLASTIC STABILITY LOSS OF ROCK MASS IN THE VICINITY OF THE SINGLE WORKING

Beginning of rock floor heaving of working is always related to formation of area III. Its characteristic feature is very hasty growth of deformations during short interval of time. It takes place after achievement of critical values by deformations. From the power point of view there is next process. While a working advances back from face, at the insignificant change of the tense state, there is a rapid emitting potential energy. It is used for additional destruction of structural connections in a rock mass and loosening. After it the mechanical system finds the balance state again.

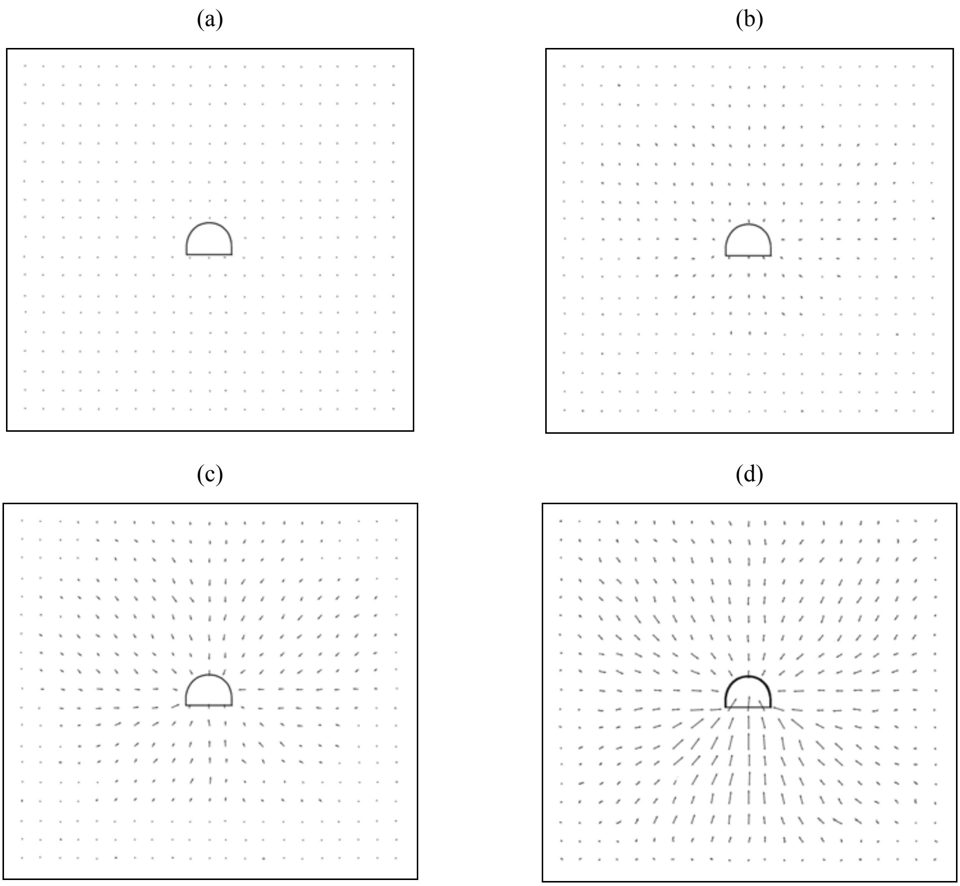


Figure 3. Vectors of angular points displacement on measured grid for homogeneous model at loadings: (a) 0.00 kN, (b) 2.61 kN, (c) 4.56 kN, (d) 7.04 kN.

This process is characterized by the significant change of one parameter (moving on the contour of working) during an insignificant increase of other (components of tensions on an external contour). In physics such transients are named as loss of balance stability. In this case, the elastoplastic stability loss of balance in rock mass occurs.

According to Shashenko (1988) the task of criterial equations determination for the prognosis of rock floor heaving is solved. For the solving of this taskthe a simplified approach was used which is showed in Ishlinskiy (1954) and Leybenzon (1951). The feature of this approach is that the parameter of flowing is entered only in scope terms which take into account the changes of border form during rock floor heaving.

Solving of elastoplastic task lies in basis of researches. In general case at the loss of equation stability of the distorted internal contour it is possible to write:

$$r = 1 + d\cos\theta\,, \tag{1}$$

where d – constant, θ – polar corner.

Thus, on a tense ground-state some protuberant state is laid on. It is caused by the change of internal scope terms. This circumstance is showed on a Figure 4. (K – coefficient, taking into account influence of face of working on external scope terms, ΔK – a small increase of coefficient K, causing a change of internal border geometry).

As a result of solving analytical task (Figure 4), a criterion is got:

$$\bar{\varepsilon}_v r_L^{*2} \ln^2 r_L^* + 2 = 0\,, \tag{2}$$

where $\bar{\varepsilon}_v$ – is an average value of relative increase of volume within the limits of the area of plastic deformations (APD); r_L^* – is a relative critical radius of plastic deformations area.

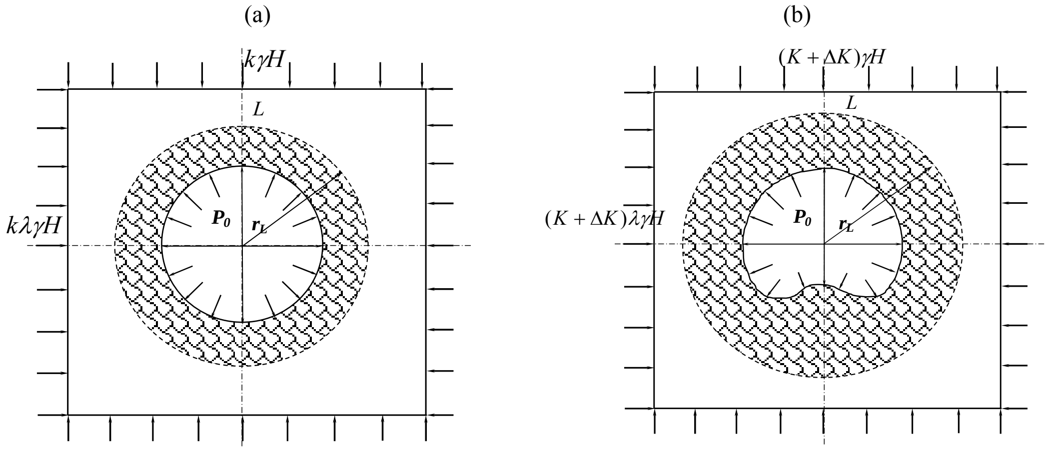


Figure 4. The solution scheme for problem of elasto-plastic stability losses: (a) an initial state of system ($r_L < r_L^*$); (b) the disturbing state of system ($r_L \geq r_L^*$).

The criterion (2) defines a possibility of rock mass transition from one equilibrium limit state to another. This process is accompanied by rock floor heaving.

This process is accompanied by ground rock rising. Its physical essence consists of the following. Subject to the condition of rock floor heaving will happen in making.

$$\bar{\varepsilon}_V r_L^{*2} \ln^2 r_L^* + 2 < 0. \quad (3)$$

It is possible to write down in expression (2):

$$r_L^* = 1 + \bar{\varepsilon}_v^{-0.4}. \quad (4)$$

In Shashenko, Solodyankin, 2007 on the basis of base decision (2), other criterion is offered. Here for the estimation of possible of rock floor heaving displacements of contour are examined:

$$u_y^* = 0.006\left(14.7 + \sqrt{1 - 67.2\varepsilon_v}\right)\left[exp\left(-\frac{1}{NB}\right) - 1\right], \quad (5)$$

where $u_y^* = U / R_0$ – size of critical displacements of soil of contour which the out of control process of rock floor heaving begins at; U – absolute value of displacements of contour of making, $B = \dfrac{r_L^2 - k_{rem}}{1 - r_L^2}$, $N_\theta = \theta\sqrt{\psi + \dfrac{2(1-\psi)}{\theta}}$, $\theta = \dfrac{R_c}{\gamma H}$ – index of terms of development; $k_{rem} = R_{rem} / R_c$ – remaining assurance factor; R_0 – radius of working, R_{rem} – remaining strength of rock on a contour, $\psi = R_p / R_c$ – factor of brittleness of rocks; R_p, R_c – standards of rocks tensile strengths on monaxonic tension and compression.

The offered criterion allows already on the stage of planning to estimate possibility of rock floor heaving in working and plan measures on the decline of harmful influence of this phenomenon.

3 NUMERICAL MODELING OF A DEFORMATION PROCESS AROUND AN UNDERGROUND WORKING

At reaching critical sizes of this area the third level occurs. Thus there is a fast transition of system from the second energy level to the third one. The transition is accompanied by major displacement of working contour. This process is termed as losses of elasto-plastic stability, or rock floor heaving.

The numerical modeling of a rock floor heaving process is carried out with use of finite element method (FEM) at the basis of the third models of the rock failure.

The size of rock floor heaving is accepted on the basis of the real dependences of $u = f(T)$, the typical type of which is resulted on a Figure 5. On this line three characteristic temporal intervals are selected: T_1 – period of realization of the elastoplastic deformations, T_2 – period of active development of process of rock floor heaving of rock and T_3 – period of passive development of process of rock floor heaving.

The temporal interval of T_3 is broken on the areas of T_i, length of which is accepted the equal size of T_2. The size of rock floor heaving u_i and picture of the stress-strain state corresponds to every interval around the working. Modeling of development of process of rock floor heaving in time is executed successive introduction of size of u_i.

The additional displacement of u_i forms the additional tense state. This tension is laid on basis and added up on the basis in accordance with the idea of Ishlinskogo-Leybenzona. Additional moving, similarly put incrementally and their size increases from a zero to the maximum with some increment.

The model allows to imitate development of process in a time and considers floor heaving as discrete quasistatic process.

Calculations started with the solution of a problem for a homogeneous rock mass. Nonhomogeneity of structure of a rock mass from task to task was increasing. The generalized scheme is presented on Figure 5.

It is shown, that sizes of APD in floor exceed original one in 2-3 times at value of a floor heaving being 0.3 m (Figure 6).

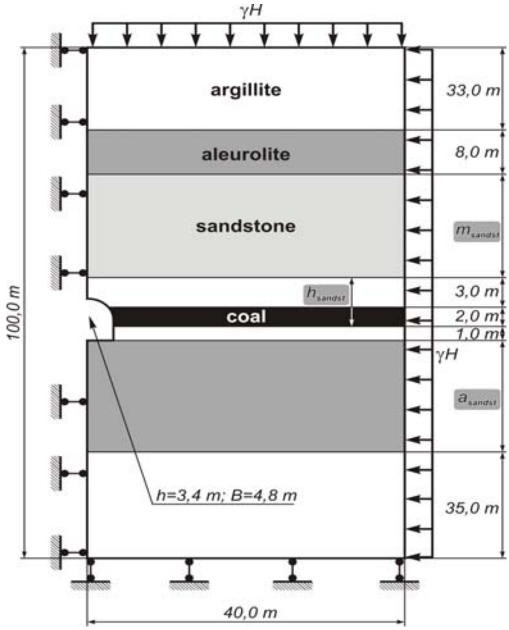
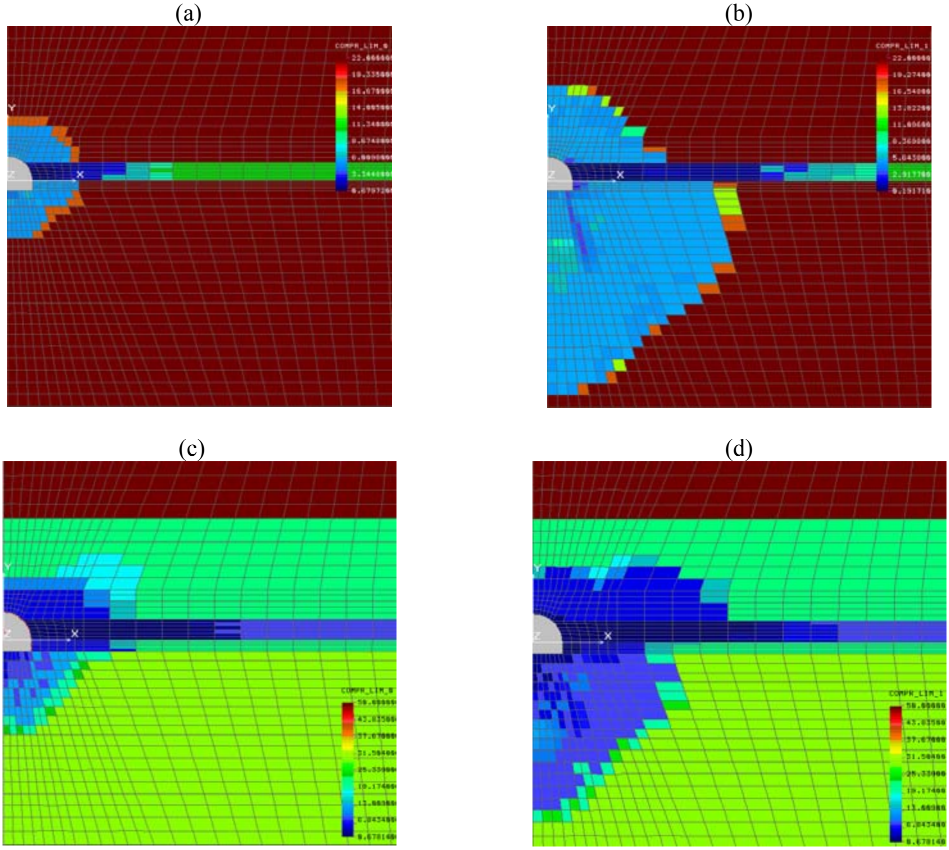


Figure 5. The generalized solution scheme.

I (a) (b)

II (c) (d)



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III (e) (f)

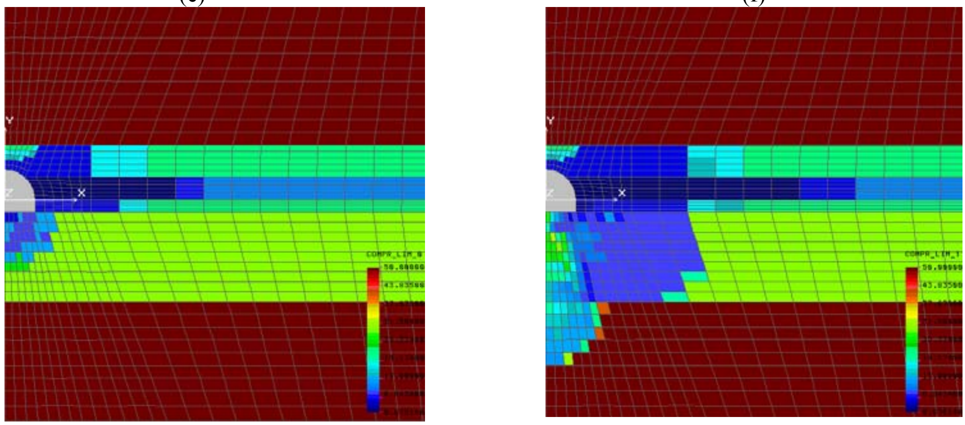

Figure 6. Configuration APD: (a), (c), (e) up to the floor heaving; (b), (d), (f) after the floor heaving, I – homogeneous rock mass including a coal seam; II – rock mass including a coal seam and seam of sandstone in roof (distance between coal seam and sandstone is $h_{sandstt}$ = 10.0 m); III – rock mass including a coal seam and seam of sandstone is floor (distance between coal seam and sandstone is a_{sandst} = 7.0 m).

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New Techniques and Technologies in Mining – Bondarenko, Kovalevs'ka & Dychkovs'kyy (eds)

Numerical modeling used for designing of coal mine roadway support

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ABSTRACT: Design of optimal support for underground headings in hard coal mines, particularly for the conditions with intensified rock pressure is a serious challenge for engineers and coal companies. Not only already existing technological practices are continually being improved but also numerous novel solutions related to such problems are being implemented. The paper presents results of numerical analysis carried out in order to design optimal support for headings driven at the depth of 1100 m and 1300 m.

1 INTRODUCTION

Underground heading drivage below the depth of 1000 m enhances serious complications and difficulties related to the following phenomena: considerable values of primary vertical stress, large deformation pressure, increased range of stress concentration zones around tectonic disturbance, raised temperature of rocks, increased seismic hazard, or frequent influence of numerous exploitation edges. Complex mining and geological conditions, as well as mining at considerable depths, force engineers to apply specific technological solutions (Hoek 1995 & Majcherczyk 2008).

Underground heading designers customarily resort to practical experience obtained in adjacent headings of a particular seam or working level. Also guidelines elaborated for a given mining basin or a specific mine prove useful in such instances (Engineer Manual 1980 & Rock Mechanics Technology Ltd. 2004). Also it is possible to use test results of underground research (Majcherczyk 2007, 2008). The situation looks slightly different in the case of designing opening-out headings, which provide access to subsequent working levels, as then no precise prospecting of mining and geological conditions is available. Hence, rock properties can only be estimated on the basis of bore holes located at large distances from one another. In such conditions it is possible to determine a proper support type only by means of statistical (Sheorey 1997) or numerical analysis (Jing & Hudson 2002).

This paper presents the results of numerical analysis carried out in order to design optimal support for headings driven at the depth of 1100 m and 1300 m. The computations helped to determine dislocation and damage zones around the headings, hence allowing to assume proper support schemes for actual conditions.

2 DEPTH-RELATED CHANGES AROUND THE HEADING

Depth of working occurrence is one of essential factors affecting stability of headings and, subsequently, selection of a particular support type to be applied. In order to determine quantitative variations in the volume of damage zone, numerical analysis was carried out using Hoek-Brown criterion (Hoek 2002, 2007). At the same time, the actual layout and properties of lithologic layers of workings designed at the depths of 1100 m and 1300 m were assumed. The results of the analysis are presented as maps of maximum principal stress with marked plastic zones.

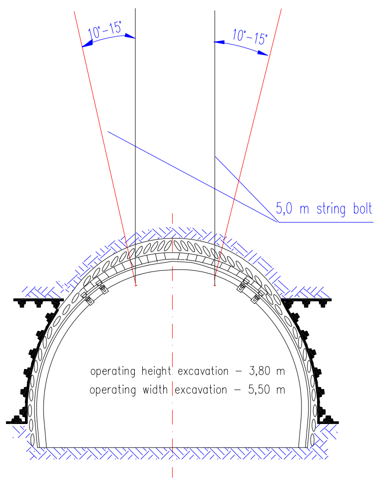

Figure 1. Support scheme for Cw-1 heading.

2.1. *Cw-1 heading at the depth of 1100 m*

Rock mass in the vicinity of Cw-1 heading was significantly heterogeneous. The roof and immediate floor of the heading mainly consisted of argillaceous schist and sandy shale with different thicknesses. The suggested support scheme for this heading included a series of LP10/V29-type steel frames (yield R_e = 480 MPa) with the spacing of 1.0 m and two 5-meter long string roof bolts in each field between the frames (cf. Figure 1).

The numerical model assumed that the walls of the heading contained 0.6 m of argillaceous schist, 1.9 m of coal and 1.3 m of sandy shale. In the immediate roof there was also 7-meter thick argillaceous schist with variable sand inclusions, above which sandy shale deposit with the thickness of 14 m having a thin coal inclusion of 0.4 m was located. Above this layer, sandstone was deposited. The floor consisted of a thin 1-meter layer of argillaceous schist, below which there was a 5.9-meter layer of sandy shale. Further on, 11.6 meters of sandstone and 6.5 meters of sandy argillaceous schist were deposited. Table 1 presents the assumed strata parameters.

An analysis of principal stress σ_1 points to stress concentration in walls (at the distance of several meters) amounting to 42 MPa (Figure 2). Increased stress at a certain distance from the heading's contour results from rock mass plasticity; hence it is related to rock mass damage and relief. The transmission of load by rock mass occurs only outside plastic zone. Also in the floor, increased stress is recorded at the distance of approximately 4 m, reaching the value of nearly 40 MPa. Such a distribution of stress may lead to considerable dislocations of walls and floor. The range of plastic zones in the roof reaches approximately 3.2 m, and damage occurs mainly as a result of shear. In the walls, damage zone reaches the range of 5.3 m, whereas in the floor – approximately 4.3 m. Despite such an extensive damage zone, only in the case of the floor and, rather insignificantly, in the case of the walls, destruction occurs simultaneously as a result of shear and tension.

Table 1. Parameters of rock layers for numerical model of Cw-1 heading.

Rock layer	Bulk density γ [kN / m^3]	Linear elastic modulus E [MPa]	Poisson's ratio ν	Compression strength R_c [MPa]	Empirical Hoek-Brown Parameter m	Empirical Hoek-Brown Parameter s
coal	12.032	1800	0.30	6.104	0.821	0.0004
argillaceous schist/ sandy argillaceous schist	25.805	5000	0.26	44.458	1.094	0.0014
sandy shale	26.178	7000	0.24	51.459	1.436	0.0018
sandstone	25.514	11500	0.22	56.789	2.908	0.0075

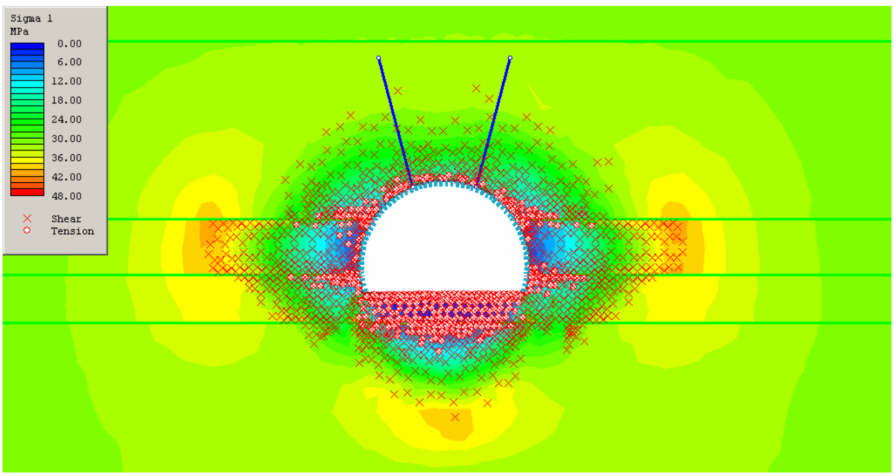

Figure 2. Volume of plastic zone around Cw-1 heading.

In the case of Cw-1 heading, coal-mine tests (*in situ*) were carried out after the heading construction. The tests embraced, *inter alia*, measurement of fracture zone range and volume of roof-bolt loading (Table 2).

Table 2. Comparison of *in-situ* test results with numerical analysis results for Cw-1 testing gallery (seam 401).

Analyzed/measured volume	Coal-mine tests (*in situ*)	Numerical analysis
Range of fracture zone f_s, [m]	4.70	3.20
Maximum axial force in a roof bolt N_k, [kN]	105	280

The comparison of roof-bolt loading obtained from numerical analysis and coal-mine tests clearly indicates that the value of roof-bolt loading in the former case (i.e. determined by means of numerical computations) is higher than the one measured *in situ*. What was decisive for the occurrence of such a disproportion was the assumption of spring-elastic model of rock mass for the computations. In the case of fracture zone range, a lower value was obtained from numerical analysis.

2.2. *Cross heading at the depth of 1300 m*

Numerical analysis assumed a strata layout, in which the following layers are deposited above the working floor: 3.5 meters of argillaceous schist and 0.60 m of sandstone. Immediately in the heading's roof there appear 0.4-meter layer of sandstone, 2.4-meter layer of coal and 7.2-meter layer of argillaceous schist. Further strata include: 7.8 m of sandstone, 6.5 m of sandy shale, 0.7 m of argillaceous schist, 2.0 m of sandstone, 0.2 m of coal and 2.7 m of argillaceous schist. The heading's floor consists of argillaceous schist layer with the thickness of 5.5 m, in which 1.0 m of coal is deposited. Below there is argillaceous schist with a 0.8-meter thick coal layer. The values of physical and mechanical parameters assumed for the analysis are presented in Table 3.

Table 3. Properties of strata assumed for numerical analysis.

Rock type	Young's modulus E [MPa]	GSI index	Poisson's ratio ν	Compression strength σ_c [MPa]	Hoek-Brown constant m_b	Hoek-Brown constant s
coal	3500	40	0.30	10.54	1.374	0.0013
argillaceous schist (sandy)	10000	50	0.26	60.73	1.677	0.0039
sandy shale	15000	60	0.24	94.00	2.157	0.0117
sandstone	17878	65	0.24	71.55	4.298	0.0205

In order to achieve working stability at the depth of 1.300 m, having in mind a predicted considerable volume of plastic zone and serious dislocations, a slightly different scheme of support was assumed, i.e. a LPZ 11/V32 (yield $R_e = 480$ MPa) – type of full framing with the spacing of 0.5 m, alongside with two rows of rope bolts in the roof and one row of steel bolts in each wall. Additionally, bolting with steel binding joists fixed alternately at every second frame was assumed (Figure 3).

Numerical analysis indicates that a plastic zone appears in the excavation's contour: up to 2.5 m in the roof, to 3.5 m in the floor, and up to 2.0 m in the walls (Figure 4). The values of maximum principal stress reach 50÷55 MPa. It is located in the floor-side part of walls, as well as in the upper part of walls, at the distance of approximately 1.5 m from the heading's contour.

The calculated values of dislocations range between 0.07÷0.08 m in the floor and walls, and 0.05 m in the roof (Figure 5). Additionally, a prediction can be made that fixing the support simultaneously with mining the excavation will also result in massive loading of bolts, since the axial force will reach its maximum value almost at all length.

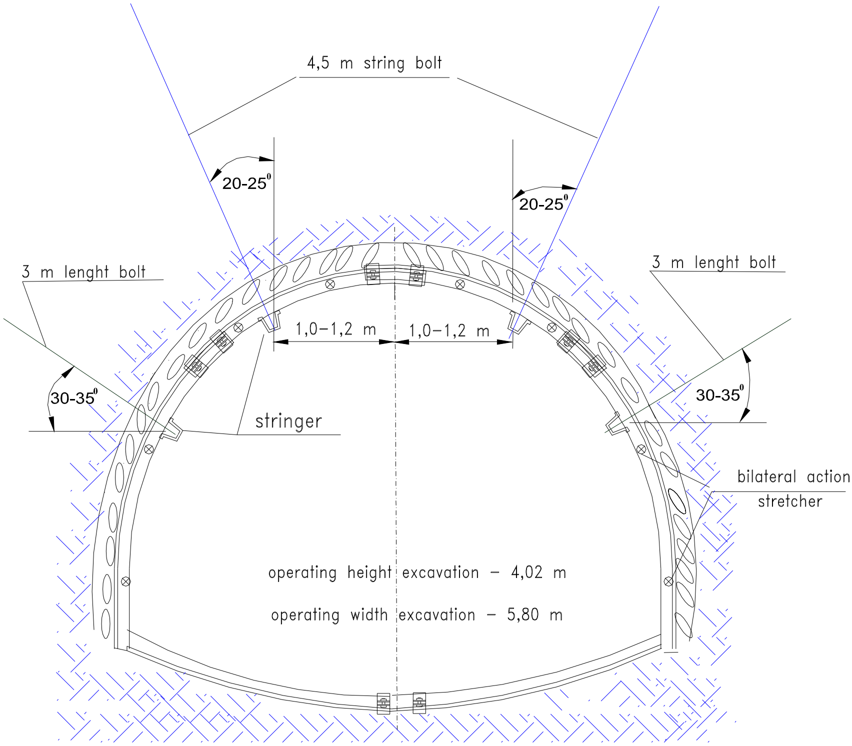

Figure 3. Scheme of cross heading's full support.

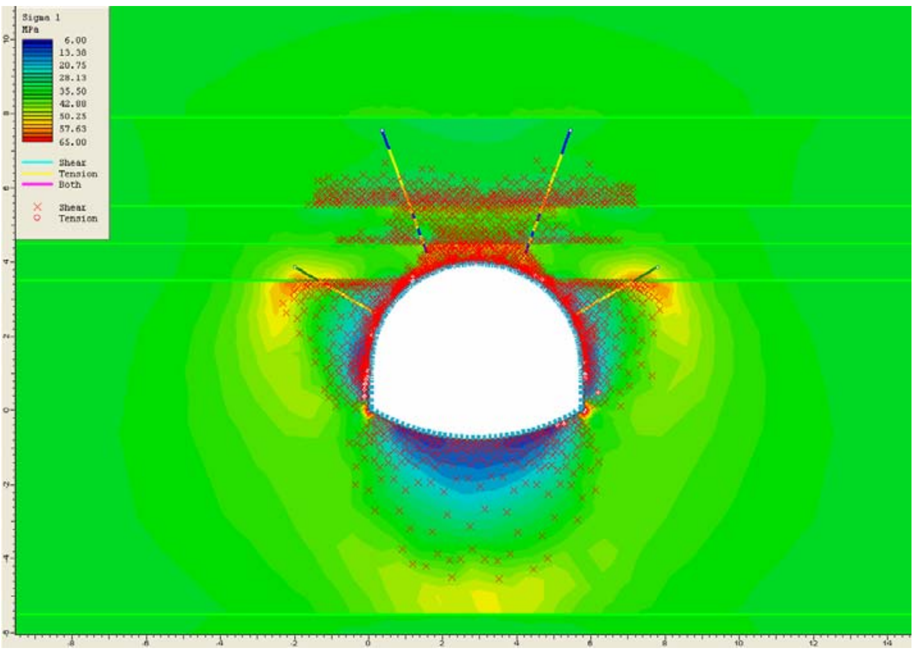

Figure 4. Plastic zone around the cross heading.

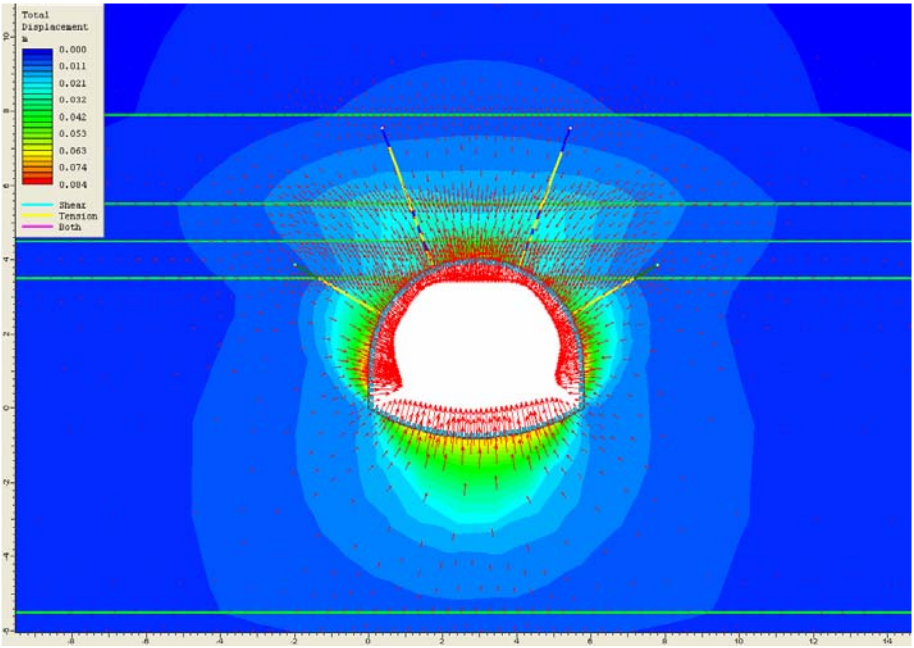

Figure 5. Map of dislocations around the cross heading.

3 CONCLUSIONS

Following conclusions may be drawn on the basis of the analysis presented above:

1. Support of headings should be designed individually and specifically, assuming the existing mining and geological conditions, as well as the intended use and estimated lifetime of a given working. In this respect, particular attention should be paid to designing headings located in complex mining and geological conditions.

2. Implementation of novel schemes of support should be based on a series of coal-mine tests, which should allow to verify earlier-accepted assumptions and to evaluate actual behavior of particular elements of support and surrounding rock mass.

3. In the case of designing headings in completely new (i.e. earlier-unrecognized) conditions, for instance, in new, deeper excavation levels, analytical or numerical calculations prove to be the only way to select the most effective support scheme. Nonetheless, during excavation, a routine evaluation of geomechanical properties of rocks and a regular control of interaction between strata and support should be made. If necessary, some needful alterations in support scheme should be implemented.

4. Support of headings excavated at considerable depths and adjacently to the rocks of low strength and deformation properties should be reinforced by means of support with bolts in the heading face. The application of full frames will allow to limit the volume of dislocation, especially in the floor, whereas bolting with binding joists will guarantee frame stability. Reinforcing roof strata with roof bolts will help to diminish the range of fracture zone and, *ipso facto*, to reduce support loading.

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New Techniques and Technologies in Mining – Bondarenko, Kovalevs'ka & Dychkovs'kyy (eds)

Calculation substantiation of the yield lock model of the polygonal yieldable support with elongated props by means of experiment

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ABSTRACT: Computer modeling of geomechanical system "rock massif-polygonal yieldable support with elongated props ("PYSEP") is substantiated on full diagram of deformation of all its components with a model of yield lock that allows to ensure stability of the computation process with formed wide base of considered parameters of the system. Spatial geomechanical model of displacement around in-seam opening of thin-seam massif of weak rocks is substantiated, taking into consideration finite-elements pattern of real geometrical parameters of a frame support, step bearings, laggings and support area with appropriate conditions of their contact and real mechanical characteristics. The model is performed with help of finite-elements method based on modern computer programs and their applications.

1 INTRODUCTION

Accumulated experience of computer modeling of stress and strain state of a rock massif that contains an opening and its support, indicates on the necessity of preliminary creation of geomechanical model for consecutive solving of a line of tasks with compulsory testing of gained results to determine if they correspond to classical regulations of mechanics of underground constructions and natural observations in concrete mining-geological and mining-technical conditions. This analysis is needed not only for control of modeling process and elimination of possible technical errors in technology of calculations execution but also it is always needed to constantly look for a compromise between computational ability of computers, performance of software and tendency for maximum degree of presentation of real object. With regard to construction of the support, this requirement for modeling of the system "rock massif-support" acquires increased currency. The difficulties lie in the reflection of real design features of the support elements, linear dimensions of which are many times smaller than the average dimensions of the coal-containing strata components, and that requires smaller finite-element mesh and considerably increases time needed for calculations even when using quite powerful computer equipment.

2 STAGES OF MODELLING

Nevertheless, task of adequate reflection of construction of a frame support "PYSEP-15.0" in system "rock massif-support" was considered in the previous researches. Real cross section of a frame made of special profile SCP-27 (special curved profile) with precise reflection of construction of the yield lock, step bearing and interframe shield made of metal mesh, and also shape of the frame support have been modeled.

Special profile SCP is made of steel-5 (Geleskul 1982) with following mechanical characteristics: rated yield point $\sigma_y = 270$ MPa, modulus of elasticity $E^f = 21 \cdot 10^4$ MPa, Poisson's ratio $\mu^f = 0.3$.

These characteristics are accepted for calculation when modeling the full diagram of deformation of steel-5, that includes elastic stage ($\sigma < \sigma_y$), area of almost ideal yield ($\sigma = \sigma_y$) and stage of metal strengthening until it reaches the value of temporal tear resistance $\sigma^t = 560$ MPa ($\sigma_y < \sigma \le \sigma^t$).

Reflection of the full diagram of steel-5 deformation allows to consider occurrence and development of plastic deformations in a frame that are often observed in mine conditions. Such approach contributed to increase of adequacy of modeling to a real object. But construction and operating regimes of the yield lock required conduction of special researches in order to make a full diagram of deformation of the material it is made of.

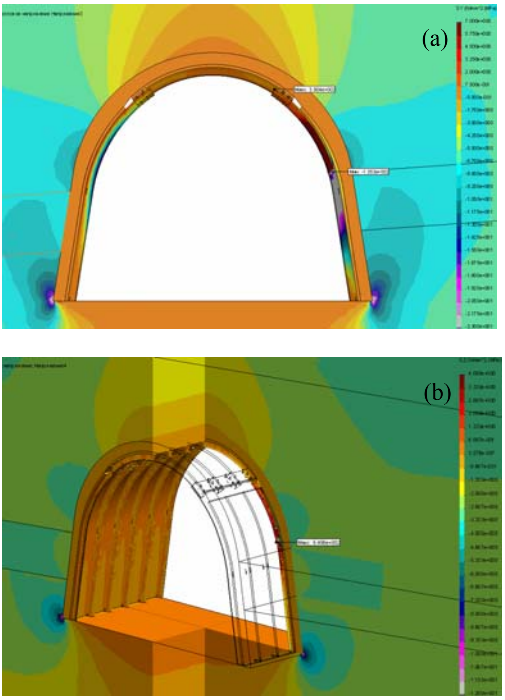

Figure 1. Stress and strain state of the system "rock massif-support" with full reflection of the yield lock's design: (a) distribution diagram of vertical stresses σ_y; (b) distribution diagram of axial stresses σ_z .

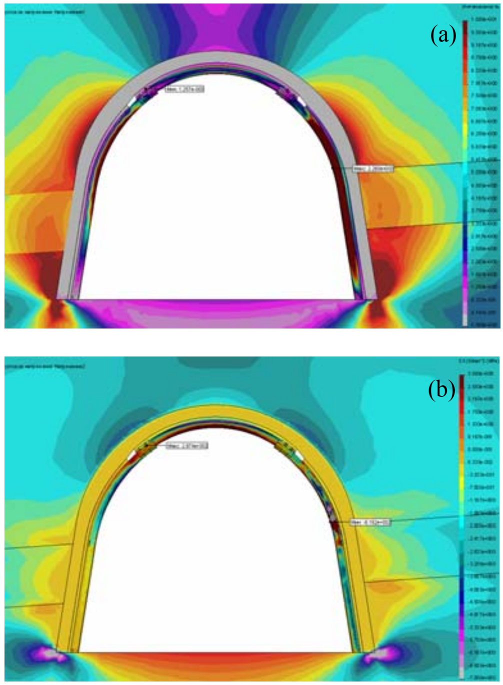

Figure 2. Stress and strain state of the system "rock massif-support" with full reflection of the yield lock's structure: (a) distribution diagram of calculation stresses σ ; (b) distribution diagram of horizontal stresses σ_x .

At the first modeling stage of the yield lock, an attempt was made to precisely reflect its structure with regard to the support "PYSEP-15.0". Resulting from a quite labor-intensive and long process of adaptation of the software with the features of real construction of the lock, initial and boundary conditions of its modeling, it was possible to perform a calculation of stress and strain state of the system "rock massif-support": distribution diagram of the vertical stresses σ_y (Figure 1a) and axial stresses σ_z (Figure 1, b); distribution diagrams of the shown stresses σ (Figure 2a) and horizontal stresses σ_x (Figure 2b). But the calculation of stress and strain state of the system was successful only in elastic condition until the support subsidence (until the yield lock actuation) that is characterized by little dislocation of the opening's contour. An attempt of modeling of full diagrams of deformation of all elements of the system "rock massif-support" has discovered instability of the calculations performance technology using finite-elements method (FEM) because of, as it appears to us, static unbalance of a frame in the area of lock joint. Analysis of the given situation has discovered that because of "point" contact of end sections of the props and roof beam with each other in the lock and a small area of this contact, a possibility of deformation (dislocation) of the frame by a big value emerges (along coordinates X, Y and Z) with quite little increment load that malfunctions in the program and stops stress and strain state computation. The situation gets even more complicated while modeling the yielding process of the lock when roof beam slides relatively the prop because of existing geometry of the frame, the area of their contact on the sector of the lock changes arbitrarily with increase of free dislocation of the frame (static imbalance) in any direction. Also perspective of development of given geomechanical model is needed to be taken into consideration in order to reflect real scheme of the bolts installation, possibility of plastic flow of armouring metal of the bolts and frame, emergence of areas of not only softening but also loosening in adjacent rock massif, taking into account rheological factors and rock fracturing. Thus, consideration of all mentioned factors led to essential complication of geomechanical model that objectively increased instability of technologies of stress and strain state calculations and real indefiniteness of time of overcoming of the given difficulties of modeling. Therefore, it seemed more rational to idealize construction of the yield lock that would allow to stably continue calculations of stress and strain state of the system “rock massif-support”. But simplification of the lock design is not supposed to deviate its deformation-load characteristics when working in the yielding mode. Given condition, as it seems to us, will allow to fully adequately simulate the work of the yield lock.

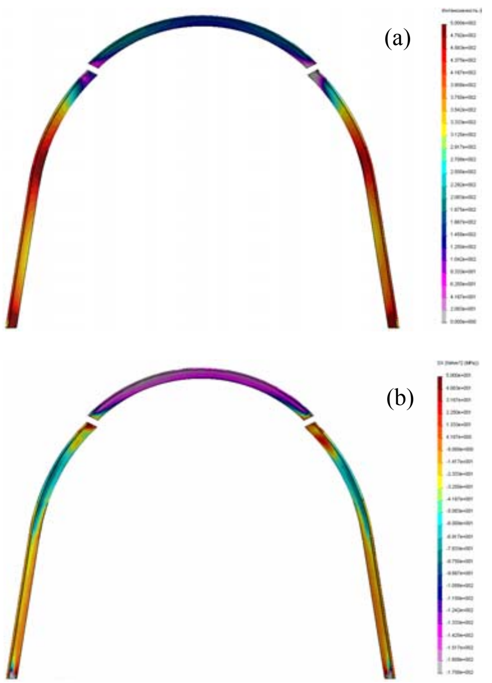

Figure 3. Stress and strain state of the frame support with simulators of the yield locks: (a) distribution diagram of calculation stresses σ ; (b) distribution diagram of horizontal stresses σ_x .

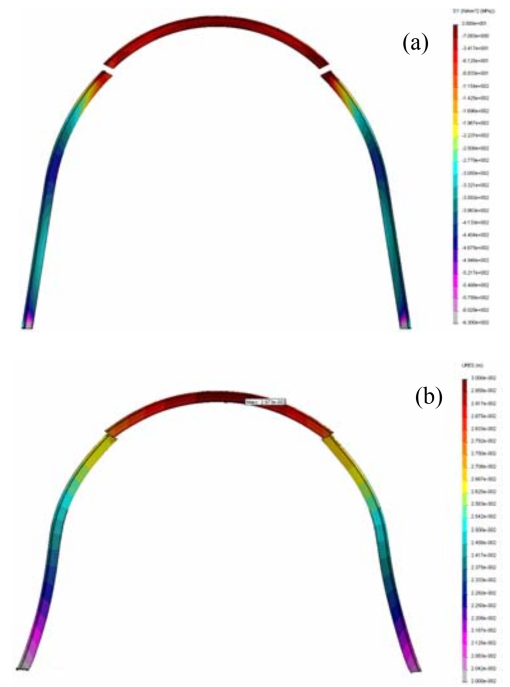

Figure 4. Stress and strain state of the frame support with simulators of the yield locks: (a) distribution diagram of vertical stresses σ_y; (b) distribution diagram of full displacements U.

3 PRELIMINARY RESULTS

Simulation of the idealized yield lock design was performed in two stages. On the first one – a pad was placed in the area of the yield lock location, it is made of easily-deformable material that under influence of compression forces in the frame ensured its yielding. With that, stability of process of stress and strain state system calculation was gained with full diagram of deformation of its elements, and to prove that, some distribution diagrams of deformations and dislocations in a frame support were shown on the Figures 3 and 4. Analysis of distribution diagrams σ_y and σ_x has shown their correspondence to classical regulations of mechanics of underground constructions: roof beam of the frame is under influence of local tensile stresses and little compression stresses σ_y , and the props are subjected to considerable compression stresses σ_y because they accumulate all vertical load on themselves; horizontal stresses σ_x have a quite high value of compression in roof beam (accumulates side pressure of rocks on itself) and sign-changing stresses σ_x act along the contour of the props and this is provided by their bend deformations. Shown stresses σ (using which we perform further strength calculation) indicate on the average level of load on the roof beam (from 20% till 60% from yield point of steel-5) and high load on the props with formation of limited plastic areas in zones of their bearings and at the level of height of the zone of people passage. Artificially increased scale (for presentation) of full dislocations (Figure 4b) indicates on bending of the props at height of 0.8…1.5 m with range of movements of frame into the workings' space $U = 20...30$ mm. These results do not contradict visual observations of the support's condition on an experimental area of a conveyor footway at mine "Yubileynaya" OJSC "Pavlogradugol".

Thus, simulating of a yield lock with deformational pads made of easily-deformable material does not considerably deviate the pattern of load of a frame support.

4 NUMERICAL MODELS

The second stage of the lock simulation is performed in order to achieve adequacy of the real yield process in a numerical pattern. In this connection deformation-load characteristics of the lock presents dependence of reaction of its resistance P from the value of yield U_y (as a rule, in a vertical direction along coordinate Y).

A row of laboratory researches of the frame three-link yieldable support, in particular on the stands of Institute of Mining of A.A. Skochynskiy (Russian Federation) shows (Figure 5a) the kind of work of the lock made of special profile SCP that approaches a mode of constant resistance (Melnikov 1980). Using the same experiments it is established that having correct assembly of the lock, frame support transfers into mode of yield with resistance being $P_y = (0.7...0.8)P_{max}$, that approaches maximum load capacity of the frame P_{max}. Described mode of quasi-constant resistance of the yield lock can be quite adequately expressed by a diagram of the ideal plasticity of material (Figure 5b), that is located in the frame along the coordinates of the locks installations. Parameters of the material should express real deformation-load characteristics of the lock for concrete typical cross-section and a special profile SCP. Yield point σ_y^l of the lock simulator should be such as if when having average normal stresses in cross-section area of SCP (in the middle of the lock's length) with load on the frame equal to 80% from its load capacity, material of the yield lock's simulator would transfer into ideal plasticity and ensure yield of the support.

5 RESULTS

Calculation yield point of the material from which the yield lock is made of is determined according to the following formula

$$\sigma_y^l = \frac{0.8P_{max}}{2F} sin\alpha ,$$

where F – cross-section area of SCP; α – angle of position of the lock's center relatively to the vertical axis of the opening.

So, for standard type of the support PYSEP-15.0 and SCP-27 at the experimental area of conveyor passage $P_{max} = 490$ kN, $\alpha \approx 44^\circ$, $F = 34.37 \cdot 10^{-4}$ m^2. Then, accordingly to the formula we get the value of the lock's material's yield point $\sigma_y^l = 39.3$ MPa, which we choose for the calculation. Modulus of elasticity and Poisson's ratio we take as for steel-5 ($E^l = E^f = 21 \cdot 10^4$ MPa, $\mu^l = \mu^f = 0.3$) and that elastic deformation characteristics would be constant along all frame's contour.

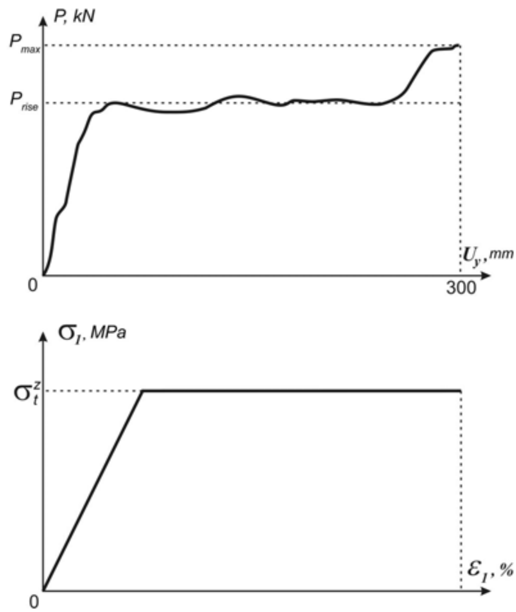

Figure 5. Substantiation of a yield lock model of a frame support made of SCP: (a) deformation-load characteristics of the real construction of a lock; (b) full diagram of deformation of the lock simulator's material.

Simulator of the yield lock is placed in a shape of the cross-section of the appropriate special profile SCP along coordinates of the support locks of real length equal to 400 mm. This helps to simulate a solid frame (from left prop to right one) along the support contour with two pads made of the lock simulator's material that differs from steel-5's mechanical characteristics only by decreased yield point. Such idealization of the yield lock has provided stability of the stress and strain state calculation procedure of the system “rock massif-support” on all deformation diagram of the materials of all its elements.

Conducted test of geomechanical model of the system “rock massif-support” with simulator of yield lock of the PYSEP support did not reveal principle contradictions compared to modern researches of dislocation processes of coal-containing strata near in-seam opening on the one hand and mine observations – on the other hand, and this substantiates reasonability of application of designed model for prognostic assessment of the openings’ condition of block #3 of mine “Yubileynaya” of OJSC “Pavlogradugol”.

Geomechanical model is opened for its improvement (based upon results of mine researches) particularly for consideration of factors weakening the rock and construction-technological features of the building and maintenance of the openings.

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Design of mine working network on the basis of their functional-structural description

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ABSTRACT: Mine working network is a complex multifunctional dynamic system, which is adequately described by the complex of functional, functional-structural and structural-functional models. Functional description of this system provides generating of project's ideas using a theory of solving the invention problems. Searching the options of material carriers of functions is provided by making and estimation of functional-structural models of mine working networks. They reflect the interconnection between the workings and technological functions performing by them. The original number of options of functional-structural models is formed using morphological analysis, and separating the subset of preferable ones from it is carried out on three private criteria using Paretto's procedure.

1 INTRODUCTION

Mine working network (MWN) is a main structural element of coal mine. Nearly 70% of its capital costs are expended onto its construction. In recent years the cost of mining has increased significantly and is 15-20 million UAH (1.8-2.5 million USD) per 1 km of mining working (Gogol 2008). Hence, even a slight decrease in the length of carrying out excavation leads to significant savings in financial resources for the renewal of mining works.

At the same time, practical data show that the total length of the ongoing and supported headings is increasing. Thus, in "Dobropoleugol" company the increase for the period from 1970 to 1990 is 43.7% and if no measures are taken, the extent of heading in 2010 can be 780 km, and in 2015-825 km (Topolov 2005). On the basis of these data they came to conclusion about the necessity to justify the best optimal concentration parameters of mining works before the project implementation that is not carried out at the present moment.

2 MODERN CONTENT OF THE THEORY OF OPTIMAL MINE WORKING NETWORK DESIGN

One of the reasons for present situation is the imperfection of existing methods of designing optimal networks of mine workings. Their analysis showed that the methods based on mathematical programming do not guarantee finding the global optimum.

For methods based on technical-economic comparison of variants and the economic-mathematical modeling a procedure of forming the initial list of options for MWN is complex and ambiguous. So far, the dominant factor is using the experience and intuition of designers that does not guarantee inclusion in the initial set of options. An attempt to solve this problem using combinatorial methods did not produce the desired result, since on the one hand it sharply aggravated the problem of the dimension of the original set, and on the other – the imperfection of individual criteria for its evaluation aimed at reducing lead to formation of excessively large subset of preferred options (up to 80% of the their original number). In this case the procedure of phased optimization was extremely hard. Nevertheless, the general approach to solving the problem by many researchers has been recognized as very promising. It was emphasized that if we can develop a fairly simple procedure of its implementation, it will be a major contribution to the development of a general theory of optimal design of complex systems.

Impact of the degree of favourable conditions of coal layers and their individual parts onto the topology of MWN didn't find any adequate reflection in the known methods of MWN. This factor influences greatly the order of their processing and, consequently, the configuration of MWN.

The proper attention is not paid to explanation of the list of technological functions the mine workings perform and their opportunities to be mixed that doesn't allow finding the specific project ideas on maximum simplification for MWN and increasing their reliability.

There are still actual questions during MWN design concerning the probability nature of the initial information and reasonability taking into account possible errors in initial data. Existing solution methods for these questions are extremely complex and labour-intensive to implement. As a result, practically the dominant models are deterministic models for MWN design.

In connection with the presented above the development of improved methods for the optimal network mining design is a highly relevant scientific and technical problem.

3 RESULTS

The proposed design method of optimal networks of mine workings is based on functional-structural approach, which forms the basis of the functional-cost designing of technical systems (Moiseyev 1988).

At the initial stage of its implementation the information supply for the design process is provided. For mine workings networks it covers a prediction of engineering-geological conditions that affect the topology of the MWN and techno-economic parameters of the mine workings operation.

A particular importance in this case should be given to identifying heterogeneous environment conditions of coal beddings and their individual sites. These problems are solved using the methods described in the papers (Okalelov 1988, 2001 & 2008).

In the second stage the generation of principal project ideas for mine workings network is made. With this purpose, the generalized functional model (FM MWN) is used, which is a logical-graphical representation of the content and interlinks of technological functions performed by the mine workings network (Okalelov 1992). It is presented in Figure 7 and includes the following functions: F_1 – to provide the coal production; F_2 – to provide the access from the surface to mineral extraction area; F_3 – to provide transfer of mineral to the surface; F_{21}, F_{22}, F_{23} – to provide opening, preparing and formation of production face; F_{31}, F_{32}, F_{33}, F_{34}, F_{35}, F_{36}, F_{37}, F_{38}, F_{39}, F_{310} – to provide people's movement, placing the equipment, coal and material transporting, power supply, rock transporting, ventilation of mines, water supply, water pumping, safety for the layers with dangerous emissions; $f_{211}, f_{212}, f_{213}, f_{214}$ – to provide access from the surface to transport, ventilation and drainage horizons, access to coal layers; f_{221}, f_{222}, f_{311}, f_{223} – to divide coal field into the areas, to group the seams, to prepare the seams; $f_{231}, f_{232}, f_{233}$ – to form taking-out fields, sites (columns), to set the heading direction; f_{311}, f_{312} – to provide people's transporting and escape way; f_{361}, f_{362} – to move the rock to OKD and to put it out to the surface; f_{371}, f_{372}, f_{373} – to supply with fresh air, to refresh the outgoing air jet, to delete the outgoing air jet; f_{374} – to provide degasation of coal-rock massif; f_{391}, f_{392} – to move water to main water storage and pump it out to the surface.

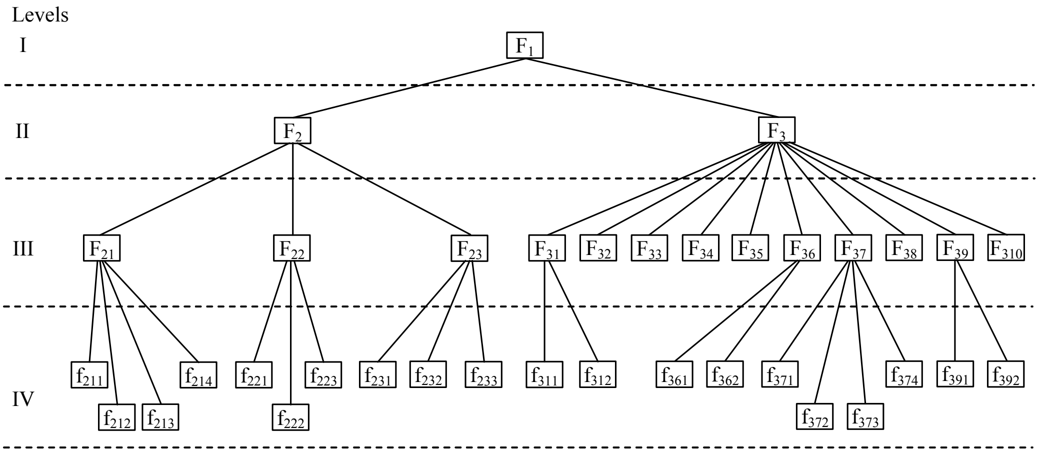

Figure 7. General functional model of mine working network.

Function F_1 – is the main, F_{21} and F_{22} are the main, the rest are secondary of III and IV levels.

An analysis of the MWN FM stipulates an exception of those functions from it which perform in a specific context and are not required, as well as combined with the implementation of related ones. After that, the possibility of excluding the remaining functions is revealed by finding the answer to the question: how will the superior function be performed, after the exclusion of inferior function, which provides it? Technical contradiction occurring then is overcome with the help of heuristic methods of theory of inventive problem solving. Analysis of great number of inventions in the field of mine workings topology showed that the most frequently used methods of inversion (vice versa), fragmentation, taking out, local quality, integration and versatility.

While generating ideas of the project then several kinds of design decisions may be obtained each of which corresponds to a functional model of its own. Choosing the best of them at this stage using techno-economic calculations is impossible because of their complexity and lack of necessary information. Therefore the individual evaluation criteria of perfect functional model have been developed, which characterize the degree of function concentration k_{kf}, presence of negative affects for environment after their implementation k_{on} and level of neutralization of these affects k_{on}: $k_{kf} = \frac{1}{n-1}\sum_{i=1}^{n-1}\frac{F_i}{F_{i+1}}$, $k_{on} = 1-(F_{on}/F_в)$, $k_{on} = N_{nn}/(F_n + N_{on})$, where F_i и F_{i+1} – number of functions at *I* and *i*+1 levels; n – number of FM's levels; F_{on} – number of functions, resulting in negative affects; $F_в$ – general number of secondary functions for all levels of MWN FM; $N_{нп}$ – number of neutralized negative affects; N_{on} – general number of negative affects; $F_н$ – number of functions required for neutralization of negative affects.

These criteria meet the standard requirements for solving of multiobjective problems: the dimensionless, independence, equivalence and presence of sense. Therefore, for the integrated assessment of FM their average k_u is used. For the initial MWN k_u is equal to 0.400. Performed calculations showed good sensitivity to the degree of perfection of MWN FM, which realization a priori requires the least expenses.

After selecting the most advanced functional model searching for the variants of the effective implementation of all listed functions is performed. This problem is solved by designing and evaluating the functional-structural models of the network mining (MWN FSM), which represent different variants of combining the functions with the structural elements of the system. Their formation is made separately for the opening, preparation and excavation systems through morphological analysis. General view of MWN FSM is presented in Table. 1.

Table 1. General view of functional-structural model of MWN.

System elements	Functions						
	Main	F_i			F_{i+1}		
	Secondary of the 1-st level	F_{ij}	F_{ij+1}	F_{nm}	$F_{i+1,j}$	$F_{i+1,j+1}$	$F_{n+1,m+1}$
	Secondary of the 2-st level	f_{ijg}	$f_{ij+1g+1}$	f_{nmt}	f_{i+1jg}	$f_{i+1j+1g+1}$	$f_{n+1,m+1,t+1}$
1 2 … $N_э$	…	+ … +	 + …	 …	+ …	 …	 … +

In Table 1 index + marks necessary functions, performed by certain heading, and index + potentially possible.

As the initial elements of MWN FM are the certain mining headings, performing the certain combinations of technological functions, then it leads to essential development of mining heading classification which is adapted to the morphological analysis procedure. These classifications are given in Table 2, 3, 4.

According to the conformity indicator of the above subset there was determined the opening, preparing and developing headings. Within each group there are determined subgroups of main and additional, principal and secondary headings.

Main and additional differ on the indicator of access providing to the mineral. Main headings are intended for transporting of mineral, and secondary – for performing the rest technological functions. For each group and subgroup combinations the topological indicators are determined as well as variants of combining the technological functions performance.

Based on this classification the original list of mines and their characteristics are formed which correspond to the specific geological and technological conditions of mining operations. For this purpose, an area of rational using of mines and their characteristics is determined. When choosing indicators, characterizing the spatial arrangement of openings, it is taken into account the optimal order of coal seams mining and their plots, bedding in dissimilar geological conditions.

Table 2. Classification of opening headings.

Headings	Classification of opening headings	
	Characteristics	Features
Main and additional, principal and secondary	Type	Vertical, inclined, horizontal
	Place of heading	Through coal, through the rock
Main and additional principal	Combination variants of technological functions performing	F_{33}, f_{362}, f_{373}; $F_{33}, f_{362}, f_{373}, f_{374}$, F_{35}, F_{38}; F_{33}, f_{362}; F_{33}; F_{33}, f_{373}; F_{33}, f_{371}, F_{34}; F_{33}, F_{35}, f_{361}, f_{311}, f_{312}, F_{34}, f_{371}; F_{33}, f_{361}, f_{373}, f_{374}; F_{33}, f_{361}, F_{34}, f_{371}; F_{33}, f_{361}; f_{371}
Main and additional secondary	The same	f_{311}, F_{34}, f_{371}, f_{392}; f_{362}, f_{311}, F_{34}, f_{371}, f_{392}; f_{373}, f_{312}; F_{34}, f_{373}, f_{392}; f_{361}, f_{391}; f_{311}, F_{34}, f_{373}; f_{311}, F_{34}, f_{371}
Main and additional, principal and secondary	Bedding place along the line of stratum stretching	Central
Main and additional secondary	The same	Flanking, unit-type
Main and additional, principal and secondary	Bedding place along the line of stratum falling	Central, concerned (to upper or lower boundaries of mine field, horizon boundaries)

Table 3. Classification of preparing headings.

Headings	Classification of preparing headings	
	Characteristics	Features
Main and additional, principal and secondary	Type	Inclined, horizontal
Temporary	The same	Vertical, inclined, horizontal
Main and additional, principal and secondary, temporary	Place of heading	Through coal, through the rock
Main and additional principal	Combination variants of technological functions performing	F_{33}, f_{361}, f_{373}; $F_{33}, f_{361}, f_{373}, f_{374}$, F_{35}, F_{38}; F_{33}, f_{361}; F_{33}, f_{373}; F_{33}, f_{371}, F_{34}; F_{33}, F_{34}, $F_{35}, f_{311}, f_{312}, f_{361}, f_{371}$; $F_{33}, f_{361}, f_{373}, f_{374}$; F_{33}, f_{361}, F_{34}, f_{371}; F_{33}, f_{361}; f_{371}
Main and additional secondary	The same	f_{361}, f_{311}, F_{34}, f_{371}, f_{391}; f_{373}, f_{312}; f_{311}, F_{34}, f_{373}, f_{391}; f_{361}, f_{311}, F_{34}, f_{371}; f_{373}, f_{391}; f_{311}, F_{34}, f_{373}; f_{311}, F_{34}, f_{371}
Main principal and secondary	Bedding place along the line of stratum stretching	Central
Additional principal and secondary	The same	Central, flanking, in the center of wing of mine field, unit, panel
Temporary	The same	At the boundaries of taking-out fields, central
Main principal and secondary	Bedding place along the line of stratum falling	Central, concerned (to upper or lower boundaries of mine field, horizon boundaries)

Table 4. Classification of development headings.

Headings	Classification of development headings	
	Characteristics	Features
Main and additional, principal and secondary	Type of heading	Horizontal, inclined
Additional principal	The same	Horizontal, inclined, vertical
Additional secondary, temporary	The same	Horizontal, inclined
Main and additional, principal and secondary, temporary	Place of heading	Through coal, through the rock
Main and additional principal	Combination variants of technological functions performing	$F_{33}, F_{34}, F_{35}, f_{311}, f_{312}, f_{371}$; $F_{33}, F_{34}, F_{35}, f_{372}, f_{373}, f_{374}$; $F_{33}, f_{311}, F_{34}, F_{35}, f_{371}$; $F_{33}, F_{34}, F_{35}, f_{311}, f_{361}, f_{371}$; $F_{33}, F_{34}, F_{35}, f_{311}, f_{372}, f_{373}, f_{374}, F_{33}$; $F_{33}, f_{312}, F_{311}, F_{34}, f_{371}$; f_{372}
Main and additional secondary	The same	$f_{312}, f_{372}, f_{373}, f_{374}$; $f_{311}, F_{34}, F_{35}, f_{371}$; f_{312}; F_{34}, F_{35}, f_{371}; $f_{312}, F_{34}, F_{35}, f_{372}, f_{373}$; f_{312}, f_{373}; $f_{312}, F_{34}, F_{35}, f_{361}, f_{372}, f_{373}, f_{374}$; $f_{311}, F_{34}, F_{35}, f_{372}$
Main principal. Temporary secondary	Description on the number of longwalls maintain	Combined, individual
Main secondary. Temporary principal	The same	Individual
Main principal and secondary. Temporary principal, parallel to the main	Order of heading	After longwall, before longwall, passed beforehand
Temporary principal, normal to the main. Temporary secondary	The same	Passed beforehand
Main principal, secondary. Temporary, parallel to the main	Cancel order	Is cancelled after longwall developed, is supported for repeated usage, is cancelled following the longwall
Temporary principal and secondary	The same	Is cancelled after longwall developed
Main principal and secondary	Placing relatively to the bedding elements of stratum	Along stratum bedding, stratum falling

After making a morphological list of openings and their characteristics the conditions of their incompatibility are formulated. Taking these conditions into account, in the process of generating a set of morphological variants leads to its significant reduction. A preliminary assessment of the quantity (N_κ) of those variants is therefore necessary to select the algorithm for generating preferred subsets from them.

For pairwise incompatible non-intersecting characteristics $N_\kappa = N_o - \prod_{i=p}^{k} n_i^u \cdot \prod_{i=t}^{m-k} n_i$, where N_o – initial number of variants $N_o = \prod_{i=1}^{m} n_i$, n_i^u – number of incompatible characteristics for i-th element (heading); k – number of p- elements, which contain pairwise incompatible features; t – element's order number, not containing incompatible characteristics; n_i – number of characteristics of elements m.

For pair-wise incompatible intersecting characteristics of related elements

$$N_\kappa = N_o - \prod_{i=p}^{k} n_i^u \cdot \prod_{i=t}^{m-k} n_i - \sum_{i=s}^{d} \left[\left(n_i - n_i^u \right) \cdot n_i^u \cdot n_{i+1} \right],$$

where d – number of elements, which contains pairwise intersecting incompatible characteristics; s – element's order number, which contains intersecting pairwise incompatible characteristics.

For all other conditions of incompatibleness the upper limit N_κ is determined on the developed algorithm, based upon graph's theory in combination with matrix method and morphological analysis. The developed algorithm allows determining the number of technically performed variants of any system that makes it universal.

Finding from the initial variety of variants the subset of preferable is performed by using Paretto's procedure. For its realization there were developed three private functional-structural criteria: compatibility degree of functions k_{cF}, their width $k_{шF}$ and concentration of structural elements k_N

$$k_{cF} = \frac{\sum_{i=1}^{m} N_i^c F_i^c}{N_{com} F_{com}}, \quad k_{rf} = \frac{\sum_{i=1}^{m} N_i^n F_i^n}{N_{com} F_{com}},$$

$$k_N = \frac{\sum_{i=1}^{m} N_i n_i}{N_{com} n_{com}}, \qquad (1)$$

where N_i^c – number of elements, performing i-th set of functions combination F_i^c; m – number of variants of functions combination; N_{com} – total number of elements; F_{com} – total number of functions performing; N_i^n – number of elements, performing i-th set of potential functions combination F_i^n; N_i – number of elements (headings), maintaining the i-th set of mine field parts n_i; n_{com} – general number of mine field parts, maintained by all headings.

These criteria are obtained using matrix approach and accordingly characterize specific weight of filled matrix cells $N_{com} \times F_{com}$ and $N_{com} \times n_{com}$.

Criterion k_{cF} characterizes the multifunctionality level of of system elements, k_{rf} – its adaptive ability and k_N – concentration level for mine works.

Applying only 3 local estimative criteria allows determining of about 10-12% morphological variants from the initial number into the number of more preferable ones that makes it able to significantly smooth the actuality of its dimension problem.

From formed subsets of preferable complexes of opening, preparing and developing headings the initial variety of variants of MWN FM is synthesized, which includes all mentioned groups of headings. Synthesis and estimation method for these variants is similar to considered above. As a result the subset of preferable synthesized variants of MWN topology is formed where it is chosen the optimal variant using the criterion (2).

$$\sum_{t=0}^{T_O} \left[\sum_{i=1}^{n} P(B_i)(Ц_{ti} - C_{ti}) + \sum_{j=1}^{m} P(A_j)(Ц_{tj} - C_{tj}) \right] Z_{np.t} \geq$$

$$\geq \sum_{t_1=0}^{T_1} K_{t_1} (1+E)^{t_1}, \qquad (2)$$

where $P(B_i)$ and $P(A_j)$ – probability of developing the stocks on the i-th site with favourable and j-th site with unfavorable bedding conditions; n and m – number of favourable and unfavorable sites within the mine field thereafter; $Ц_{ti}$ and C_{ti}, $Ц_{tj}$ and C_{tj} – price and totally mined cost price per excavation of 1 ton of coal at i-th favourable and j-th unfavourable sites relatively, UAH.; $Z_{np.t}$ – industrial reserves, developed in t-year after a mine put into operation or its reconstruction, tons; K_{t_1} – capital expenses in t_1-year of construction or reconstruction, UAH; E – accepted standard for benefit ability of capital expenditures; T_0 – recoupment period of initial capital expenditures, years; T_1 – period of mine's construction, years.

Risk level estimation for realization of the optimal variant is made by generating of quasirandom values of total effect $Э_c$ from development of deposit, which is calculated using the optimal criterion (1) due to excluding from the probable total income the value of given initial capital costs.

Combination of factors and indicators which influence $Э_c$ is generated using planning principles for experiments and morphological analysis.

In the complex of formed values $Э_c$ a theoretical law of their distribution is set. After that for minimum hurdle in terms of investor value of the total income $Э_{c\,min}$ the probability of its unconfirmed α is calculated, which is taken into account when calculating the potential damage from denial $У_1$ or agree to invest the project $У_2$: $У_1 = (1-\alpha) \cdot Э_{c\,min}$, $У_2 = \alpha \sum_{t=0}^{T_1} K_t (1+E)^{t_1}$, according to these results the final decision on reasonability of investment into realization of optimal MWN project is accepted.

4 CONCLUSIONS

Based on the obtained results we can conclude the following:

– it is proved that mine workings network (MWN) is adequately described by complex of their functional and functional-cost models;

– project's ideas generation is more effective on the basis of analysis of organization level of functional models of MWN and their estimation using functional criteria;

– the developed classification of mine headings allows forming of morphological variants of their networks and determining the subset of preferable ones from them using developed functional-structural criteria;

– determining from the subset of preferable variant is made by means of proposed optimal criterion, based on the internal income standard and distribution of probabilities of an arbitrary number of joint and mutually exclusive events that characterize the different degree of homogeneity and complexity of the conditions of coal seams bedding;

– risk level estimation for realization of the optimal variant is made on the basis of analysis of theoretical distribution law for quasirandom values of total income from project performance and probability of its non-confirmation volume from the investor's point.

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Conditions for safe underground gasification of lignite in Poland

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ABSTRACT: The paper contains the analyze of possibilities and conditions for underground gasification of lignite. Standing regulations with regard of lignite deposit utilization as well as requirements concerning keeping the sustainable development policy while implementing this method are presented. Particular attention was paid to geotechnical problems encountered during lignite mining in Polish deposit conditions. Geotechnical protection of safe lignite utilization by its gasification involves many issues such as presence of underground water, predicted land subsidence and deformation, mining technology etc. Great importance is attached to surface facilities and buildings safety and to protection of urban areas. The analyze showed that underground gasification of lignite gives a great chance to utilize the big part of resources which are very difficult to extract using other methods, due to the surface conditions. It offers also a diversification of energy source.

1 INTRODUCTION

Methods of lignite utilization by its gasification have been known for a long time and have been successfully applied during last decades, especially the underground gasification. Most of output in this method development comes from numerous studies and applications carried out in former Soviet Union countries. Those achievements were implemented in dynamic development of many countries.

Currently the research studies on using the underground gasification of lignite are carried out in Poland funded by the science grants but also by KGHM Polska Miedź SA. The dominating opinion in the company says that the word experience should be utilized in order to shorten the research period before implementation of commercial installations. In light of previous technological achievements such approach will substantially accelerate the possibility of practical application of underground gasification of lignite (PZWB).

One of the barrier, in implementing this method, is a way of reporting the lignite resources and legal procedure while applying for exploration and mining concession. Standing regulations concerning the above issues is very favorable for the open pit mining methods. According to these regulations, underground gasification of lignite is regarded as lignite mining through unconventional method.

2 GEOLOGY OF LIGNITE DEPOSIT AREAS IN RESPECT OF PZWB IMPLEMENTATION

Geology of lignite deposit areas in Poland is very complex. The lignite beds are usually horizontal or with little inclination but also a disturbed beds are encountered. Thus the techniques of underground gasification specific for horizontal beds will dominate. They are much more difficult in practice than the ones applied in case of deposits having big inclination.

The lignite may have a form of many beds or one very thick bed, for example 35 m ore more, but also it may be divided into several layers. Typical is high humidity of lignite beds, reaching 40-50%, what in turn causes problems with dewatering and dosing the vapor necessary in gasification process. Moreover, in the overburden multi-layer complex systems of different soils such as: sands, clays, muds, aggregates i.e. mostly crushed formations, occur. Very often the over-lignite beds are aquifers, what is a great challenge while draining them and preparing the operation of underground gas generators. Complex structure of the overburden requires such location of gas generators to have the impermeable soils within the zone of continuous deformations, therefore at the proper distance over the high temperature sectors. It is aimed to protect the tight bed from the thermal impact and the structural changes. If it is impossible, the additional engineering methods of sealing the zones over the gas generators should be used.

Separate and very important issue is a protection of surface and underground water reservoirs. Since the researches show attenuation of contamination 30 meters from its source, it is deemed that minimal distance from the reservoir should be 40 m. Utilization of such 10 m long safety zone is, however, not always sufficient. Additional analyze of geological and water conditions of the deposit area where gasification is planned becomes necessary. In such cases model studies using advanced software, basing on finite elements method, are the most justified.

Table 1. Hydrogeological conditions.

Utility aquifers and main underground water reservoirs	Beds occurring below utility aquifers (minimal distance 40 m), absence of main underground water reservoirs in close vicinity of prospective extraction field (minimal distance – 2 km)
Porosity of surrounding rocks	Rocks above and below the deposit sholud have smaller gas permeability than the lignite bed, Thickness of rock with low permeability surrounding the lignite bed should be 1-2 m for 2 m thick lignite bed or 2-4 m for 3-10 m lignite bed
Water inflow	Maximum volume of water necessary gas production is 0.5 ton per 1 ton of gasified lignite, water surplus should be pumped out
Filtration properties of rocks	Lignite bed porosity vs. surrounding rock porosity should not be smaller than 18:20

One of the crucial issue while using PZWB method is protection against excessive subsidence, especially of non-continuous nature. In many countries using PZW installations, the surface subsidence problem is of smaller importance due to location of the site where the extraction is carried out or because of specific outcrop structure. It may be a depopulated land or located on mining area, where extraction using other methods takes place or the overburden is formed of soft rocks. However, in Poland the population is high, the surface infrastructure is complex: roads, numerous buildings, railways, power supply lines etc. In such situation the land subsidence is not acceptable.

The overburden occurring over the lignite deposit is formed of loose soil beds. These are sands of different granulation, often water saturated, clays, silts, muds and others. Layers of such soil are often interrupted, have numerous lens and admixtures and structural discontinuities. Under such conditions, subsidence is integral process accompanying this type of extraction. In case of PZW the subsidence is bigger because the over lying soils are the subject of additional deformations resulting from thermo-consolidation processes i.e. changes of humidity, swelling and shrinking, cracking and uncontrolled water flows.

3 SUSTAINABLE DEVELOPMENT AND PZWB METHOD

Objective with regard of sustainable development is an effort to maintain the balanced relations between technical progress, environmental protection and economic development. PZW method is relatively cheap, however, in case of adverse deposit parameters costs of its application increase. Also protecting the environment from its negative impact may be expensive. In this instance the location of deposit is of vital importance, Table 2.

Table 2. Site conditions for PZW installation.

Area required for PZW installation	minimal area for pilot installation is 50-100 ha (0.5-1 km^2), for commercial installation over 100 ha, changing location
Safety conditions	Minimal distance from: residential areas (1-3 km), rivers and lakes (1-3 km), protection areas (5 km), operating mines/mining areas (5 km), closed mines/workings (3 km), power lines and railroads (1-3 km)
Legal conditions	Preparing the opinion on environmental impact, identifying land property, getting concession for deposit exploration, geological study concerning probable mineral reserves, public consultation, delimitation of mining area and getting concession for deposit extraction, buying the land

When planning the location of area for underground gasification, the critical elements of infrastructure and environment, present over the deposit, must be taken into consideration. Among these elements are: land development (buildings, telecom and power lines, gas mains, roads, railroads), level of land use, distance from old mine workings and operating conventional mines, environmental conditions (soils, forests, protection areas, lakes, rivers). All those factors may be decisive with regard of possibility of using the method in agreement with environment and community development.

Currently high capacity installations for underground gasification are widely used. However, they are not suitable for very big deposits, which, when mined using open pit method, give 20-30 million tons per year during many years. Conventional method does not need also great volumes of backfilling material preventing the subsidence. While the PZW method does not guarantee the development of whole big deposit but only a part of it, due to the demand for huge amount of backfilling material.

In such case constructing many smaller installations spread over the big area seems to be reasonable. It is important to diversify the energy sources and to limit the power loses on big transmission lines.

Underground gasification systems give the wide range of products with different application such as energetic gas, heat, power energy, liquid fuels (oil and gas) and numerous valuable but less commonly used products. For first Polish implementations the most reasonable is production of energetic gas, power energy and heat energy manufactured in the co-generation systems.

4 GEOTECHNICAL PROTECTION IN PZWB METHOD IMPLEMENTATION

Like in every mine, the geotechnical issues are extremely important in case of underground lignite gasification. They decide about the possibility of using this method. Geotechnical processes occurring during the lignite mining from Polish deposits concern many elements of their development. Geotechnical protection aimed to assure the safe utilization of lignite reserves using this method refers to underground water, predicted land subsidence and deformations, mining system and environmental protection.

When PZW method is used, the effective drainage of the part of lignite deposit destined for gasification is of great importance. In some cases not only drainage will be necessary but also silting-up the extremely permeable aquifers. It may be necessary to maintain the water inflow on the level enabling the proper gasification process. Otherwise the syngas parameters will be on inadequate level.

Under Polish conditions extremely important is backfilling the voids formed as a result of gasification. The geological structure and processes occurring inside the overburden layers show that these voids will be formed generally in the lignite bed and inside the overburden. It is a result of roof collapse due to the stability loss. The overburden falling into the voids and solid products of gasification are porous. Blow gases flow well through the empty spaces despite the rubble formation, and at the same time the loosen overburden and empty rooms shift upwards. In such case the backfilling may be held even in the over lignite zone. It requires the proper identification of room location in order to fill it with backfilling material as fully as possible. Both hydraulic and pneumatic methods may be used there. When the beds are thicker and there are several gas generators (lignite beds) caving results in adding the post-reaction voids what in turn allow for better filling them. Thus the backfilling effectiveness is better.

Further subsidence protection may be achieved by using the proper organization of mining operations. Gasification may be carried out along the line courses, leaving the temporary, safety pillars between them. After gasification of lignite along two lines and backfilling them, the gasification of pillar between two lines can be started. Such organization of mining effectively prevents the subsidence and at the same time allows for efficient utilization of deposit as well as reduction of loses.

Using adequate and properly selected geotechnical solutions is a condition for correct operation of the installation and protection against adverse impact on the environment.

5 CONDITIONS FOR IMPLEMENTING THE METHOD IN POLAND

The basis during implementation of PZW method and constructing the installation for underground gasification are the project assumptions which describe the function, power and period of enterprise in details and thereby specify the minimal level of lignite reserves. When it is planned to extend the project it is necessary to provide the possibility of mining further mining fields.

Table 3. Specification of qualification criteria for lignite beds destined for underground gasification.

Qualification criterion	Characteristics
Type of lignite	xylite, xylite-earthy, earthy, bituminous
Physical & chemical properties of lignite critical for gasification process	Preferred lignite with ash content below 20%, and low sulphur content
Thickness of lignite bed	More than 2 m, optimal – 4 m and more
Depth of lignite bed	Recommended more than 150 m
Inclination of lignite bed	preferowane pokłady poziome lub o niewielkim kącie nachylenia
Rock mass and surrounding rocks tectonics	Preferred absence of cracks and big tectonic disturbances (faults)
Lithology of overburden rocks	Preferred grounds of small permeability (clays, silts, muds)
Water conditions	Preferred beds laying below utility aquifers (minimal assumed distance – 40 m), absence of main underground water reservoirs in the vicinity of planned installations
Reserves volume	– research installation; minima reserves of 75,000 – 450,000 tons of lignite, – commercial installation; min. 3.5 million ton sof lignite.
Preliminary safety conditions	Preferred absence of surface facilities, rivers, lakes, protected areas

Planned installations for underground gasification must meet the requirements of:

– *Geological and mining law;*
– *Power energy law;*
– *Environmental protection law;*
– *Building law;*
– *Land development plan.*

Consistency with above legal document impose many, difficult to meet, conditions, which implementation may continue over even several years.

The possibility of using underground lignite gasification in Poland meets with big public expectations. It mostly results from the belief that this method can totally replace open pit mining, widely used now. Disadvantage of the latter, in public opinion, is the degradation of environment. Most often it is caused by shortage of knowledge, among local community, about proper procedure of carrying out mining activity and about its effects. There is also lack of common knowledge concerning the legal requirements with regard of planning and monitoring such projects. Before launching any project the vide information campaign should be carried out among local population about safety issues concerning social and environmental aspects of the investment.

Under the shortage of information the method of underground gasification is regarded as equal alternative for open pit mining. However, it is not always reasoned.

Poland has quite big reserves of lignite. Thus all available and technically processed methods of its extraction should be used. It is also possible to use the PZW method as supplementary with relation to open pit mining on the same deposit or its part. Anyway the correctly elaborated technical design is necessary.

The method of lignite deposit evaluation if it qualifies for gasification, presented here, is only a preliminary one. Different engineering measures are possible to improve the gasification conditions resulting from geological and hydrogeological conditions. However, additional engineering methods may be expensive thus the final decision about applying the gasification depends on the cost-effectiveness of the project. The latter will change in the course of time depending on costs, prices, technical progress etc. Each project requires complex economic evaluation in order to identify possible to obtain profits offered by this method.

In case of PZW installation the biggest impact on costs have drillings made to build the exploitation (air blow and syngas extraction) and drainage wells. Therefore the most productive is gasification of lignite beds having great thickness.

Therefore economic, technical, legal and social factors decide about PZW method implementation.

6 CONCLUSIONS

Popularity of underground lignite gasification method consistently increases. It is a result of Clean Lignite Technologies development. Probably in Poland it will be implemented within several years, what is a consequence of its development in many countries. Operation of gasification systems for example in China (and other countries) in the commercial scale, gives a chance for constructing, in a short time, such installations also in Poland.

Implementation of this method requires meeting many legal conditions concerning mining, power energy and building operations. Extremely high expectations concern the environmental issues. Polish environmental standards are currently compatible wit European Union ones. The EU also controls observing them.

Mining activity subject the special requirements. Basing on to date experience concerning the preparation of underground gasification it may be concluded that:

1. Construction of commercial installation in Poland is possible within 6-8 years.

2. The conditions for PZW installation implementation are as follows:

– proceeding formal and legal action in cooperation with relevant authorities and elaboration of necessary regulations,

– finding the social acceptance for new method,

– training the technical staff for handling the gasification system,

– carrying our reasonable strategy of lignite deposit development (including their protection) with regard of environmental protection policy.

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Control and power supply of highly efficient mining plough systems

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ABSTRACT: The paper presents experience gained during development, implementation and commissioning of high performance plough units rated to new generation of machines as well as outlines specification of the power supply and control systems for coal plough units.

1 INTRODUCTION

Detailed analyses of the balance on the market of power and fuels demonstrate that demand for coal, gas and coking grades shall be gradually increasing (Tor 2008). Even under circumstances of the current economic crisis one can clearly see that reserves of crude oil and natural gas are still less and less whilst the hard coal still remains quite cheap and reliable energy carrier that guarantees safety of power supplies to Poland and improves fuel balance in other countries of the European Union. However, the indispensable provision that the verified and balanced reserves of coal are used in the appropriate manner is the wise and efficient management of the possessed coal deposits. Reasonable means that the management is aimed at limitation of coal losses to the minimum amount, which is achieved by extraction of thin seams, whilst efficient mining technologies are understood as those that are modern, safe, highly productive and reduce involvement of human personnel in mining operations (Tor 2008).

Such definition of trends and demands to the mining technologies encourages application of mining plough units due to their obvious advantages (Myszkowski & Paschedag 2008):

a) possibility to effectively extract thin coal seams;

b) less disintegration of the virgin coal and, consequently, possibility to extract coarser coal grades;

c) possibility to extract with shallow webs with quick advance or power roof support in areas with the hazard of brittle roof;

d) simplified and more lightweight design of the mining equipment;

e) improvement of occupational comfort and safety owing to automation of the extraction process.

Automation and control facilities are the most important components of the highly efficient plough technology that is essentially based on electrohydraulic control of the powered roof support (Myszkowski & Paschedag 2008). Operation of the plough-based set of equipment is carried out from the dedicated control panel that is placed either in the end tail gate or on the mine surface – no personnel is needed on spot to directly operate the plough. Operators at the control panel are on-line informed about essential operation parameters associated with the equipment status and are able to make corrections of these parameters (Stopa 2008). The plough-based technology for extraction of thin seams is successfully applied in Germany, Russia, China, the USA, Czech Republic and Kazakhstan (Materiały reklamowe).

Over the years 2006 to 2008 the plough-based technology was thoroughly examined with the aim of its application to extraction of newly developed seams at the "Zofiówka" Colliery of the Jastrzębska Coal Company plc. as well as Lublin Coal "Bogdanka" plc. In case of the both mines the analysis has led to the decision to launch coal extraction with use of high-performance plough systems.

Innovative solutions of explosion-proof and intrinsically safe design developed by Elgór+Hansen and intended to supply extraction machinery with electric power have proved their applicability under tough conditions of underground excavations. In addition, the acquired operation experience with simultaneous tracing the newest research and scientific developments have enabled company to offer, within a relative short time period, dedicated solutions that not only guarantee high level of safety but also meet requirements of applicability of the delivered equipment for the high-performance plough units.

The attention must be paid that the innovative approach covers also automation of the machinery operation with computer control as well as connection of computers by means of IT networks in order to achieve full-speed and comprehensive data exchange. The degree of technical advance of computer networks in the mining industry, availability of access to networks and popularity of such solutions are on a very high level, both on the side of technological processes and the equipment that is intended for automatic control of the same. That is why manufacturers, designers, users and operators of modern equipment have to cope with really demanding challenges.

2 POWER SUPPLYING AND CONTROL EQUIPMENT FOR THE HIGH-PERFORMANCE PLOUGH UNIT

Technological solutions that were implemented in high-performance plough units enforced application of medium voltage to supply machinery incorporated into the unit. Elevated supplying voltage not only improves reliability of the power supplying means to the machinery but is also conducive to enhancing of occupational safety and demonstrates a series of additional advantages (Morawiec 2009):

1. Makes it possible to deploy a part of electric power supplying equipment, in particular portable flameproof transformer station, outside the longwall length or within a substantial distance from the same, which allows to decrease:

a) ambient temperature in excavations;

b) extraction expenses due to reduction of the amount of electric equipment.

2. Eliminates problems associated with installation of more powerful drives in the machinery of the unit,

3. Allows to use hose cables with smaller diameters.

As early as in 1999 Elgór+Hansen focused its attention on designing and implementation of equipment for the voltage of 3.3 kV. Approving assessments, gained experience associated with operation of flameproof contactor switches, thyristor starters and transformer station are the reason for continuous perfection of products offered by the company with particular attention paid to improvement of operational safety and selection of the best available and the most advanced subassemblies that make up together really high-performance longwall extraction units as well as plough-based units, ready for challenges of 21st century.

2.1 *Transformer stations of the explosion-proof design*

Transformer stations offered by Elgór+Hansen for to supply plough-bases sets of equipment meet really high technological demands, guarantee operational reliability and definitely improve occupational comfort and safety of maintenance servicemen in mines. Power supplying systems of plough units are supplied from transformer stations encapsulated into d31 casings, so called “large” ones that can house stations equipped with transformer with the power of 1750 kVA, 2100 kVA and 2600 kVA where the transmission ratio is 6/3.3 kV.

All the aforementioned transformer stations are of the explosion proof and dry design, with the resin insulation of the rowing type and dedicated for purposes of the mining industry (Rozporządzenie Ministra Gospodarki 2003). Individual segments of the stations are mechanically connected with use of flameproof bolted flanges which not only assures bodily integration but also assured correct and safe operation of the entire station.

The upper voltage chamber (GN) of stations in the d31 enclosures are equipped with protections for the power transformer and a compact connecting switch with the insulation SF_6 400A, which substantially improves ability to protect protection the power supplying circuits in a reliably and efficient way. The GN chamber is furnished with a fast earthing switch that has been introduced as a mandatory standard for all the range of the offered transformer stations. It is the chamber that serves simultaneously as the medium voltage distribution bay and decides about supply capacities as well as reliable and correct operation of the entire transformer station.

Chambers of low voltage (DN) of the transformer station involve innovative solutions that enable transmission of the entire power offered by the station from the fully protected outlets to the common connection for the switching appliances – parallel supplying via two, three or four cables and or hose cables at the outlets.

The applied protection means of electric power functionalities as well the adopted standard for control and collaboration with technological units benefit from application of explosion proof (intrinsically safe) electric circuits, which makes it possible to supply the equipment that is operated in potentially explosive areas. The transformer stations are equipped with the explosion proof (intrinsically safe) system that is meant for safe repositioning (ON/OFF) of the main switch on the primary (GN) side. All transformer stations are also furnished with a fast earthing switch installed on the secondary (DN) voltage that guarantees extremely high level of operation safety for the personnel (Rozporządzenie Ministra Gospodarki 2003). The station can be relocated in underground areas of mines by means of the wheeled undercarriage with use of fixed transportation holders.

Figure 2.1 presents a typical schematic (one-wire) diagram of a transformer station on the example of the unit EH-d31-2600/6,0/3,3/02.01 with the transformation ratio 6/3.3 kV.

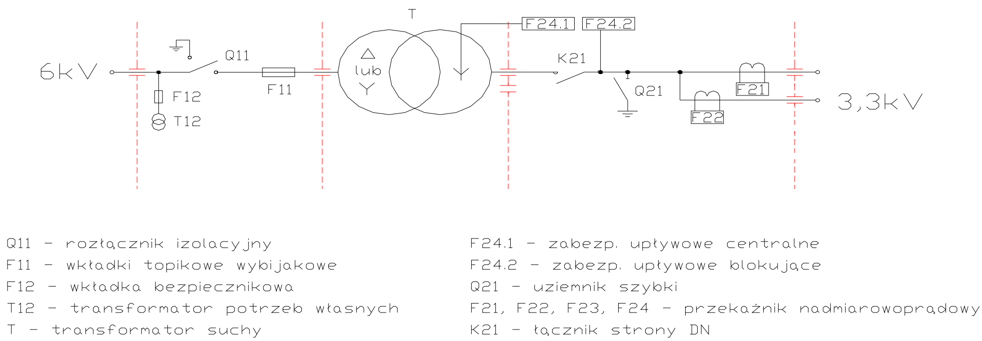

Figure 2.1. Schematic (one-wire) diagram of the transformer station type EH-d31-2600/6,0/3,3/02.01.

2.2 *Switching appliances of explosion proof (intrinsically safe) designed intended to handle individual appliances of the plough unit*

Specific examples of the appliances that are used for high-performance plough units are the following devices that have been recently introduced to the commercial offer of E+H: a flameproof contactor switch with four outlets, a switch with a thyristor starter as well as a transformer with power of 160 kVA and transformation ratios 3.3 kV / 1 kV and 3.3 kV / 0.5 kV (Morawiec 2009).

The flameproof contactor switch, type EH-d03-W/3.3/I/01.xx or type EH-d03-W/3.3/I/02.xx (ELGÓR+ HANSEN Sp. z o. o.) as well as the switch type EH-d03-WR/3.3/I/01.01 (Elgór + Hansen Sp. z o. o.) with a thyrisor starter represent appliances that make difference as compared to the range of switching equipment for the voltage of 3.3 kV that has been offered to date. That new equipment incorporates circuit breakers with fast earthing switches capable to earth wires at the power supplying connector as well as additional protecting switches (circuit breakers) that make it possible to install the appliances everywhere, regardless to the short-circuit hazard of the power supplying network in underground areas of mines. The switching equipment is equipped with a system of mechanical and electric interlocks that also substantially improves occupational safety of maintenance staff employed for mining enterprises (Rozporządzenie Ministra Gospodarki 2002).

The installed switching appliances have also fast earthing switch at every output, which is the solution that sets up new standards in the area of safety. To supply the entire set of the integrated longwall machinery with the voltage of 3.3 kV with no need to deliver other supplying voltages, different from 3.3 kV, to the extraction area the range of the dedicated electric equipment can be supplemented with the flameproof transformer unit of the type EH-d03-160/3.3/0.5(or 1.0)/0.2/4/01. The unit was developed in 2006 and offers the output power of 160 kVA and transformation ratio 3.3 kV / 525 kV or 3.3 kV / 1050 V. Apart from four three-phase outlets for 500 V or 1000 voltages it is provided with two fully protected outlets with the voltage 230 V and output load of 500 VA as well as one fully protected outlet for the voltage of 24 V and output load of 500 VA. This unit, similarly to the new generation of switches for 3.3 kV, is furnished with a disconnecting switch and an earthing switch, the both installed on the power supply (primary) side Figure 2.2 (a) and 2.2 (b) present the schematic diagram for the new generation of switching devices for the voltage of 3.3 kV as well as the schematic diagram of the transformer unit and the switch with a thyristor starter. In turn, Figure 2.2 (c) presents the schematic diagram of the power supply system for the entire set of the plough-based extracting machinery operated in the Hard Coal Mine "Zofiówka", whilst Figure 2.2 (d) comprises the schematic diagram for a similar power supply system for the extraction set but deployed at the Lublin Coal "Bogdanka" plc.

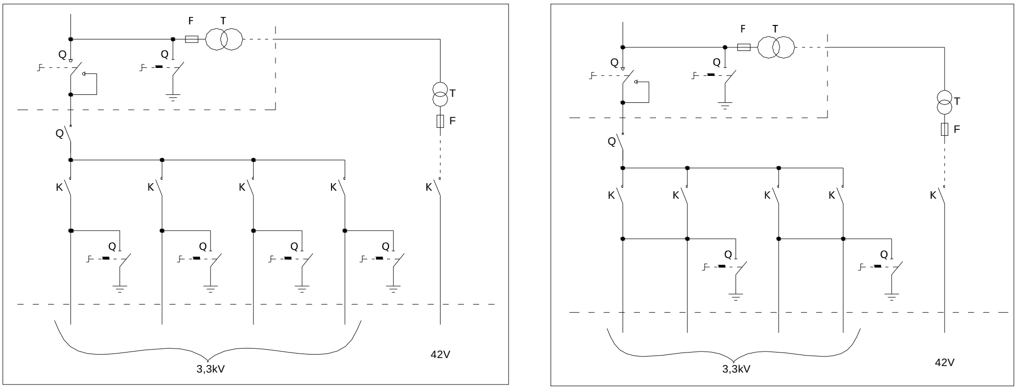

Figure 2.2 (a). The schematic diagram of the four-outlet contactor switch of the type EH-W/3,3/I/01.xx and the schematic diagram of the contractor switch type EH-d03-W/3.3/I/02.xx intended for reversible supply of main motors intended to drive the coal plough.

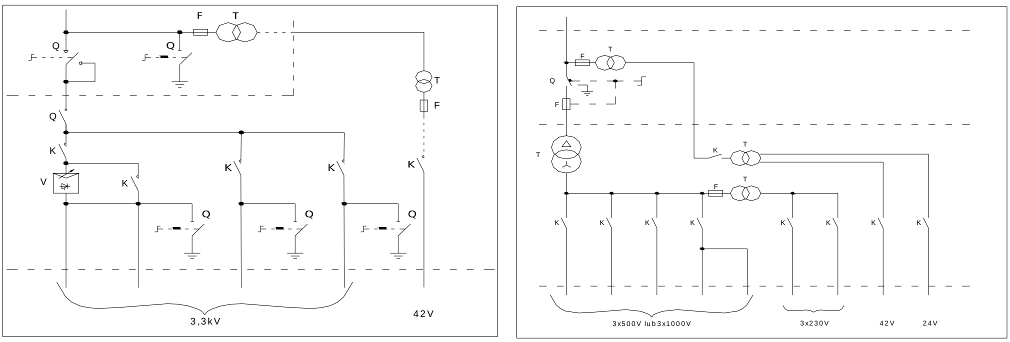

Figure 2.2 (b). The schematic diagram of the contactor switch type EH-d03-WR/3,3/I/01.01 furnished with a thyristor starter as well as the schematic diagram of the transformer unit, type EH-d03-160/3.3/0.5(lub1.0)/0.2/4/01.

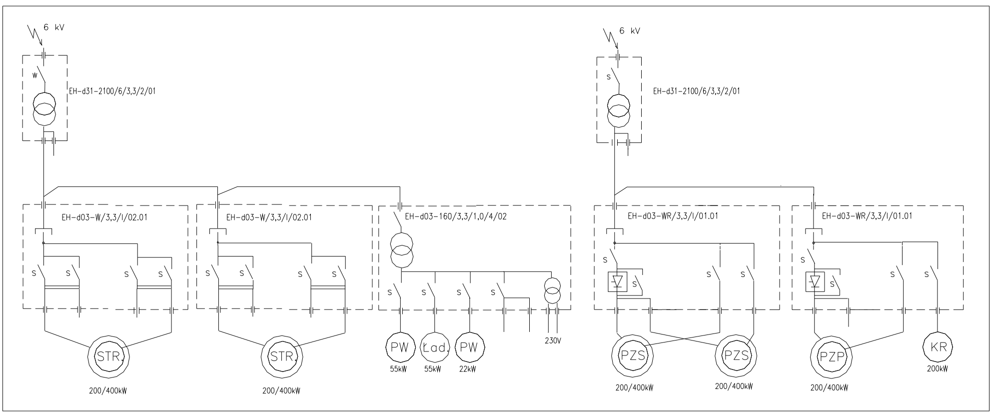

Figure 2.2 (c). The schematic diagram of the power supply system for the high-performance longwall set of equipment operated in the Hard Coal Mine “Zofiówka”.

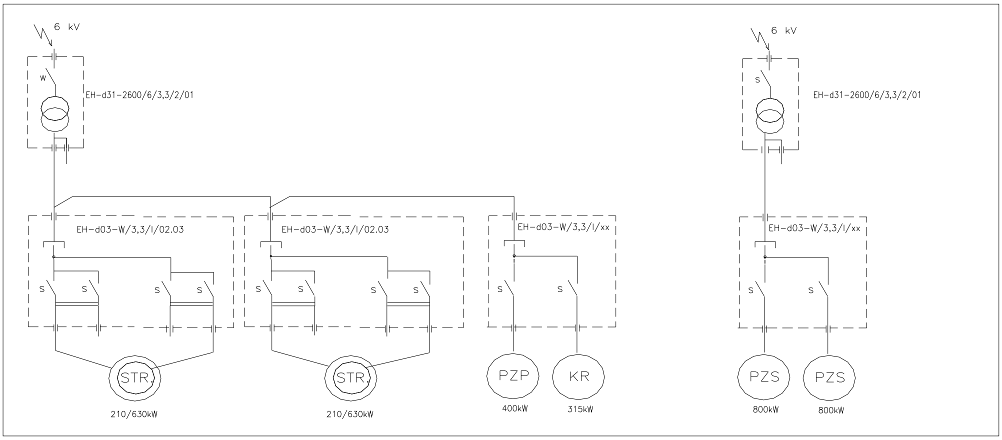

Figure 2.2 (d). The schematic diagram of the power supply system for the high-performance longwall set of equipment operated at the Lublin Coal "Bogdanka" plc.

2.3 *Control system for the plough-based extraction set*

The control system for the machinery incorporated into the high-performance plough-based extraction set represents an advanced and highly sophisticated microprocessor system that is meant to carry out the following operations (Czechowski 2008):

a) direct control,
b) visualization,
c) keeping logs of events,
d) keeping historical records,
e) diagnostic operations.

Deployment of the extraction system components on a large area entails application of a field structure of the control system with use of industrial IT networks as data transmission carriers. The plough-based extraction system is made up of a number of subsystems that are responsible for individual tasks associated with control and visualization, namely:

a) control system for the electric equipment delivered by Elgór+Hansen;
b) local visualization system for the electric equipment delivered by Elgór+Hansen;
c) central visualization system for the electric equipment delivered by Elgór+Hansen;
d) visualization system for the plough position and for diagnostics of the driving gear PMC-D from Bucyrus;
e) control system for the hydraulic jacks (powered roof support) PMC-R from Bucyrus;
f) local visualization system PMC-V from Bucyrus;
g) system of transmission modems PMC-LD from Bucyrus;
h) intrinsically safe system for loud-speaking communication, signal transmission and interlocks, for this purpose the UGS-01/2 from Elekrometal was used at the Hard Coal Mine "Zofiówka".

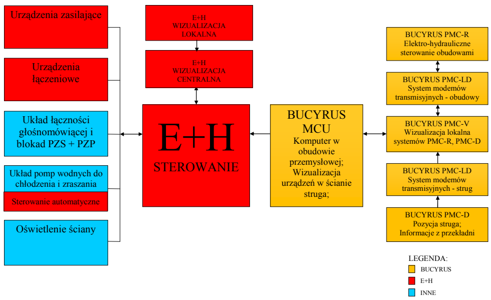

Figure 2.3 (a). The mimic diagram of individual subsystems within the high-performance longwall extraction set.

Information processing within the system is carried out in accordance with the hierarchic structure, starting from the level of control subsystems for individual components, through parameterization of the equipment, diagnostic procedures, enrolling current events to the system logs, visualization, keeping local and central records up to the level of central management for the entire technological process.

By its principle the system analyses a number of relationships and attempts to the maximum possible number of cases, situations and states of both the machinery themselves and the heavy-duty surrounding environment (Czechowski 2008). By the logic structure the control system is subdivided into two mutually interconnected parts: control and visualization.

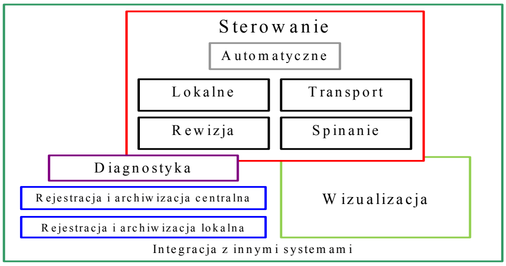

Figure 2.3 (b). Tasks to be carried out by the control system for the plough-based set of extraction machinery.

2.3.1 *Control system for the electric equipment delivered by Elgór+Hansen*

The system has been designed as a set of general-purpose functional appliances that mutually communicate with each other via a serial data bus:

a) the control unit of the type EH-O/06/02 (programmable logic controller PLC);

b) the control unit of the type EH-O/06/03 (PC computer of the industrial design /IPC).

The system follows the rule of the field control. Individual controllers (with purposefully adjusted configuration) have been placed at the control panel of the plough operator, at the drive of the push-plate armoured face conveyor (AFC), as well as at the main drive of the plough and inside the compact stations. The designers have put stress onto operation and control simplicity as well as easiness of possible extensions and alterations that can be implemented in future.

One of the major functions that is to be carried out by this control subsystem is to assure operation reliability and safety embedded into the control algorithm for the technological process. It is why the system was provided with an independent analog hardware system for the emergency stop function. Selection of subassemblies for assembly of the control system was performed with the prevailing criterion of reliability, surety that the operation parameters shall be continuously maintained with simultaneous high degree of integration. As much as possible, the application software for individual components performs also the testing functions, including both self-tests and dedicated tests for subunits.

Individual switching devices, including switches for movement signals and switches that incorporate thyristor-based starters have been connected to a common industrial network that is operated under the master-slave principle, where the diagnostic information is passed through the central controller and is then visualized and stored by means of the central visualization system developed by Elgór+Hansen.

All the devices and modules that are embedded therein are furnished with the built-in firmware intended to perform basic functions of monitoring and control for individual, componential communication processes to resources of the PLC and the system, with issuing addresses for each individual modules as well as to execute communication services to enable remote diagnostics and test operations.

The system communication between separate units and the controller employs the system bus to the standard RS485 and the MODBUS RTU protocol. In turn, communication between controllers is also based on the bus to the standard RS485 along with the PROFIBUS DP protocol. On the level of a singe unit the device controllers serve as masters while all the functional modules are considered as slaves and are identified by their types and the preliminarily assigned addresses. The structure of connections within the network of controllers is based on a similar solution that is made up of one master (supervising) station and a number of slave (subordinated) units. The system has been designed for the transmission rate of 9.6 and 19.2 kBits / s.

The next stage for development of the control system for the coal plough consists of accurate definition of the control algorithm. The algorithm takes account for correlations between all the input and output signals, e.g. status of sensors received from the EH-O/06/02 devices, information from the Bucyrus control system, status of actuators incorporated into the switching devices, etc., These signals have served as a basis to develop the application software for the central controller and for local control units. A part of tasks that are to be performed by the control process have been assigned to individual local controllers where the information between controllers are exchanged with use of the network protocol.

2.3.1.1 *The programmable logic controller EH-O/06/02*

Owing to implementation of an industrial programmable controller, the control device of the type EH-O/06/02 enables execution of complex logic input and output functions within the control system where they are used. The device can receive signals form explosion proof (intrinsically safe) external sensors connected thereto as well as from external actuators (e.g. solenoid valves, signalling /warning/panic devices, separators, other systems, control circuits of flameproof connecting switches, systems for visualization and data transmission, etc.).

2.3.1.2 *Industrial PC, type EH-O/06/03*

Owing to implementation of an industrial PC, the control device of the type EH-O/06/03 enables execution of complex logic input and output functions within the control system as well as functions intended to supervise the entire control system that is made up of many subordinated (slave) subassemblies and is capable to implement various transmission interfaces and various transmission protocols.

The device can receive signals from explosion proof (intrinsically safe) external sensors connected thereto as well as from external actuators (e.g. solenoid valves, signaling /warning/panic devices, separators, other systems, control circuits of flameproof connecting switches, systems for visualization and data transmission, etc.).

2.3.1.3 *Industrial PC, type EH-O/06/04*

Owing to implementation of an industrial PC along with a full-screen display, the control device of the type EH-O/06/04 enables execution of complex functions associated with visualization of the control processes as well as parameterization of the control functions for the entire control system. In addition, it is capable to implement various transmission interfaces and various transmission protocols.

2.3.1.4 *Digital control panels EH-O/01/05 and EH-O/01/06*

The control panels of the types EH-O/01/05 and EH-O/01/06 present digital electric devices of explosion-proof (intrinsically safe) design and they are used to control circuits of the mining machinery. The panels collaborate with explosion-proof circuits of the ia category, but operation of the control panels with circuits of the ia category is also allowed. The devices have been designed in accordance with requirements of the EU Directive 94/9/EU (ATEX). Parameters of the explosion-proof design and intrinsically safe features make it possible to classify the control panel to the group I and category M2, therefore the device can be used in areas with potentially explosive atmospheres.

The EH-O/01/05 control panel has the modular design, i.e. types and number of control instruments and indicating devices are adjusted in accordance with the specific requirements of the control system where the panel is to be applied. To the terminal strip one can connect rectifying diodes or other terminating components. The panel communicates with the supervising (master) unit by means of an intrinsically safe interface for data transmission, compatible with the RS485 and CAN standards. The panel is capable to implement various data transmission protocols, such as Modus RTU, Profibus, CAN 2.0B and CANOpen.

The control panel is meant to switch on and off any specific electric circuit within the control system of the dedicated mining machine and to provide information with use of LED indicators whether the desired operation status of electric circuits has been achieved or not.

2.3.2 *Description of the logic workflow associated with operation of the plough-based set of extraction machinery and executed by the control system from Elgór+Hansen*

The following operation modes have been prescribed and then implemented within the control system dedicated for the plough-based set of extraction machinery:

– automatic mode;
– local control mode;
– inspection /tensioning;
– transportation.

The extraction set can be controlled from the control panel of the underground control station of the plough operator, from local control panels deployed at specific drives or from the control room on the mine surface. The system described in Section 2.3.2 serves merely as an example and can be modified in accordance to requirements of various customers.

2.3.2.1 *Automatic mode*

When the automatic mode of operation is selected the set of extracting machinery can be activated from the control room of the plough unit located on the mine surface. After depressing the button “Załącz Automat” (“Automatic On”) individual pieces of equipment are subsequently switched on in accordance to the start-up sequence and with prescribed time intervals between activation of individual devices. The pumping unit is switched on the first, then the crusher, the push-plate haulage conveyor and the push-plate armoured face conveyor. Setting of each individual device in motion is preceded with an acoustic warning signal as well as with activation of the motor cooling system.

The plough itself is started up with use of separate push buttons. The mandatory condition to do this is preliminary actuation of the AFC (armoured face conveyor). Prior to setting the coal plough in motion an appropriate warning signal is produced. The initial direction of the plough operation is defined by the plough operator.

In the automatic operation mode the plough reaches end of the longwall, then stops and automatically reverses its motion towards the opposite direction after a predefined time interval that is defined as a system parameter.

2.3.2.2 *Local control mode*

In the local control mode individual pieces of the extraction equipment are switched on by means of local control panels deployed at each drive. Starting sequence and hierarchy of individual devices is predefined by the extraction technology and must be carried out in the same manner as in case of the automatic mode, i.e. pumping unit → crusher → haulage conveyor → AFC → coal plough.

The pumping unit, the crusher and the haulage conveyor are started up from the control panel that is placed nearby the extraction equipment in the end tail gate at the drives.

The AFC and the coal plough are activated from the control panel on the main drive. In this case it is possible to use a router to hand over control of the plough to the operator's control station and the operator is capable to see full visualization of the plough position within the longwall.

Setting of each individual device in motion is preceded with an acoustic warning signal as well as with activation of the motor cooling system.

When the local control mode is activated, the plough reaches the longwall end and stops. Its restart towards the opposite direction is possible after the operator depresses one of the switching pushbuttons.

Even if the plough is started up with use of the 'high speed' pushbutton the machine starts initially at low speed and then, after a time period that is predefined by the operator within the system parameters, the high speed operation mode can be activated. When the plough approaches its driving unit at high speed the machine switches over to the low speed mode within the already predefined distance from the driving unit. However, the operator can decide to cut the entire longwall at low speed and start the machine up with use of the 'low speed' pushbutton. If so, the control system shall abstain from switching the plough to the 'high speed' operation mode.

2.3.2.3 *Inspection /tensioning*

The inspection and tensioning mode enables individual operation of each drive within the set of extraction equipment. However, the mandatory provision to start up the crusher, conveyors or the plough is to keep the pumping unit running, therefore it is the subassembly that is switched on with the positive feedback to maintain its continuous operation, in contrary to all the other drives.

Operation of the equipment for inspection or tensioning is activated form local control panels of individual devices. During inspection /tensioning of the armoured face conveyor it is disabled to switch the plough and only one motor can be set in motion at a time. Inspection /tensioning of the plough chain disables actuation of the AFC. Also in case of the crusher inspection it is impossible to switch on the haulage conveyor and *vice versa* – inspection /tensioning of the haulage conveyor prevents the crusher from being switched on. Activation of any piece of equipment is preceded with an acoustic warning signal.

2.3.2.4 *Transportation*

The transportation mode enables to set in motion solely the conveyors. However, the mandatory provision to start up the conveyors is to keep the pumping unit running, therefore it is the subassembly that is switched on with the positive feedback to maintain its continuous operation. The possibility to activate the crusher of the plough is disabled in this mode. Setting the conveyors in motion is preceded by a warning signal.

2.3.2.5 *Stop and shutdown of individual devices within the set of extraction machinery*

Individual components of the extraction set can be switched off in the following cases:

a) depressing the "OFF" pushbutton for individual devices depending on the operation mode and with respect to the shutdown sequence of mutually dependent devices;

b) depressing the 'emergency stop' pushbutton;

c) triggering of appropriate sensors depending on the operation mode and importance of each sensor for the equipment, d) tripping of protections on the electric power lines;

e) switching of the downstream device that is preceding in the start-up sequence;

f) failure of data transmission,

g) lack of control signals form the contacts of the system intended to control powered roof support units, both in the local and automatic control modes;

h) no acknowledge of the switchover completion from the activated contractor.

2.3.3 Description of the visualization system provided by Elgór+Hansen and intended to trace operation of the plough-based set of extraction machinery

The system intended to visualize operation of the electric equipment for the plough-based set of extraction machinery was subdivided into:

a) local visualization system;

b) central visualization system.

The local visualization system, by its nature, is dedicated exclusively to a single unit, e.g. a compact transformer station or a transformer unit. The scope of information that was subjected for visualization has been limited to one operator and appropriately structured to make the local diagnostics easier for the operator. The microprocessor system that is responsible for local visualization procedures may be differentiated depending on which device operation is to be visualized. For instance, local visualization for switches and starters of the plough-based extraction machinery is carried out by means of the KSK-1 appliance from Somar while a LED display EH-P05/03is used for the transformer unit. As for some other appliances, such as control units EH-O/06/02 or EH-O/06/03 the procedures of local visualization indicate status of individual devices of the equipment by means of dedicated LEDs. In addition, the visualization system based on the KSK-1 device enables to keep records on the machine operation on a local data carrier. An example of a local visualization screen is shown below.

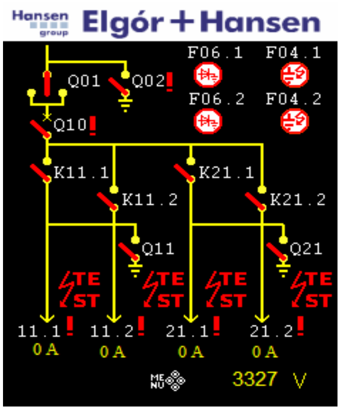

Figure 2.3.3 (a). The example of a main screen for local visualization.

The central visualization is based on the device type EH-O/06/04 that is made up of a computer along with a full-screen display and an intrinsically safe keyboard.

The central visualization covers all the devices that are included into the plough-based set of extraction machinery and carries out the following major functions:

– visualization of the control system status;

– visualization of equipment incorporated into the train of the electric equipment;

– visualization of devices incorporated into the control system;

– parameterization of the control system;

– central data storage;

– viewing of historical information for the purposes of maintenance analyses.

Figure 2.3.3 (b). The main screen for central visualization.

3 CONCLUSIONS

Comprehensive and thorough analyses for efficiency of coal extraction from thin coal seams are currently being carried out in many countries. The experience acquired for extraction from low longwalls with use of coal ploughs provides the proof that the plough technology is a reasonable method for extraction of such seams due to technical reasons. In addition, application of the plough technology is backed by essential economic arguments that confirm the technology as the most beneficial for extraction of coal seams with their thickness up to 1.5 m (Stopa 2008). The Polish mining industry has to deal with and will continue to extract still more and more seams of such a thickness, therefore number and technical level of plough-based extraction sets shall be continuously improved. It is why permanent perfection of power supply and control systems dedicated for such equipment is becoming a hot issue. Manufacturers are enforced to offer more advanced power supply and control systems purposefully dedicated for high-performance plough-based sets of extraction machinery to be up-to-date with the offer of the solutions that are capable to perform all the demanded technical tasks and meet the cost-effectiveness criterion. It must be emphasized that the plough extraction technology presents new challenges that must be withstood by each component of the equipment set. Only meeting the challenges guarantees successful performance of mining enterprises which, in turn, is directly and proportionally transformed to economic healthiness of companies that are leading suppliers for the mining industry. The new challenges inflict a great leap forward in terms of technology but it is also a mental conversion that will definitely change scope of operation tasks for maintenance staff of coal mines. Consequently, amendments to the existing regulations and rules of mining operations are expected. In this context, standardization and modular design of the equipment present major directions for further development of the Elgór+Hansen company.

Since the very beginning stage, when the Polish mining industry only initiated the renaissance of the plough extraction technology, Elgór+Hansen has been actively participating in research and development jobs on the power supply and control equipment. It is why the company presents here the offer of broad and extensive collaboration in that area with all the existing and new partners as well as with all possible stakeholders.

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Elgór + Hansen Sp. z o. o.: Instrukcja Obsługi i Bezpiecznego Użytkowania wyłącznika z rozrusznikiem tyrystorowym typu EH-d03-WR/3,3/I/01.01. *Manual for Operation and Safe Use of the Switch type EH-d03-WR/3,3/I/01.01 with a thyristor starter*

Rozporządzenie Ministra Gospodarki z dnia 28.06.2002 roku w sprawie bhp, prowadzenia ruchu zakładu górniczego oraz specjalnych zabezpieczeń przeciwpożarowych podziemnych zakładów górniczych wraz z późniejszymi zmianami (Dz. U. nr 139. poz. 1169 z dnia 02.09.2002 roku wraz z późn.zm.) w zakresie zapewnienia bezpieczeństwa ich użytkowania w warunkach zagrożeń występujących w ruchu zakładów górniczych. (Ordinance of the Minister of Economy of 28th June 2002 on occupation health and safety, operation rules for mining enterprises and special fire protections for underground mining enterprises, with further amendments, Journal of Laws, No 139, pos. 1169 of 2nd September 2002 with further amendments) with regard to assurance of safety of their operation under conditions of hazards that occur during operation of mining enterprises).

Rozporządzenie Ministra Gospodarki, Pracy i Polityki Społecznej z dnia 28 lipca 2003 r. w sprawie zasadniczych wymagań dla urządzeń i systemów ochronnych przeznaczonych do użytku w przestrzeniach zagrożonych wybuchem (dyrektywa 94/9EWG–ATEX) w zakresie grupy I – urządzeń przeznaczonych do użytku w zakładach górniczych, w których występuje zagrożenie metanowe lub zagrożenie wybuchem pyłu węglowego. (Ordinance of the Minister of Economy, Labour and Social Policy of 28th July 2003 on essential requirements to protective equipment and systems intended for operation in potentially explosive areas (Directive 94/9/EEC – ATEX) for the equipment of the 1st group – appliances dedicated for use in mining enterprises with the hazard of methane inflow or the hazard of coal dust explosion).

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New Techniques and Technologies in Mining – Bondarenko, Kovalevs'ka & Dychkovs'kyy (eds)

Possibility of production complex of sufficient gasses in Ukraine

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ABSTRACT: The questions of complex technologies use of energy production are considered. Technological schemes of generator gas production oriented on biogas creation from different sources including stations of underground coal gasification are offered. The different methods of coal gasification are presented. Such developments allow at high-quality new level to product power and chemical raw material and provide efficiency functioning of all of the system.

1 INTRODUCTION

The most important direction of increase the functioning power and heat efficiency in Ukraine is application the combined (compatible) production on one gas-productivity enterprise which will be used the gases of underground and surface gasification and also biogas.

Only at new building of such enterprises the economy of traditional fuel at the combined production arrives at 25-30%, and annual charges diminish on 20-26%. Thus approximately contamination of environment diminishes on 30-45%.

Separate technologies of gas production by underground gasification, surface gasification and biogas are created and used in the industrially developed countries, such China, Australia, Japan, South Africa.

Nowadays the use of restoration energy sources in Ukraine is on the initial stage. Their particle make about 0.5% power potential. At the same time biomass power potential is estimated in more than 7.5 mln.t. / year accumulates in the agro-industrial complex of Ukraine.

Counts show that biomass power potential would satisfy to a 10% general consumption energies in Ukraine.

Scientific experience allows forecasting, that the use of biomass as raw material for the gases production is the perspective additional of fuel source.

At processing gaseous fuels can be got mainly two types: biogas, the basic combustible component of which is methane, and generator gas.

UCG gases is especially attractive in ecological advantages, as during underground gasification an earthly surface is not almost violated and this gases is ecologically acceptable type of fuel.

2 UNDERGROUND COAL GASIFICATION

Underground coal gasification can be examined as one of examples of generator gas reception which carry out new principle – combination of coal extracting with its simultaneous processing in the unique technological process.

For today's day in the world there is the new interest to development of new mine and borehole underground coal gasification (BUCG) technology.

BUCG interest is conditioned that the prices on electric power, gas and oil grows constantly, that is why a price on gas from underground coal gasification in this situation will appear considerably below than price on natural gas. Accordingly power and thermal energy, produced from UCG gases, will be cheaper of analogical products, got after use of natural gas. UCG gases is especially attractive in ecological advantages, as during underground gasification an earthly surface is not almost violated and this gases is ecologically acceptable type of fuel.

The history of realization the underground coal gasification technology in Ukraine can be dividing on two stages. The first stage was begun from 1930 year, when two stations of «Pidzemgaz» worked in our country: Lisichanska and Gorlivska. The enormous reserves of natural gas and oil were discovered in 60th. On a background such enormous oil reserves which seemed inexhaustible in those years, underground coal gasification appeared unnecessary and not perspective. That's why stations of "Pidzemgaz" were closed.

The first power crisis in 1973 year stimulates interest of underground coal gasification. All country which have a large coal reserves in the world began intensively to conduct experimentally industrial experiments on perfection of underground coal gasification technology.

In the National mining university (Dnipropetrovsk, Ukraine), beginning from middle of 60th for a present time, developed at the level of inventions on principle new technology of borehole underground coal gasification with preparation underground gasgenerators from a surface and in mine terms (Kolokolov 1991).

Basically, an unworked seam of coal in the ground is accessed by two drilled wells, into one of which an oxidant is fed, the coal is ignited and part-combusted, and a product gas flows from the other well, as shown in Figure 1.

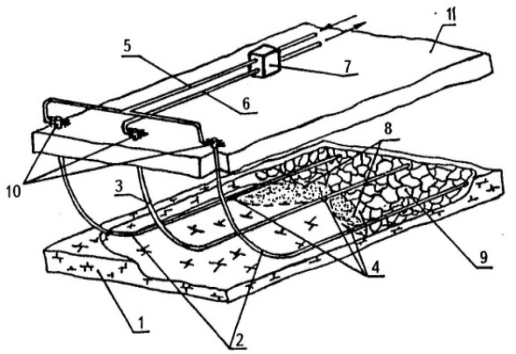

Figure 1. Technological scheme of underground coal gasification from a surface: 1 – coal seam; 2 – injection wells; 3 – production well; 4 – movable ignited devices; 5 – air pipeline; 6 – gas pipeline; 7 – heat – utilization; 8 – combustion face; 9 – goaf; 10 – adjutage of the directed boreholes; 11 – surface.

This technology can be realized thus. From a surface, to crossing with a coal seam 1, the directed (oriented) injection wells 2 and production well 3 are bored. Inclined parts of boreholes are fastened by pipes with cementation of annulus space. Horizontal parts of boreholes, bored on coal are not strengthened. Coal seam ignition is conducted by the special ignited cartridges which refer by air to the place of ignition.

As far as coal seam gasified the serve of blowing and gas discharge are carried out from mobile controlled retraction injection point. Coal gas which appeared is burned in the special free piston generator (FPG) setting.

Now in the Kharkov city are develop the original constructions of free piston generator, Figure 2.

This new modern highly productive setting, owns a high efficiency, small sizes, does not pass vibrations on foundation, works reliably with simple service. FPG can be set near by boreholes and to generate here electric power and heat power.

The necessity of building of gas pipelines falls off in this case, and energy (for example, electric) is passed to the users on send-offs.

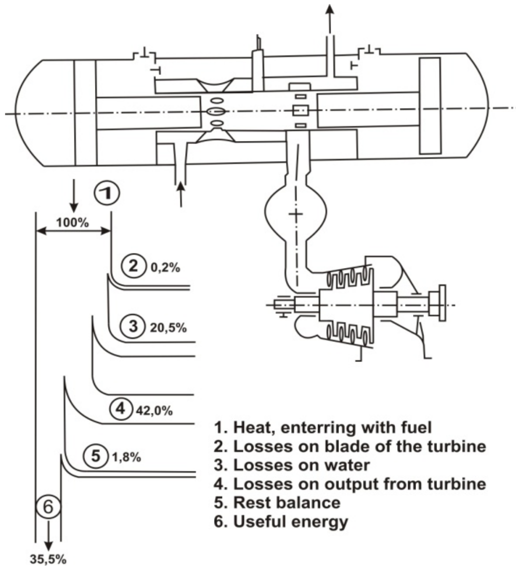

Figure 2. Scheme of equipment situate and power diagram of engine with free pistons.

For the increase of efficiency of the above-mentioned technology of BUCG developed by us row on principle new technological decisions which consist in the following.

1. The constructions of underground gasgenerators which provides coal seam gasification with the controlled retraction injection point and gas discharge are improved.

2. The criteria of BUCG technology fitness are extended with providing of technological schemes adaptation with gasgenerator preparation from surface or in a mine conditions at economic and uneconomic coal seams gasification in difficult mine conditions.

3. The process of coal seams gasification is conducted without burning out of combustible gases which appeared.

4. The germetization of deformed rocks and goaf due to the artificial injection stowing is provided.

5. For goaf germatisation we can use next material: coal and rocks dust, clay and ashslags. Thus escape of germatisation material from a gasgenera-

tor on a surface will not exceed 0.1 g / m^3. Insignificant part of germatisation material which darts out on a surface with the products of gasification will be caught in superficial purification options.

6. The coal gas calorific value and volume of gasification products grows due to burning out of carbon in the ashslags (on 20-30%), ash (25-35%) and coal-rock dust (30-60%). On the whole taking into account all technological decisions, a calorific value grows to 10-15 mJ / m^3, that in 4-6 times higher than on the former stations of "Pidzemgaz".

7. The technological schemes of clean coal technology are developed on traditional power enterprises. The ashslags, and extraction rocks from underground mine enterprise can be used for making stowing. After this we can see improve ecological situation.

8. Using direction drilling we can reduce capital costs. It allows sharply improve a landscape and fertile seam of surface in the district of underground coal gasification.

9. The technology of gases division is used for separate some components of gas for a serve them in a separate kind to the users and, in particular, back in an underground gasgenerator (for example, CO_2 for the receipt of combustible gas for the reactions $CO_2 + C = 2CO$).

10. Technology of BUCG foresees a serve in the underground gasgenerator of smoke wastes traditional fuel of power complex. In an underground gasgenerator smoke gases (CO_2, H_2O) are under act of high temperature (1000-1300 °C) at a contact with an incandescent carbon, pass to combustible gases (CO, H_2, O) on reactions:

$$2CO_2 \rightarrow CO + O_2 ;$$

$$CO_2 + C = 2CO + O ;$$

$$H_2O + C = CO + H_2 .$$

It is a new and original decision allows sharply to intensify the process of underground coal gasification; effectively to contest with a green house effect; from wastes to get profits at incineration of the additionally got combustible gases; returning of smoke gases in an underground gasgenerator improves an ecological situation in a region.

11. A technological block "underground gasgenerator – heat utilization complex" creates the incorporated structure of two productions in the reserved technological cycle with the use gases and thermal energy which get from physical heat utilizations of BUCG gases. The construction of the directed boreholes is changed for this purpose.

Thus one part of the overheated vapors follows again in an underground gasgenerator for the increase of his work efficiency, other – in a steam-turbine for the power production.

12. Building of such blocks will allow deciding the followings tasks:

– providing of reliability of electric power and heat supplying of adjoining settlements and industrial enterprises;

– organization of cost-effective production, electric power and heat (as compared to a low cost-effective mine boiler room and necessity of electric power purchase from a state grid);

– complex decision of ecological and social questions;

– creation of afore-mentioned technological complexes satisfy strategic directions of fuel and energy complex of Ukraine development.

13. Technology and technician of BUCG becomes universal as compared to traditional energy. It is related to that in converting actions (converting of hard coal into a gaseity) is never used mechanical mediator (for example, wallface combine); electric energy becomes a general mediator, and technology – without machine.

Certainly, it follows to establish, that presently technology of BUCG in Ukraine is not claimed. Our official engineering and scientific idea which is responsible for a technical process is so conservative and convinced, that subsequent development of energy – in the traditional mining by mine and opened methods.

However in the world looks to development of energy quite another. In Europe, for example underground and surface gasification of coal is used all wider and wider. Technical decisions are developed in the National mining university on untraditional development of hard combustible minerals is necessary for Europe. We accept more active participating in an international project, where involved 12 companies from Germany, Great Britain, Belgium, France, Poland and other countries. The experimentally industrial tests of underground gasification technology was conducted on experimental mine "Barbara" in Poland.

14. Underground coal gasification is a high-quality jump in development of productive forces, which allows blocking a raw material and ecological crisis.

3 SURFACE GASIFICATION

Modern technical potential of artificial gas production from coal presented by many industrially mastered technologies, proper names of firms, – producers of basic equipment of technological process – gasgenerators (Lurgi, Koppers – Totcek, Vinkler, Teksako). Last years there were 248 gasgenerators units of different type in exploitation, in particular in the south Africa republic – 92, the USA – 18, China – 6, to Czech and Slovakia – 62, Germany – 32 (Kolokolov 2000 & Kreinin 2006).

Superficial coal gasification, peat and biomass in the world got considerable development in 50th years of the twentieth century. In this period made more than milliard m^3 of power and technological gas (Kreinin 2006). In connection with stormy development of natural gas industry this direction of hard combustible minerals processing was constantly abbreviated.

In 1990 year considerable development was got by coal gasification for the electric power production, which used of binary cycle at which combustible gas was utilized in a gas turbine, and the products of combustion are used for the vapors generation for a steam-turbine. After 1993 year in different countries it was entered in exploitations 18 power-stations with superficial hard fuel gasification by power from 60 to 300 MWh. Presently the use of superficial hard fuel gasification is considered the most perspective direction in surface gasifcation. Growth of hydrocarbon fuels cost, which takes a place the last years, stipulates considerable interest in the whole world to the use of alternative energy sources.

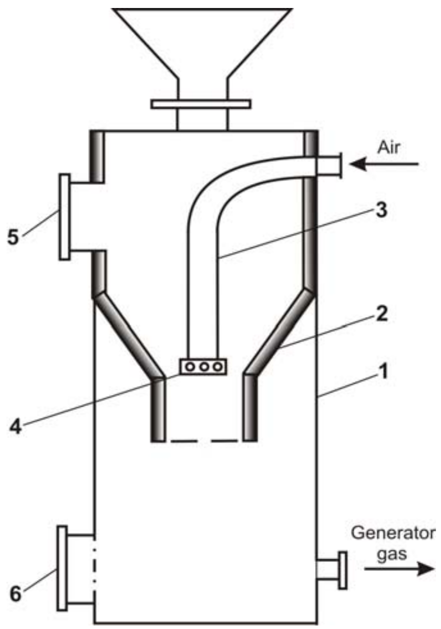

Figure 3. Gasgenerator on the generator gas production: 1 – metallic corps; 2 – cone; 3 – pipe; 4 – snuffled the air serve in the burning zone; 5, 6 – air-tight doors.

Ukraine owns the considerable peat reserves, coal and biomass which presently practically not utilized for the production of electric and heat power. The coal reserve in Ukraine makes 2166 million tons. Peat is obtained mainly used as a fuel in communally – domestic sector. The supplies of subbituminous coal are estimated in 3.5 milliard tons.

The basic criteria of the use of surface gasgenerators presently are their simplicity of making and exploitation, reliability and low cost. Such gasgenerators construction was developed, in the institute of technical physics "National Academy of Science of Ukraine".

A gasgenerator (Figure 3) consists of corps 1 with the internal diameter of 600 mm, corps 2 with grate, pipes 3 with nozzles 4 for the air blowing serve. Doors which are closed are located in overhead and lower parts of corps, 5 and 6 accordingly for the load of fuel and delete of ash. As a fuel waste-woods are used. A gasgenerator product gas with next composition CO – 21%, H_2 – 17%, CH_2 – 2%, N_2 – 48%, CO_2 – 12%. A temperature in the burning zone makes from 1000 to 10000 ^oC. An ash concentration in generator gas makes 2-3 g / m^3.

Total potential resources of wood wastes, including a cortex in forestry and woodworking industry makes 984 thousand tons in a year (Falshtyns'kyy 2009). Neither forestry nor woodworking and cellulose – paper industry cannot use all of wood wastes and can or to supply or use for an own energy supply.

In surface gasgenerators technology the petrol and diesel wastes utilization is widely used. Services enterprises and service centers, transport companies and build enterprises often throw out exhaust fuel, or pay a large money for it regeneration or utilization.

Heat generator, that work on exhaust fuel allow to burn it without an additional regeneration and cleaning. Transport charges are thus saved at an export on the place of regeneration, the risk of environment contamination really diminishes.

The general view of gasgenerator is shown on Figure 4.

Such fuels as peat, subbituminous coal and biomass are low-caloric – characterized the large volatile escape, that can substantially promote work efficiency of the offered new gasgenerator enterprises. The basic elements of gasgenerator is a steel cylinder combustion chamber, in which a plate is situated and filled by exhaust fuel. Air in a combustion chamber is given an axial ventilator tangentially with the crack ducting in an order to create an involutes gas stream above the surface which evaporating.

In the developed countries on the last few years technology of biomass gasification got development with the receipt of electric and thermal energy. Produced gas from gasgenerators burned in a gas engine or in a coal cauldron. The examples of realization the biomass gasification technology and wastes with subsequent burning of generator gas are power-stations in Australia, Belgium, Finland, Netherlands, Great Britain and other countries.

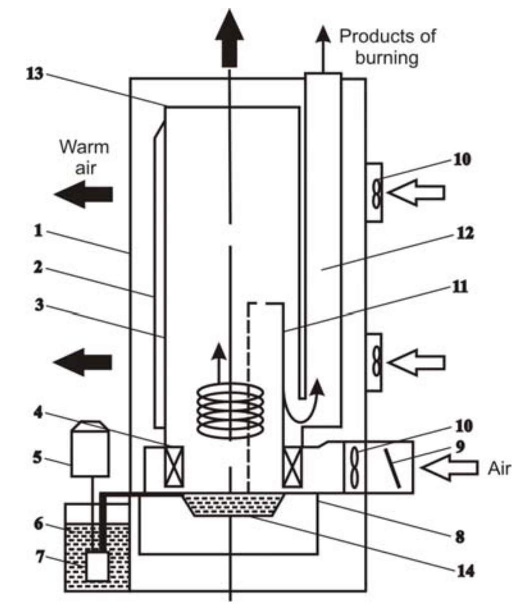

Figure 4. Gasgenerator which works on exhaust fuel: 1 – corps; 2 – pipe; 3 – combustion chamber; 4 – mixer; 5 – an electric motor; 6 – tank with fuel; 7 – pump; 8 – pallet; 9 – bolt; 10 – ventilator; 11 – screen; 12 – place of gas moving; 13 – lid; 14 – plate.

Therefore, the public policy of Ukraine on an energy supply predict substantial expansion of the use of untraditional energy sources of volumes. In Ukraine in 2010 year must be the economy of traditional energy resources on 8-10% from their general consumption. According to strategy of untraditional energy development in the EU countries “White Pape” 2010 year biomass will cover an about 74% general payment of refurbishable energy which will make an about 9% general consumption of primary power mediums sources (Pivnyak 2002). Obviously, biomass will be the most mighty sector of the use of refurbishable energy sources.

Thus, technology of biomass gasification with the purpose of making the cogenerating energy (electric, thermal) attained a high level and while does not bring in large payment in world energy.

Technologies of biomass gasification own considerable potential and prospects of development. They allow to carry out making of electricity and heat-energy with high-efficiency to 45%. In addition, they own the row of ecological advantages.

4 BIOGAS PRODUCTION

Nowadays the problem of quality rising and as result efficiency of biogas using gradually goes out on the foreground of development of the proper technology. The basic elements of the biogas setting are: chamber of зброджування (methane-tank), a device of stationary temperature support is in methane-tank, device which provides biomass interfusion in methane-tank, device of accumulation and biogas storage (gasholder).

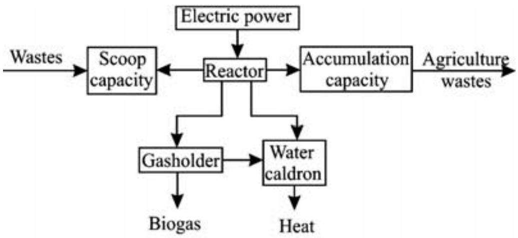

Figure 5. Scheme of setting for the biogas production.

As a result of anaerobic fermentation (bioconversions) appear: a biogas, which consists of methane (65-80%), carbon dioxide CO_2 (15-25%), carbon monoxide CO (2-3%), nitrogen (N_2), oxygen (O_2), sulphurhydrogen (H_2S), and high-efficiency environmentally clean disinfected fertilizer. One ton of it is equivalent 3-4 tons of nitric-phosphoric fertilizers which are produced industry. In the process of methane fermentation pathogenic microorganisms perish and the unpleasant smell of wastes is neutralized.

The method of biogas reception is widely widespread in a number of countries with a warm climate, a high enough temperature, conditioned natural terms, is needed in which for the effective flowing of bioconversion (India, Brazil, Chinese Folk Republic and other). Most active activity of bacteria at temperature 35-45 ^{o}C.

Fluidizers anaerobic fermentation serve as the effective mean of agricultural and stock-raising wastes processing (leaves, stems of plants, weeds, straw, sunflower wastes, corn heads, pus of different farms and dung of poultry factories), and also communal wastes on purification buildings and other. Their application allows to decide three tasks, important from the power, agricultural and ecological points of view, namely: to get a biogas; to convert wastes into high-showy profits (fertilizers); an environment impact.

In the conditions of Ukrainian continental climate (especially in cold times of year) continuous during throughout the year exploitation of bio-setting which require a stationary positive temperature (35-45 ^{o}C) is economic advantageous at heating of methane-tanks due to utilization of warmth of hot generator gas at borehole underground coal gasification. The got biogas which contains to a 70-80% methane allows at his addition to generator gas of underground gasification substantially to rise the heating value of this gas (Figure 6) essence of this technical decision consists in the following.

The area of coal seam 1 is opened by injected 2 and production 3 boreholes which on a coal seam connect between itself by hydraulic fracturing or filtration connection, forming the gasification channel which transforms after coal ignition on a combustion face 4. On a borehole 2 to the combustion face 4 the air blowing are given which here reacts with coal, forming generator gas which is produced on an earthly surface on a borehole 3.

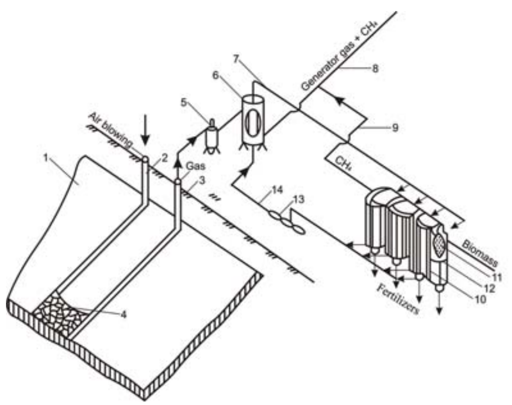

Figure 6. Technological scheme of biogas production with heating of reactors in cold season of year.

On an earthly surface generator gas follows to the purification setting 5, where it is purged from coal particles and phenols, after this skip through heat-exchange 6. In the last hot generator gas gives the warmth to water or air which is given in the capacity of heat-exchange 6. Cool gas in heat-exchange is sent to the main pipeline 8, and water or air pipeline 7 send to the biological reactors (methane-tanks) 10 in the heating cavity 11 between two reactor walls which surrounds the bioconversion reactor 12. Formed biogas in reactor 10 selected in it overhead part which carries out the role of gas-exchange, (gasholder), from where methane on the pipeline 9 send to the main pipeline of productive gas 8, where interfuses with generator gas of underground gasification. As a result of this mixing in eventual productive gas which sent to user, maintenance of methane which owns the most calorific value of combustion from combustible all of the tools is increased, that makes productive gas. The same substantially the general calorific value of combustion productive gas rises to 10-15 $\mathrm{Mj/m^3}$.Unnecessary heating conducter from an interwall cavity 11 methane-tanks 10 by a pump 13 (if heating conducter is water) or by a ventilator on a pipeline 14 again follows for the repeated use in heat-exchange 6, completing a technological rotation. Past in a biological reactor biomass which does not have a bad smell and disinfected as a result of fermentation process are unloaded and send to the user as a valuable and environmentally clean fertilizer as mash or, powder grainy mass, packed up in the proper container.

Thus, the station of underground coal gasification due to passing heat utilization of generator gas, provides support in a biological reactor permanent, in spite of seasonal changes, temperatures on a necessity levels. Thus the necessary terms of high-efficiency process of biomass fermentation and biogas receipt which is folded to 80% from methane are created. The station of “Pidzemgaz” becomes a clean complex of mine power enterprise. Such enterprise can be as the standard of highly remunerative industrial enterprise of XXI century.

5 CONCLUSIONS

The necessity of alternative energy sources research is dictated by the rising of natural gas, oil and coal costs. Due to the mining in deep mine in remote districts, and also by the worsened geological conditions and increase of distance of their transporting makes power costs unbelievable high.

BUCG can become an alternative to traditional energy, successfully working out technological and social problems, and can become the active ecological hospital attendant of our degrading environment as a result of negative influence of existent power complex.

Surface gasification is one of perspective technologies of energy reception from biomass. Interest to gasification technologies is more displaced from a production only of thermal energy to possibility of the cogeneration production of thermal and electric energy. Most attractive for the commercial use with the purpose of making of electric power there are presently technologies of atmospheric gasification in a circulatory boiling seam.

By the source of biomass as a power medium for the biogas production there can be wastes of plant-grower and stock-raising. Technology of biogas reception is exhaust enough in countries with a warm climate and cannot be realized in a continental climate (in Ukraine) without heating of reactor. Our technology of biogas production allows the joint (combining) production of BUCG gas and biogas at thermal support of temperature in methane-tanks (concrete or metallic) due to the utilization of thermal energy – BUCG gas has large prospects. In spite of obvious advantages and benefits of energy production from biomass, bio-power technologies develop in Ukraine very slowly.

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Researches of influence of depth of an in-seam working on displacement field of rocks in its vicinity

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ABSTRACT: Laws of displacements of stratified massif of weak rocks around in-seam working with increase of its depth are presented and degree of influence of strength and deformation characteristics of adjacent rock layers on these displacements is presented as well. Sufficient independence of stress field of a rock layer from condition of the adjacent layers is discovered: reduced strength layers with increase of H transform into limiting (out-of-limit) state virtually regardless of strength characteristics of adjacent harder rock layers; stress field in rock layers with higher strength characteristics almost does not depend on the adjacent layer transformation into limiting state; stress field in a coal seam, in the second layer of bottom and the third layer of roof, is quite stable compared to condition of adjacent rock layer.

1 INTRODUCTION

Modern methods of prognosis of rock pressure manifestation are being constantly improved towards more thorough consideration of features of a support interaction with a rock massif containing a working and its physical and mechanical characteristics. These studies are extremely important for the development of general concepts about interaction of "massif-support" system and surveys of optimal parameters of given interaction, taking into consideration final aim, which is the control of steadiness of the working using low-cost technologies. So, supporting process should be continuously connected with geomechanics of border zone rock massif.

During determination of tasks of geomechanics, it is constantly required to face a problem of calculation of systems that have complex geometrical configuration and irregular physical structure. Rock massif and its host rocks have a big number of characteristics, computation of which with help of mathematical modeling is possible only with use of finite-difference computational schemes. At present time, finite element method (FEM) has become factual standard during solution of geomechanical tasks.

Resulting from analysis of works dedicated to finite-difference modeling of geomechanical processes close to underground workings, following tendencies of given research trend is clearly observed:

First of all, finite-element method is the most widespread usage and it has the most universal capabilities. But there is a possibility to use combination of FEM and BEM (boundary-element method) in order to receive more reliable solution during determination of complex geomechanical tasks that simultaneously consider many factors of rocks mechanical characteristics, non-uniformity of massif structure and geometrical parameters of the object. Basic complexity of application of such combination lies in objective reflection of physical essence of the process at the border of abutment of the model elements that are being explored by various finite-difference methods.

Secondly, many tasks require 3-D formulation of a question for more reliable reflection of geomechanical processes, what is being observed on a modern stage of rock massif condition evaluation.

Thirdly, a bigger number of works is dedicated to calculation of non-uniformity of mechanical characteristics of massif and its structure that quite essentially influences on the results of calculation of the field of stress and strain state around the workings and this was determined during research of a wide sphere of tasks.

Fourthly, number of works in which not only plastic and rheological characteristics of rocks are considered but also a whole diagram of their deformation (limiting and out-of-limit states) is steadily growing, that, in authors opinion, increases reliability of calculations and advances geomechanical model to real object. Listed points have made up methodological basis of the performed investigations.

2 GEOMECHANICS OF SYSTEM "MASSIF-SUPPORT"

In the previously executed work (Bondarenko 2006) there were stated results dedicated to researches of stress and strain state of the system "stratified massif-support of a development working" with linear connection of stresses and strains within its elements.

At the first stage all basic factors that influence stress and strain state of the system were analyzed and the evaluation was given to them according to mining-geological and mining-technical conditions, for example, considered for mines of Western Donbass.

Geomechanical substantiation of model using a line of initial parameters and ranges of their variation were performed:

– depth of an in-seam working and its location relatively to a coal seam;

– structure of a coal-containing strata of adjacent rock layers, interval of change of their mechanical characteristics and properties of a coal seam;

– mining and technical parameters of in-seam working: shape and dimensions, type and parameters of the support, supported area and their mechanical characteristics;

– research of stress and strain state of the test geomechanical models to substantiate their dimensions based on the conditions of stabilization of stress components on the borders of the model according to initial non-hydrostatic condition;

– testing of geomechanical models to determine qualitative conformity of their stress and strain state to modern notions and results of researches of rock displacement processes close to a mine working;

– substantiation of contact conditions along stratification planes of rock layers of coal-containing strata.

It is well known that the most applicable numerical methods for solving of complicated geomechanical tasks of similar class, despite their high precision, do not possess a generality of this solution. Thus, methodological approach for solving of the given task is necessary, the aim of which is the search for the set of conditions of the widest application area of research results with a condition of minimal admissible volume of calculations dedicated to searching of various variants of combination of geometrical, mechanical and load-bearing parameters of system "stratified massif-support of the development working". It is quite hard to execute the second condition though, because in order to increase the correspondence of solution results to real mining-geological and mining-technical conditions, it is necessary to analyze a big number of variable parameters in all range of their variation. That is why on the second stage of the research, the geomechanical substantiation of structure of coal-containing strata of rocks in the vicinity of an in-seam working is performed and this most adequately reflects mining and geological conditions of coal seams at Western Donbass mines. Stress and strain state results of several models of coal-containing strata structure with variable mechanical characteristics of rock layers and at different mining depth have been received. Based upon the stress and strain state calculations, there was given a substantiation to the adequate model of coal-containing strata.

On the third stage, laws of change of stress and strain state system were explored, triggered by variation of deformation characteristics of rock layers with specified interval for Western Donbass.

The fourth stage is dedicated to laws of change of stress and strain state system with increase of depth of in-seam working. Influence of deformation characteristics of adjacent rock layers is simultaneously estimated for various depths. Tendencies of occurrence of limit-state of roof, wall and bottom rocks of the working are determined depending on the above stated geomechanical parameters and also strength characteristics of rock layers.

Thereby, conducted researches in quasi-elastic process, using finite-element method, made up first part of geomechanical processes modeling near in-seam working. When limiting state occurs, further description of displacement process of coal containing strata is performed according to full rock massif deformation diagram, that, according to modern views, contains stages of softening and "ruining" breakage (Bondarenko 2007, 2009 & Vinogradov 1989).

3 RESULTS

Displacement field of rocks that contain the working essentially depends on the condition of surrounding coal-containing strata that is defined by correlation of strength characteristics of rock layers and depth of the working. Analysis of laws of relation of wall rocks displacement field with depth of a working is performed, starting from variant of decreased strength and deformation characteristics of all adjacent rock layers of the system (Figure 1). At $H = 200$ m, some part of the coal-containing strata is located in prelimit state, therefore the displacements are quite moderate (230-480 mm). At $H = 400$ m, biggest part of the adjacent rock layers transfers into limit state that provokes increase of displacements in massif, including the contour of the working. This change of state of massif's part triggers clearly expressed non-linear connection of displacement with stratification depth. For example, with increase of H by two times (from 200 to 400 m), displacements in roof of a working increase in 2.7…2.9 times, in wall rocks and bottom – 2.5…2.8 times; in quality sense, distribution diagram of displacements distribution along working's contour slightly changes with smaller gradient of change of displacements in roof and bottom. At $H = 600$ m, displacement field changes even more both in qualitative and quantitative relations and this is connected with transfer not only into limit but also into out-of-limit states of overwhelming near-to-contour area of coal-containing massif:

– distribution diagram gains more asymmetric look relatively to vertical axis of the working: displacements of bottom and walls of the drift slightly prevail from the side of seam rise;

– displacements of roof and bottom are so big (up to 2.8 m in roof and 1.8 in bottom) that rock practically fills up entire cavity of the working, that is explained by process of loosening of a considerable volume of the massif in vicinity around the rock;

– displacements of rocks under props of the frame reach 0.7...1.2 m from each side, that also leads to entire loss of support stability and of the working in whole.

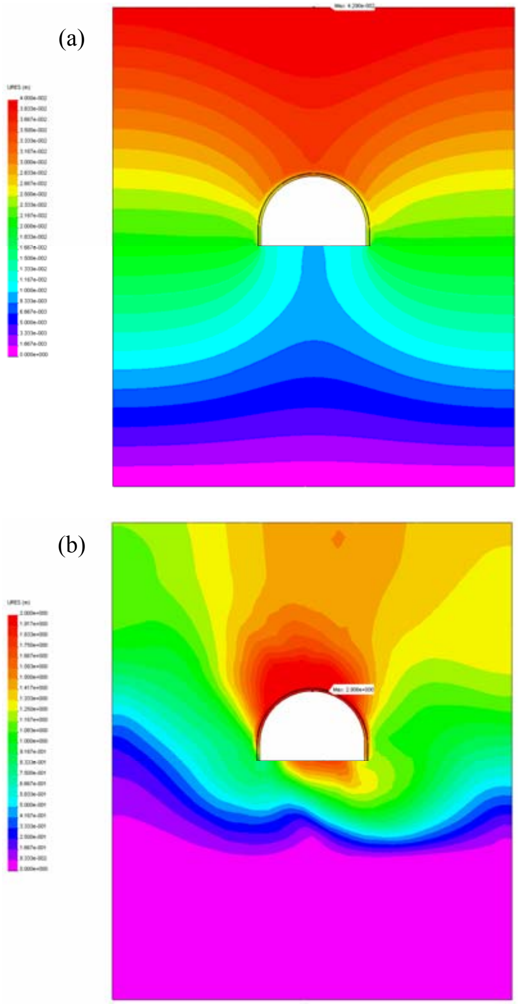

Figure 1. Distribution diagram of displacements in system "stratified massif-support of working" at decreased strength and deformation characteristics of all adjacent rock layers: (a) $H = 200$ m; (b) $H = 600$ m.

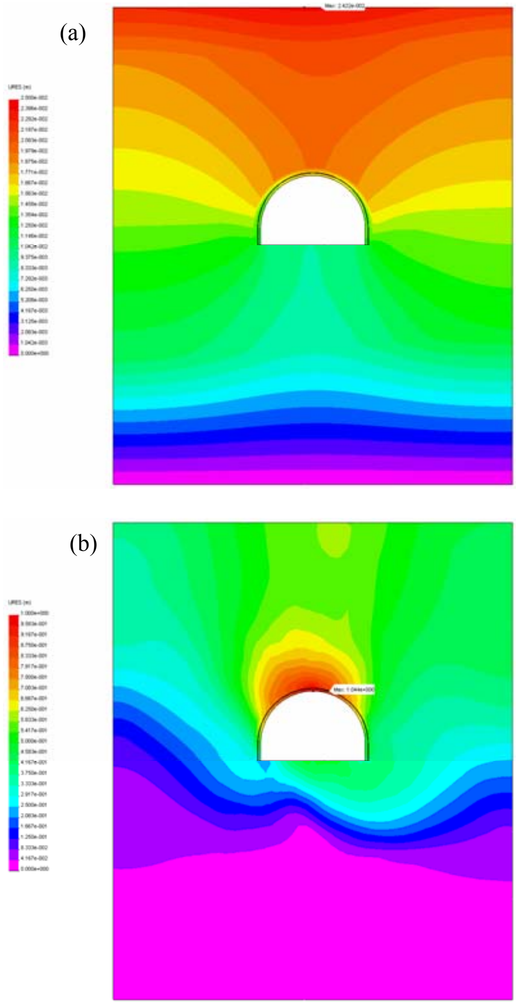

Figure 2. Distribution diagrams of displacements in system "stratified massif-support of a working" at decreased strength characteristics and increased deformation characteristics of all adjacent rock layers: (a) $H = 200$ m; (b) $H = 600$ m.

Listed factors testify about impossibility of exploitation of a working in considered mining and geological situation without application of corresponding technical measures for support stability increase.

Increase of modulus of deformation of all adjacent rock layers from 0.3×10^4 MPa to 1×10^4 MPa with their decreased strength leads to the following results (Figure 2). General tendency of increase influence $E_i^{r,b}$ is such that displacements of a massif in any point decrease at any depth of the working: at $H = 200$ m – in 1.5…2.3 times in roof and walls of working (displacements practically do not change in bottom); at $H = 400$ m – in 1.7…2.3 times; at $H = 600$ m – in 2.1…2.2 times in roof, in 1.7…1.9 times in walls and in 2.0…2.5 times in bottom. Therefore, there is a conclusion follows that in out-of-limit state of adjacent rock layers, influence of their deformation characteristics increases, especially in the working's bottom. Here the value of rock heave makes up 370…650 mm and allows to exploit the working after appropriate recovery operations.

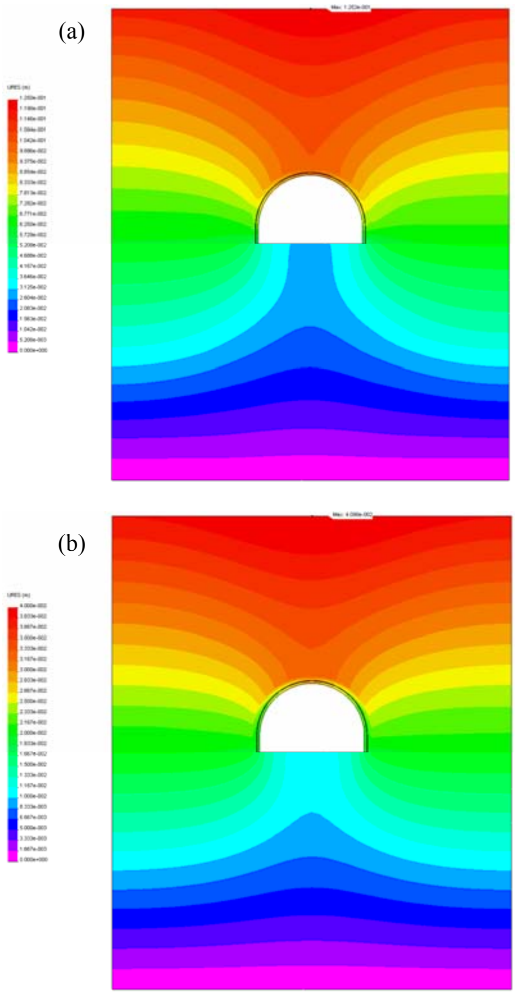

Figure 3. Distribution diagrams of displacements in system "stratified massif-support of working" at increased of strength and decreased of deformation characteristics of all adjacent rock layers: (a) $H = 200$ m; (b) $H = 600$ m.

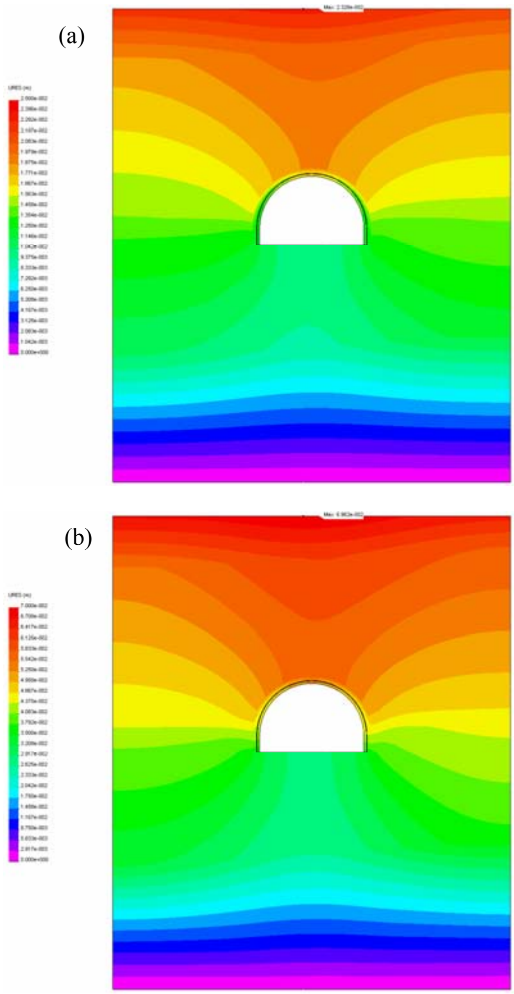

Figure 4. Distribution diagrams of displacements in system "stratified massif-support of working" at increased strength and deformation characteristics of all adjacent rock: (a) $H = 200$ m; (b) $H = 600$ m.

Increased strength characteristics of all adjacent rock layers radically change not only stress but also strain state of coal-containing strata.

At decreased deformation characteristics of all adjacent rock layers, the following features of change displacement field with increase of H are established (Figure 3):

– with different depth of working, qualitative distribution diagram in massif is virtually constant;

– displacements of rock contour of the working even at $H = 600$ m are relatively small (in roof – up to 490 mm, in bottom – up to 430 m, in walls – up to 170 mm) that is conditioned by predominantly prelimit state of coal-containing strata;

– for this reason, almost linear connection of displacement value in any point of a massif with depth of a working is observed.

Increased deformation characteristics of all rock layers (Figure 4) do not change previous conclusions, taking into account that value of displacement of a working's rock contour decreases in roof and walls in 1.7…2.0 times.

Change of deformation characteristics of the roof's second layer changes just a little the displacement field with all its features revealed earlier.

In conclusion of this stage of researches, influence of strength and deformation characteristics of the second rock layer on tendencies of change of displacement field with depth increase of a working is estimated. It is established that influence of deformation characteristics of the second layer of roof with different combinations of strength properties of adjacent rock layers with increase of parameter H is analogical to previously described variants.

As to the strength characteristics of the second layer of roof – their diverse influence depending on state of first rock layers of roof and bottom are discovered. At increased strength characteristics of the first layers of roof and bottom, transfer of the second rock layer of roof into limit state because of its decreased strength contributes to increase of displacements of rock contour of the working up to 20…28%. This is partially conditioned by prelimit state of rock layers that adjoin the working even at depth of 600 meters. When the first layers of roof and bottom transfer into limit (out-of-limit) state with depth increase – increased strength of the second layer of roof allows to restrict the displacement of rock contour of the working by 29…56%.

4 CONCLUSIONS

Results of conducted researches of tendencies of increase of rock massif displacements into a working's cavity with its depth increase, have allowed to formulate a line of conclusions:

– various variants of distribution of strength and deformation characteristics of adjacent rock layers cause different level of their influence on displacement of rock contour of the working with increase of mining depth. The defining factor is the condition kind of adjacent rock layers at specific value of parameter H: prelimit, limit, out-of-limit;

– at increased strength characteristics of adjacent rock layers, their partially prelimit state predetermines close to linear connection of displacements with parameter H, at which fluctuations of their deformation properties in range $E_i^{r,b} = (0.3...1)\cdot 10^4$ MPa change displacement value up to 70…100%;

– at lowered strength characteristics of rock layers, dependence of displacement of a working's contour on H becomes nonlinear when limit state of any of layers occurs and intensity of increase of displacements grows by several times.

The established laws will serve as a basis during development of prognosis method of exploitation condition of in-seam development workings that are driven within stratified massif of weak rocks.

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Research of influence of support resistance of the stope in the immediate roof condition

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ABSTRACT: The article covers results of studies of support resistance effect on stress-strain state of stope's roof rocks. It is shown that modification of support resistance at the face and character of shearing forces, moments of deflection take place within roof stratum. Increase in support resistance at the face space helps to avoid tension stresses near the longwall face in the lower part of a roof layer. In turn, it reduces the risk of its crushing and development of sudden outbursts from coal layer.

1 INTRODUCTION

As it is known, displacement of rock massif during underground coal mining takes place as layer-by-layer movement. In this case, each bed is not a rigidly restrained thin plate that has nonuniformity of basic load from overlying rocks and nonuniformity of normal reaction from underlying rocks. Maximum value of basic load and reaction is available within massif. They drop down to zero over stripped area gradually rising over stripped area where their maximum value becomes equal to gravity forces. Under all other conditions the character of distribution of basic load and reaction depends on coal-overlying unit structure and technological parameters. It is known that lower layers fall near mined bed. Thus, the two design models are formed while mining coal-overlying unit (Poltavets & Savostyanov 1994): thin plates not rigidly restrained from both sides and those not rigidly restrained from one side. The latter are layers located directly above the seam, and called roof stone. Under all other conditions basic load on the roof stone including technological parameters depends on its thickness as well as on thickness of overlying rocks. Thus, to determine modification of conditions of roof stone rocks it is required to determine parameters of load curve and reaction for all layers of coal-overlying unit from daylight area.

As it is a known, safe and efficient operation in stopes depends on ability to control rock condition within face space. As a rule, powered roof supports consisting of bearing elements (mostly they are two hydraulic legs, a bed, and console with a hood) are used for that today. Lately, a tendency to increase bearing force of hydraulic legs with similar support has been initialised. It considerably increases steel intensity of powered roof supports resulting in their high cost which cannot always be paid while operating. Practice of mining with the help of powered roof supports that have increased resistance often leads to emergencies connected with rigid support caving. Specific literature pays much attention to interworking relationship of stopes supports (Kiyashko 1984). But it is impossible today to answer questions connected with bearing force distribution between support legs. Thus, problems concerning effect of bearing force of support legs in state of roof rocks are current and well-timed.

2 BASIC PART

To determine supposed interaction mechanism of roof stone there is modeled stress-strain state of rocks while mining coal layer C_7 within mine field of Zapadno-Donbasskaya mine (prospecting bore #8231). There was longwall mining to the rise with the longwall's length being 150 м, roof control is defined as cave-in, width of the shearer is 0.8 m, and rate of face advance is 2 m / day. Coal-overlying unit was divided into 28 beds according to rock types during modeling process. Calculations for each bed were divided into two stages. At first, parameters of load curves depending on varying abovementioned technological parameters were determined for each of them. Key parameters of load curves for beds located nearby mined bed if the rate of face advance is 2 m / day are in Table 1. Chain of detailed calculations of load curve parameters and indices required to determine stress-strain state of roof stone rocks is specified in paper (Savostyanov & Klochkov 1992).

Table 1. Parameters of load curves for lower beds of coal-overlying unit.

Thickness of bed (m)	If rate of advance is 2 m / day			
	Geometrics (m)		Physical parameters (MPa)	
	a	d_0	S_2	ghp
0.7	1.0	0.8	19.1	17.0
2.0	2.9	2.0	24.3	21.1
12.0	12.8	8.5	31.2	15.0

Design of simulation model of interaction between main roof and roof stone with 0.7 m thickness is shown in Figure 1. Not rigidly restrained console experiences basic loads from the main roof, normal reaction from coal layer, and reaction from support. Central point of overhanging beam (Figure 1) was determined taking into account its thickness and rate of the face advance. Reaction of support is shown as trapezium with running values of r_1 and r_2.

Shearing forces, bending forces of its fault were calculated on the basis of parameters of basic load curve. The calculations took into account below mentioned expressions (Savostyanov & Klochkov 1992). In the expressions, the $f(K)$ is the modulus of deformation which is modified along the layer length according to the expression (5).

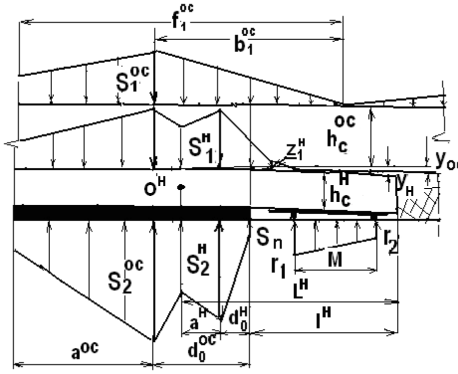

Figure 1. Design model to calculate stress-strain state of roof stone rocks.

$$O(x)=\sum_{1}^{3} B_k \sin\frac{k\pi}{L}x \tag{1}$$

$$M_x=\frac{L}{\pi}\sum_{1}^{3}\frac{B_k}{k}\sin\frac{k\pi}{L}x \tag{2}$$

$$\theta_c=\frac{2L^2}{\pi^2 f(K)I}\sum_{1}^{3}\frac{B_k}{k^2}\sin\frac{k\pi}{2L}x \tag{3}$$

$$Y_x=\frac{L^3}{\pi^3 f(K)I}\sum_{1}^{3}\frac{B_k}{k^3}\left(\cos\frac{k\pi}{2L}x-1\right) \tag{4}$$

$$K=E_0 \text{ if } 0<x<a$$

$$K=(E_0-E_n)\frac{f_2-x}{b_2}+E_n \text{ if } a<x<f_2 \tag{5}$$

$$K=E_n \text{ if } f_2<x\le L$$

To determine effect of stope's support, the stress-strain state of support of roof rocks was calculated. While simulating support, resistance near face and near border of face space varied from 50 tf / m^2 to 250 tf / m^2, and technological parameters of mining as well as rate of face advance stayed to be invariable.

Figure 2 shows lines of behaviour of shearing forces within the bed of roof stone depending on support resistance of stope. The lines' data study shows that the most important modifications of shearing forces take place when support resistance near longwall face is more than that near the goaf. Thus, if support resistance near face is 250 tf / m^2 then shearing forces of the face increase from 250 tf to 800 tf. With it, shear stress of shearing forces increases from 535 tf / m^2 to 1700 tf / m^2, practically thrice. Then increase in shear stress will take place within lower fourth of roof stone layer.

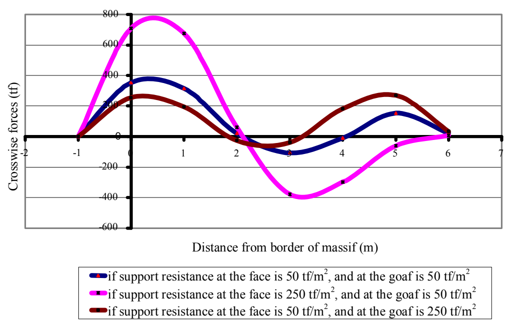

Figure 2. Lines of deflection moment modification and shearing forces of roof.

Over the central part of the face space the shearing forces change sign, and maximum value is within upper fourth of roof thickness. Here maximum value of shear stress is 850 tf / m^2. The most favourable conditions from the viewpoint of roof stone continuity take place when support resistance at the goaf is more than support resistance at the face space.

Under standard conditions if support is not available the moment of deflection along the length of layer changes sign. Within massif the moment of deflection is negative. Compression stresses originate in lower fourth of layer, and tensile stresses originate in the upper part. If support is available then behavior of moment of deflection varies. Figure 3 shows behavior of stress condition of roof stone if there is support with different resistance at the face and the goaf. Thus, if there is support at the face with important support resistance and small support resistance at the goaf near the face then tensile stresses take place within upper layer of roof stone, and compression stresses are within lower part of layer. It has favorable effect on rock continuity while mining with the help of coal shearer. If support resistance at the face is small, and support resistance at the goaf is important then character of stress distribution from moment of deflection varies. Near the face tensile stresses take place within lower part of roof stone layer, and compression stresses are available within upper part. It increases the risk of roof stone cave-in while mining with the help of coal cutter. Besides, if there is adhesion on the contact between rock and coal seam the overburden recasting during stress increases the risk of coal and gas outburst during mining. On the contrary, if we mean goaf, compression stresses originate within lower part of roof stone as a result of moment of deflection, and tensile stresses originate within upper part of the layer.

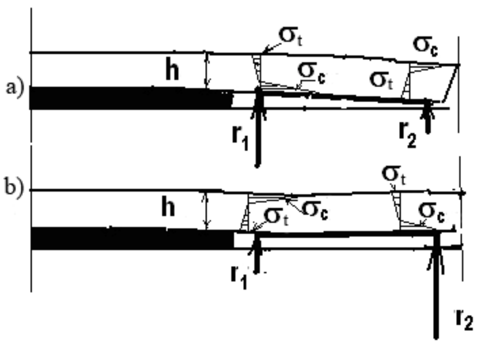

Figure 3. Behavior of stress condition of roof stone.

Moments of deflection and maximum stresses by moment of deflection are estimated for the considered conditions. Normal stresses are estimated for distinguished points at the face and the goaf where moments of deflection have maximum value. Estimation results for maximum normal stresses by moment of deflection are in Table 2. Figure 4 shows lines of behaviour of moments of deflection for the bed within roof stone depending on support resistance modification.

Table 2. Maximum normal stresses by moment of deflection.

Support resistance (tf / m^2) (at the faceat the goaf)	Normal stresses by moment of deflection (MPa)			
	At the face		At the goaf	
	Compression	Tension	Compression	Tension
50-50	51	13	26	7
250-50	123	32	28	7.2
50-250	49	12	43	10

Studies of data from Table 2 and lines show that increase in support resistance at the face influences significantly both behaviour of moment of deflection and its value. If support resistance at the face increases from 50 tf / m^2 to 250 tf / m^2 then moment of deflection at the face increased from 360 tfm to 839 tfm, more than 2.3 times. Maximum compression stresses by moment of deflection increased from 51 MPa to 123 MPa. If rock strength of roof stone is 25 MPa at the distance of 1m from the face, then roof stone crushing in its lower part should happen.

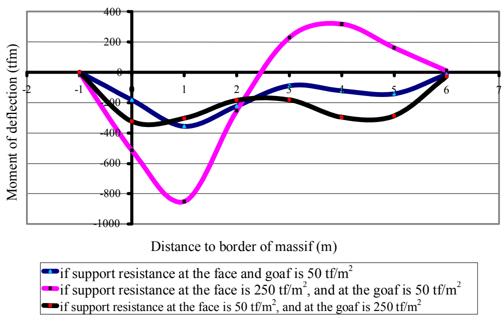

Figure 4. Lines of behaviour of moments of deflection along the length of roof stone.

As the data from Table 2 show, if support resistance increases at the goaf then moment of deflection modifies twice. But in this case, at the face it shifted to the face that is this maximum shifted to the face when support resistance increased. Before increase in support resistance was at the distance of 1m. After increase in support resistance up to 250 ts / m^2, maximum shifted to the face at the distance of 0.2 m. Thus, if support resistance continuous increasing at the goaf and support resistance continuous decreasing at the face then maximum at the face will shift to massif, and moment of deflection will change a sign at the face, and tensile stresses will originate within lower part of roof stone. Line shown in Figure 3b will appear. Within layers prone to sudden outbursts of coal and gas it may result in dynamic phenomenon development.

Regular operation in stope requires prescribed limits for value of roof stone lowering which to some extent depends on support resistance. Practice of mines shows that roof lowering at the face border is always more than main roof lowering. In other cases when main roof lowering rises to roof rock lowering then support load at the border of mining will rise sharply. If so support will be fastened “rigidly” and then damaged. In mining practice value of roof stone lowering at the border of face space attempts is engaged to be controlled at the expense of pliability of supports. In this connection, it is required to make preliminary forecasts of both main roof and roof stone lowering values.

To determine effect of support resistance on value of roof stone lowering there were estimated both the lowering and equivalent stresses within lower part of the bed. The estimation results are in Table 3. Typical sections and those along the whole length of roof stone are shown as lines in Figure 5.

Table 3. Roof stone lowering and equivalent stresses within typical section.

Support resistance (tf / m^2) (at the face-at the goaf)	Lowering (mm) and equivalent stresses (MPa)			
	At 1m distance of the face		At the goaf	
	Lowering	Stresses	Lowering	Stresses
50-50	51	22	287	16
250-50	83	51	405	72
50-250	41	18	234	17

Studies of Table 3 data help to state that within considered limits of the stope’s support resistance the most favourable conditions for continuity of bed are in variation two when support resistance is greater at the goaf than at the face space. For all variations, lowering of the face at the distance of 1m is within accuracy of estimations. But in variation three, when support resistance at the goaf higher compared to face space, insignificant increase in main roof lowering may result in rigid support fastening at the goaf.

Roof stone lowering at the goaf under main roof sharp lowering helps to avoid rigid support fastening.

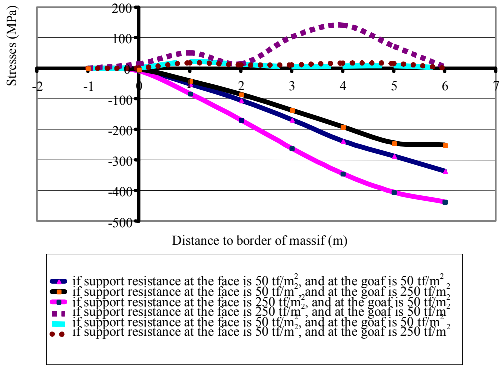

Figure 5. Lines of behaviour of equivalent stresses and roof stone lowering with 0.7 m thickness.

5 CONCLUSIONS

The research helps to come to the following conclusions:

1. Support resistance modification in the stope effects stress-strain state of main roof rocks;
2. Increase in support resistance at the face space helps to avoid tensile stresses at the face within lower part of roof stone. In turn, it reduces danger of its damage and development of sudden outbursts from a coal seam;
3. Increase resistance at the border of face space in the case of sharp increase in main roof lowering raises probability of support rigid fastening;
4. If support resistance increases greatly at the border of face space, tensile stresses originate within roof stone over the face under the layer by moment of deflection. It results in increased probability of sudden outburst development during shearer’s operation and roof stone failure.

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