DEVELOPMENT OF THE EFFICIENT AND SAFE
DOWNWARD SLICING SYSTEM WITH THE
BACKFILLING OF MINED-OUT SPACE

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ABSTRACT

There have been reviewed the ore mining options which are based on downward slicing system accompanied with the backfilling of mined-out space and use of mobile underground vehicles. There have been given both reasonable design parameters of the mining system, and recommendations on the slicing order at the sublevels and stopes in the layer. There have also been presented the ways for generation of the artificial rock mass proving efficient control over the rock pressure and backfilling stability.

KEY WORDS

Slicing system, stope, chamber, artificial overlap

At the present time, the downward slicing system with the backfilling of mined-out space is widespread. It can be applied at any angle and thickness of the steeply dipping ore body with soft ores and enclosing rocks, as well as in case of necessity to create unloading layers, mining rich ore reserves at emergency areas or under water objects by creating an artificial overlap (Figure 1; Ananin, A. I., Barilyuk, A. I., Tkachev, V. M., Makarov, A. B., & Ort, V. G., 2002; Bronnikov, D. M., & Tsygalov, M. N., 1989; Grigor’ev, A. M., Zoteev, O. V., Makarov, A. B., & Serafimin, A. P., 2013; Kaplunov, D. R., Kalmykov, V. N., & Rylnikova, M. V., 2003).

1- access ramp; 2- overlap (concrete, reinforced with arches of special replaceable material); 3- approach to the layer; 4- mined-out and backfilled stope; 5- overlap on the hanging side of the ore body; 6- ventilation and manway raise; 7- orepass
Preparation of the block includes drifting of the entries, access ramp, approaches to each layer, overpass and ventilation and manway raise. Block reserves are mined in layers in downward order. The mining of the ore in the layers is performed by stopes with sections of $4 \times 4$ m. The stopes are given an inclination angle of $2+3^\circ$ (for $1+2^\circ$ more than the spreading angle of the mixture) for completeness of the backfilling of mined-out space.

Mining can be performed simultaneously on two or three sublevels. In this case, the vertical distance between simultaneously mined layers in adjacent sublevels should be at least 15 m. Works on each sublevel are started with the generation of artificial overlap by mining and backfilling of the top layer. With a very soft ore, mining of the stopes is performed with the use of arch support of special replaceable material. Reinforcement of the protective overlap is achieved by fittings and metal mesh (Figure 2; Ananin, A. I., Barilyuk, A. I., Ryshkel, I. A., Gultyaev, V. G., Kolobov, V. G., & Faustov, S. I., 2002).
shifted by half the mining width. It is recommended to set the length of the stope taking into account the following factors:
- stand-by time of stope without backfilling (the longer the period, the less stable the roof becomes);
- the need to fix intermediate backfilling lintels;
- provision of the necessary stoping ground (the number of simultaneously mined stopes in the block).

Stopes in the layer are mined in three turns, two turns or in a cutback. Ore pillars are left between the first turn stopes in case of mining in three turns. Stopes of the second turn are adjacent to the filling mass on the one side and on the other to the ore mass. Stopes of the third turn are mined out between the backfilled stopes. Between simultaneously mined stopes there should be an 8-meter ore mass, filling mass or ore-filling mass. In this case, the strength of the backfilling should be at least 0.7 MPa. When mining stopes in cutback, a hard ore mass, which has a positive effect on its stability, is located on the one side of the mined stope.

In order to increase the stability of the roof of the mined stope at the contact with the backfilling, it is necessary to install the anchor posts before the backfilling (Figure 3; Ananin, A. I., Kunanbaev, N. S., Faustov, S. I., Ort, V. G., & Musurmankulov, S. A., 2002).

![Figure 3 - Installation diagram for anchor posts](image)

When the layer is mined out by stopes in three turns, the pit props are installed on both sides only in the first turn. During the mining of second turn stopes, the pit props are installed only from the side of the ore mass, and during the mining of third turn stopes, the pit props are not installed. The stopes are backfilled with hardening mixtures with the formation of the base layer and a layer of reduced strength.

The normative strength of the base layer is determined by the span of undermining, its thickness, the presence of layering and reinforcement, and also the order of stopes mining in the underlying layer. When the span is 4 m, the height of the base layer is 1.5 m, the strength at the time of undermining must be at least 4 MPa if the layer is not reinforced, and 3.5 MPa at the presence of reinforcement.
The strength of the artificial mass in the layer of reduced strength (the filling-up layer) is determined by the stability of escarpments, and at their height from 3.5 to 6 m it should be from 0.7 to 1.0 MPa, respectively. Losses and contamination of ore should not exceed 5% by calculations.

CONCLUSION

The ore mining options which are based on downward slicing system accompanied with the backfilling of mined-out space with hardening material were developed and protected by the patents of the Republic of Kazakhstan (Ananin, A. I., Barilyuk, A. I., Ryshkel, I. A., Gultyaev, V. G., Kolobov, V. G., & Faustov, S. I., 2002; Ananin, A. I., Kunanbaev, N. S., Faustov, S. I., Ort, V. G., & Musurmankulov, S. A., 2002) and implemented at the Orlovsky mine of the Zhezkentsky Manufacturing Complex and at the Artemeyevsky mine of Artemeyevsky Manufacturing Complex of Vostoktsvetmet LLP. These options allowed:
- to increase the productivity of stoping in the block;
- to ensure the stability of artificial roof by sequencing the stope mining in the layer and sublevels in the block without reduction of the ore mining, applying rational methods for formation of the filling mass, its reinforcement and fixation of the stopes.

REFERENCES

GEOMECHANICAL DIMENSIONING AND IMPLEMENTATION OF INNOVATIVE SYSTEMS FOR MINING OUT OF VEIN DEPOSITS

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ABSTRACT

There are several alternative systems for development of vein deposits of mineral resources in the underground mining. These are: shrinkage, top slicing caving, sublevel workings, sublevel caving, filling, supporting and filling.

This article aims to present an innovative technological approach and geomechanical assessment of the systems with sublevel caving and filling in the vein deposits by using mobile mechanization.

The study is targeted at implementation of the above-mentioned systems for mining out of the vein deposits in Varba-Batantsi mine under level 720 in the South Eastern Rhodope Mountains in Bulgaria.

KEY WORDS

Geomechanical assessment, innovative system for development, mobile mechanization

INTRODUCTION

Varba-Batantsi deposit is situated in the lands of the village of Varba and the town of Madan, Smolyan district in the Republic of Bulgaria, south-eastern part of the west Rhodope Mountains. It is opened up with three vertical shafts (Atanasov et al., 2013).

The current system of development (system with sublevel caving) in Varba-Batantsi mine is not sufficiently productive and does not guarantee the maximum mining out of all proven reserves in the deposit. In addition, it does not provide for the maximum safety of the workers. All of this generates different geomechanical problems in terms of scope and area and determines a number of interrelated risk factors such as high lateral rock pressure in the mining layers, resulting in inadequate sizing and use of supporting structures. This is also due to the unfamiliarity with the state of the country rock around the ore body.

CHARACTERISTICS OF THE PROBLEM

The problems that are presented can mainly be divided into two aspects: extraction (technological) and securing the subsequent stability when applying the adopted technologies and development systems (geomechanical).

The problems with the mining works are related to:
Problems with mining out the reserves under level 590 around one of the shaft pillars.
Problems with the systems for development applied so far (system for sublevel caving) – their low productivity.

The geomechanical problems that appear are also of two types:
The first one is the subsequent (secondary) “rock pressure” that occurs close to the servicing mining workings which are located around the extracted mining blocks (Brady, Brown. 2004).
The second is the deformations that occur in the hanging wall after the formation of one (or more) mining layer.

All these problems require a single solution of the interrelated key factors, which will solve the problems determining the future exploitation of the deposit.

The paper aims to assess the geomechanical state of the rock mass and offer an innovative development system generating a minimum geomechanical risk that will ensure adequate yield and extraction of maximum reserves in the Varba-Batantsi mine.

**ANALYSIS AND DETERMINATION OF THE GEOMECHANICAL STATE OF THE ROCK MASS**

In order to make adequate classification assessments of the rock mass, it is necessary to accumulate data about the structural disturbance of the rock, which has the information necessary to determine the technological properties of the rocks. (ISRM Suggested Methods, 1981)

The study of the structural disturbance of the rock mass in the Varba-Batantsi mine is necessitated by the need to define the stable dimensions of the open mining areas using the adopted and widely applied system of sublevel caving (Anastasov et al., 2013).

The study of the structural disturbance of the rock mass is carried out using the method of mass measurement of the joints and reporting them after the driving of controlling boreholes. When analyzing the influence of the systems of joints on the mining areas, it is necessary to assess their impact as well. The small distances influence mining workings of small size (galleries), and the large distances – the workings of large sizes (chambers). Having in mind this, to check the stability of the large open areas of the mined out spaces, the mutual orientation of the walls and the systems of joints in the mining blocks is checked as well. The study related to the structural disturbance of the rock mass ends with defining the state of the joints (solubility, saturation, weathering, etc.)

The software product DIPS ver. 7.0 has been used to analyse the results related to the structural disturbance of the rock mass.

The analysis of the results obtained (Fig. 1) leads to the following conclusions: there are three systems of joints including arbitrary ones in the host rock mass around the ore body (ore vein); the rock mass is block cracked and at their intersection the systems of joints form wedges and cause instability of the lying and hanging wall, which leads to the subsequent caving.

![Figure 1](image-url)

**Figure 1** - Results from the study of the global structural disturbance of the joints.

a) Polar stereo network of equal area with clustering and contours of the joints’ poles; b) the three systems in a diagram “rose of the joints”; c) orientation of the three main systems of joints in the rock mass.

On the basis of the results obtained, studies concerning the determination of the strength-deformation and physical and mechanical properties of the rock mass have been carried out.
After the preliminary study, drilling sites have been selected to enable characterization of the country rock mass and the ore body. The results of these studies are presented in Table 1, while Figure 2 presents some of the boreholes which were made to characterize the country rock mass in depth.

Table 1 – Physical and mechanical properties of the lithological rock varieties in Varba-Batantsi mine

<table>
<thead>
<tr>
<th>Indicator</th>
<th>Parameter</th>
<th>$\sigma_c$ (MPa)</th>
<th>$\sigma_{c, PLT}$ (MPa)</th>
<th>$\gamma$ (MN/m³)</th>
<th>$C$, MPa</th>
<th>$\varphi$, °</th>
</tr>
</thead>
<tbody>
<tr>
<td>Amphibole-Biotite-Gneiss</td>
<td></td>
<td>46.7</td>
<td>52.7</td>
<td>8.2</td>
<td>0.0262</td>
<td>9.6</td>
</tr>
<tr>
<td>Granite gneiss</td>
<td></td>
<td>58.2</td>
<td>64.2</td>
<td>12.4</td>
<td>0.0256</td>
<td>13.8</td>
</tr>
<tr>
<td>Biotite gneiss</td>
<td></td>
<td>38.9</td>
<td>32.1</td>
<td>7.5</td>
<td>0.0254</td>
<td>8.7</td>
</tr>
<tr>
<td>Quartz gneiss</td>
<td></td>
<td>31.7</td>
<td>27.7</td>
<td>6.4</td>
<td>0.0252</td>
<td>7.3</td>
</tr>
<tr>
<td>Quartz biotite gneiss</td>
<td></td>
<td>28.8</td>
<td>31.4</td>
<td>4.2</td>
<td>0.0250</td>
<td>6.0</td>
</tr>
</tbody>
</table>

Figure 2 – Controlling boreholes to characterize the host rock mass in depth

Geomechanical classification assessments of the country rock mass and the ore body

The next step for identifying the factors which determine the type, applicability and effectiveness of the applied development system is the compilation of geomechanical classification assessments. This is done in order to identify the main geomechanical risks and to take effective measures for ensuring the future system for development.

Due to the lack of data characterizing the state of the host rock and the ore body, classification assessment under the RMR system (Bieniawski, 1976) was performed first. Each constituent involved in the Bieniawski equation is defined separately. RMR’s assessment includes a study of the state of the country rock and the ore body. The results obtained for the host rocks in the mass and for the ore vein are presented in Table 2. To further verify the obtained results for RMR, another evaluation was made using the results of an additional classification assessment for Q (Barton et al., 1974). The two assessments were compared using the correlation $RMR=9\ln Q + 44$.

Table 2 – Comparison of RMR and Q using the correlation $RMR=9\ln Q + 44$

<table>
<thead>
<tr>
<th>Location</th>
<th>Assessment</th>
<th>RMR</th>
<th>Q</th>
<th>RMR=9lnQ+44</th>
</tr>
</thead>
<tbody>
<tr>
<td>Host rock – Lying wall</td>
<td>71</td>
<td>16.25</td>
<td>69</td>
<td></td>
</tr>
<tr>
<td>Host rock – Hanging wall</td>
<td>63</td>
<td>8.19</td>
<td>64</td>
<td></td>
</tr>
<tr>
<td>Ore body</td>
<td>68</td>
<td>14.16</td>
<td>68</td>
<td></td>
</tr>
</tbody>
</table>

The analysis was performed for a mining block with dimensions typical for the current system for development (Figure 2). The figure shows five major mining steps (sublevels) that reflect the development of the mining block in depth. Figure 3 A) shows the outline of the mining block (50x50m) in a longitudinal
vertical section and in Fig. 3 B) is presented the transverse vertical section of the extracted ore vein and in both figures are presented the basic mining steps for development of the block in depth.

Figure 3 - Five states of mining works reflecting their development in depth

In order to carry out an adequate geomechanical dimensioning and provision of the system for development and of the country rock around the mining areas, it is necessary to make further classification assessments characterizing the caving of the host rock mass and the dimensions of the permissible open mining areas. For this purpose, the Laubscher MRMR classifications (Laubscher, 1990, Mathews, 1995) and Mathews’s Graphical Method (Mathews, 1981, 1995) for determining the stable dimensions of underground open mining areas (Mathews Stability Graph) are applied as being the most appropriate. The second method allows a preliminary reverse analysis of the resistance at selected dimensions of the open mining space (the chamber) to be made.

The results of these two classification assessments related to the walls of the open mining areas (the mining block) are reflected in the development stages with increase of the block’s dimensions. The studies were done for one mining section and the results are presented in Table 3 and Figure 3.

Table 3 - Results from the assessment of the state of the rock mass: a) MRMR assessment; b) graphical method Mathews Stability Graph

<table>
<thead>
<tr>
<th>Mining step</th>
<th>Hydraulic radius, m</th>
<th>Surface area, m²</th>
<th>Perimeter, m</th>
<th>HR, m</th>
</tr>
</thead>
<tbody>
<tr>
<td>One mining block MRMR=46</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>+1</td>
<td>333</td>
<td>26</td>
<td>12.8</td>
<td></td>
</tr>
<tr>
<td>+2</td>
<td>580</td>
<td>40</td>
<td>14.5</td>
<td></td>
</tr>
<tr>
<td>+3</td>
<td>1650</td>
<td>54</td>
<td>30.6</td>
<td></td>
</tr>
<tr>
<td>+4</td>
<td>2140</td>
<td>68</td>
<td>31.5</td>
<td></td>
</tr>
<tr>
<td>+5</td>
<td>2940</td>
<td>91</td>
<td>33.0</td>
<td></td>
</tr>
</tbody>
</table>

Parameter Wall Lying HR, m N’
Lying 13.5 34.5
Hanging 19.8 16.2
Ceiling 7.5 10.2

The verification of the resistance of the walls (the ceiling) of the mining area is made according to the Laubscher’s graph Fig. 4 A) and to the Mathews Stability Graph - Fig. 4 B)
Figure 4 - Results from the MRMR classification assessment and the Mathews graph

As a result of the geomechanical assessment of the state of the applied system for development with sublevel caving (Figure 5), several important conclusions can be made:

1. The applied development system does not provide for the necessary subsequent geomechanical stability of the formed open mining spaces.
2. The complete mining out of the reserves is not guaranteed and, when it is achieved, the impoverishment is of high percentage.
3. There are follow-up factors which are risky and lead to the occurrence of mining induced seismicity (Board et al., 2005) in case of sudden subsequent caving.

Figure 5 - System for development with sublevel caving applied in Varba-Batantsi mine

IMPLEMENTATION OF AN INNOVATIVE SYSTEM FOR DEVELOPMENT OF VARBA-BATANTSI MINE

As a result of the detailed geomechanical assessment, an innovative system for development, which is suitable for the conditions of the mine, has been proposed. The system has a number of advantages over the system with sublevel caving (Anastasov et al., 2013) which has been applied so far in the Varba-Batantsi mine.

The proposed innovative system is for filling and using of mobile machinery for the mining and quarrying processes (blasting, loading, delivery) and filling of the mined space (Figure 6).

The access to the mining layers is made by an incline 9 and cut-off entries 8. The stope rises 3, 7 and 11 as well as roadway 1 and air gate 2 are also included in the preparatory works.

Through the cutting works - workings 16 and 17, the block is divided into two parts, and the mining is performed independently with two sets of machines - jumbos and front-end loaders. This increases the intensity of mining and the productivity of blasting and delivery.

The jumbos and the front-end loaders are of a low class with drilling performance of 1-1.5 m / min and delivery performance 24 m³/h at a distance of 105 m.
The mining face is fastened with anchor supports to the ceiling and the lateral openings in a net of 1.5 x 1.5 m, the height of the mining layer being 3.20 m.

The filling is done with rock material from the driven preparatory workings, the material being delivered along an incline 9 and rises 3 and 11.

As a result, low operating losses of 3 to 5% and low impoverishment - from 3 to 5%, which compensate for the filling and anchoring of the mined out space, are expected.

The productivity in the stope is increased 3 to 4 times in comparison to the system with sublevel caving and manual mining applied so far.

In order to mine out the ore deposits in the ends of the Varba-Batantsi deposit, we offer a system with sublevel caving and mobile mechanization (Figure 7).

The operation block is divided into four sublevels, and the movement of the machines is ensured by the preparatory incline 4.

The preparatory work includes the driving of a stope rise – ore chute 2, the road haulage tunnel 1, the cut-off entries 3 and 5.

The cutting works are shown through the driving of the sublevel roadways 6 and the slots 7.

The mining is done through a blasting technology, including borehole drilling, charging with emulsion explosive and initiation of the charges with the Nonel blasting system. At the same time, works are carried out on three sublevels, which ensure high productivity of the mining processes.

The delivery of the ore is done via front-end loaders with a basket volume of 2 m$^3$ at a maximum haulage length of 100-110 m - from the stopes to the ore chute.

The transportation of the ore from the ore chute to the final point (the capital ore chute) is with a middle-class underground truck with a load capacity of 20 t.

The expected operating losses are from 6 to 8% and impoverishment from 10 to 12%.
The presented advantages guarantee high productivity and minimum geomechanical risk for the workers. The mining processes are carried out using mobile machinery, ensuring high intensity of the mining works. Temporary non-active reserves are being put into operation in pit eye pillars, under roads, rivers, ponds and other engineering facilities.

A high level for the mining out of the reserves and several times lower losses and impoverishment are provided. The reduced losses from the damage and impoverishment compensate for the cost of the mobile mechanization and the application of the rock-cement filling of the mined out layers of the ore mass.

In addition, the advantages of the proposed systems are: innovativeness, efficiency and flexibility, as well as geomechanical provision which ensures safe mining.

CONCLUSION

The proposed innovative systems for development allow for the mining out of exposed and proven reserves in the vicinity of capital-exposed mining works, blocked – undermined reserves of medium and large depths, reserves to be mined out in newly formed mining blocks, etc.

The new development system with filling and division of the mining block into two parts with one ore pillar allows for higher productivity with the simultaneous operation of several mining stopes, which is a fundamental and substantial advantage in the case of reserves mined out in an underground way.

REFERENCES


SHEAR TESTING OF CABLE BOLTS USED IN AUSTRALIAN MINES AND TUNNELS

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SHEAR TESTING OF CABLE BOLTS USED IN AUSTRALIAN MINES AND TUNNELS

ABSTRACT

For decades cable bolt technology has been used for ground reinforcement in civil, mining and other construction projects. The strength properties of cables, used as cable bolts, have been evaluated mainly by their ultimate tensile strength as this kind of test can be carried out in the field as well as in the laboratory. Only recently there has been a growing interest in cable bolt failures in shear, because of documented field failure evidence. Shear testing of various cable bolts were made using different shear testing methods using 40 MPa strength concrete as the composite medium for cable installation. Single shear testing was carried out in cylindrically shaped concrete medium, while in double shear method tests were made in rectangular blocks. Cables used for tests include plain and indented wire cable strands as well as a combination of both types in one strand. Additional parameters considered included cable pretension load, the influence of bulbing, concrete confinement and the effect of sheared joint face contacts. It was found that the single shear testing rig produced consistent results in comparison to double shear testing methods, that the effectiveness of concrete confinement by steel clamps contributed to consistency in shear failure results, that increased pretention load resulted in lower cable failure load and reduced shear displacement, indentation contributed to reduced failure load in comparison to smooth wire strands, and bulbing appeared to have a significant bearing on the anchorage capacity of the cable in the medium. Plain wire strand cables are prone to debonding in comparison with indented cable strands.

KEYWORDS
Cable bolt, plain and indented strand wires, shear testing methods and cable pretension loads, concrete confinement

INTRODUCTION

Cable bolting has been used world-wide as a solution for structural support and in ground reinforcement in civil, mining, tunnelling and other structure projects. The strength properties of these cables, used as cable bolts, have been evaluated mainly for their ultimate tensile strength, as this kind of test could be carried out in the field as well as in the laboratory.

For the past several years, significant knowledge has been gained on tendon load transfer mechanisms and strength characterisation mainly by pull testing (Aziz and Jalalifar, 2005, Hagan, et al., 2015), however little has been known about the cable bolt shear behaviour, since the interest in cable bolt failure in shear has been confined to small amount of work carried out based on the British Standard of shear testing (BS 7861-part 2, 2009) and the work of Craig and Aziz, 2010, and Aziz, et al.2015a). Also, no credible test results are available from the field and only pictorial evidence has recently surfaced for both failed solid rock bolts and cable bolts. Typical signs of sheared tendons recovered from the field and a borehole view shear displacement in rock layers is shown in Figure 1, as reported by McCowan (2015), and Li (2017)
When a cable bolt is sheared to failure in a soft medium such as in soft rock and in weak concrete, there is little chance of the wires in the cable strand failing fully or snap in shear, instead the strand wires are likely to fail in a combination of both tensile and shear. Other influencing factors include grout strength, applied pretension load, testing method and loading condition (Yang, et al., 2017).

SHEAR TESTING OF CABLE BOLTS

Methodology

Shear testing of cable bolts has, for several recent years, been undertaken by using both the single and double shear methods. A single shear test method, based on the British Standard BS 7861 Part 2 (2009), was used to determine the shear strength of cable bolts to failure (Aziz et al., 2015b). Aziz, (2004) undertook double shear testing of 15.2 mm diameter resin coated, seven wire strand cable bolts to examine the extent of plastic surface damage with respective to increased shearing displacement. Testing of cable bolts to full failure in shear was subsequently carried out in a larger double shear machine (DS-MKII) by Aziz and Craig (2010). Further studies on the cable bolt load transfer mechanism have since been undertaken with emphasis directed solely on determining the load transfer characteristics with particular reference to evaluating failure profiles of various wires in the strand.

Currently, two types of testing rig are used for shear testing of various cable bolts, they are; Megabolt Integrated Single Shear Test Rig (MISSTR) and Double Shear Test Rig (DSTR). Prior to the construction of MISSTR, all studies in shear testing of tendons were undertaken using the University of Wollongong DSTR.

Double Shear Testing Method

Two types of double shear testing methods were available for evaluating the shear characteristics of cable bolts; (a) The DSTR-MKII with opposing concrete joint faces being in contact with each other, where the resultant shearing force is a combination of the shear failure load and friction force of the sheared host medium faces, (b) A modified DSTR (MKIII), with opposing concrete joint faces not in contact with each other and the measured shear resistance force is spent on shearing the cable wires. Determining the level of shear force spent on overcoming the friction force can be determined using the following equation based on the mathematical model based on the combination of Mohr Coulomb criterion and Fourier series scheme (Aziz et al., 2015).
\[
\tau_p = \left(\frac{a_0}{2} + \sum_{n=1}^{3} a_n \cos\left(\frac{2n\pi T}{\cos^{-1}\left(\frac{-4a_2 + \sqrt{16a_2^2 - 48a_2a_3 + 144a_3^2}}{24a_3}\right)}\right)\right) \tan(\phi) + c
\]

where, \(\tau\) is the shear stress, \(S\) is the shear load, \(C\) is cohesion, \(a_n\) is Fourier Coefficient, \(n\) is the number of Fourier Coefficient, which is considered between 0 and 3, \(u\) is the shear displacement and \(T\) is the shearing length. Aziz, et al., 2016 verified the effect of the equation with experimental test results.

Both the DSTR- MKII and MKIII rig as shown in Figure 2 were used in this study. The basic frame was the same and consisted of two 300 mm long outer cubic boxes and a 450 mm long middle central cuboid box with 300×300 mm \(^2\) cross-sectional area. A conduit wrapped with 8 mm PVC hose, was laid horizontally along the mould to precast a rifled hole through the centre of concrete blocks. Once the concrete was poured it was left to set.

![Figure 2- Double shear test rigs (a) MKII and (b) MKIII.](image)

Prior to the apparatus being assembled, the hollow central tube of each cable was filled with grout and left to harden prior to encapsulation in the concrete blocks for at least one week. During assembling, three concrete blocks were all mounted on the horizontal steel base. When assembling the DS apparatus the blocks were pressed against each other and the cable was pretensioned and then the cable was grouted as show in Figure 2 a. While in the MKIII set up the assembly was held together using a truss system/braces around the double shear assembly as shown in Figure 2 b. The truss system consisted of four 1100 mm long steel braces connected between two 30 mm thick side steel plates. The brace system impedes subjecting lateral axial load on concrete blocks during shearing. When assembled, gaps of almost 5 mm were left between concrete blocks, thus the adjacent sheared concrete faces are kept apart eliminating contact between the sheared faces and hence no friction force. Next the cable bolt was inserted into the central axial hole and was followed by mounting a100 t load cell on each protruding side of the cable in the assembled concrete blocks and tensioned to the predetermined axial pretension load, using a “Blue Healer” tensioner. Tensioning of the cable was retained by the barrel and wedge retainers. This was followed by the injection of grout into the central concrete blocks hole for bolt encapsulation. Grouting of the cable in the concrete block was achieved via 20 mm diameter holes cast on top of each concrete block. Once the cable was pretensioned, cement grout mortar was injected into the hole annulus space around the cable strand, from the vertically pre-cast radial hole on top of each concrete block. After seven days of grout/resin curing
time, the double shear assembly was then placed on the carrier base frame consisting of a parallel pair of rail track sections welded to a 30 mm thick steel plate. The outer 300 mm side cube blocks of the double shear apparatus was mounted on 100 mm steel blocks, leaving the central 450 mm long block free to be vertically sheared down using a 500 t capacity hydraulic universal testing machine at the rate of 1 mm/min for the maximum 100 mm vertical displacement. A hydraulic universal testing machine with a capacity of 500 t was used to compress the middle block for shearing the cable strand at the rate of 1 mm/min for the maximum 100 mm vertical displacement. Table 1 shows the peaks shear loads and axial forces of SUMO cable bolts with joint faces in contact with each other. The load displacement profiles are shown in Figures 3. Table 2 and Figure 4 show test results and load profiles with joints in contact.

### Table 1 Double shear test results of cable bolts with joints face in contact with each other

<table>
<thead>
<tr>
<th>Test NO.</th>
<th>Cable type</th>
<th>Nominal pre-tension (t)</th>
<th>Shear displacement at maximum shear load (mm)</th>
<th>Maximum shear load (kN)</th>
<th>Friction load 30%</th>
<th>Shear load per side (kN)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Plain SUMO</td>
<td>25</td>
<td>58.8</td>
<td>1424</td>
<td>427</td>
<td>499</td>
</tr>
<tr>
<td>2</td>
<td>10</td>
<td>10</td>
<td>78.9</td>
<td>1318</td>
<td>395</td>
<td>462</td>
</tr>
<tr>
<td>3</td>
<td>25</td>
<td>25</td>
<td>32.6</td>
<td>829</td>
<td>249</td>
<td>290</td>
</tr>
<tr>
<td>4</td>
<td>ID SUMO</td>
<td>10</td>
<td>46.0</td>
<td>933</td>
<td>280</td>
<td>327</td>
</tr>
</tbody>
</table>

Figure 3: Double shear load-displacement results of with concrete joints surface in contact with each other contacts

### Table 2: Double shear load-displacement results with no concrete joints face surface contact

<table>
<thead>
<tr>
<th>Test No</th>
<th>Cable type</th>
<th>Nominal pre-tension (t)</th>
<th>Shear displacement (mm)</th>
<th>Maximum shear force (kN)</th>
<th>Maximum shear force per side (kN)</th>
</tr>
</thead>
<tbody>
<tr>
<td>5</td>
<td>Plain SUMO</td>
<td>15</td>
<td>88.2</td>
<td>852</td>
<td>426</td>
</tr>
<tr>
<td>6</td>
<td>0</td>
<td>10</td>
<td>105</td>
<td>886</td>
<td>443</td>
</tr>
<tr>
<td>7</td>
<td>ID-SUMO</td>
<td>15</td>
<td>85.7</td>
<td>767</td>
<td>384</td>
</tr>
</tbody>
</table>
Figure 4: Double shear load-displacement results of with concrete joints surface in no contact with each other.

**Single Shear Test**

To replicate as closely as possible field conditions for installed cable bolts, the Megabolt Integrated Single Shear Test Rig (MISSTR) shown in Figure 5 was used to evaluate the behaviour of cable strand in shear (McKenzie, & King, 2015). Based on the principle of British Standard 7862-part 2 (2009) the whole length of the 250 mm diameter concrete cylinder used in MISSTR is 3600 mm (1800 mm on each side) with central hole diameter of 28-55 mm. The diameter of the central axial hole in the concrete was dependent on the diameter of tested cable bolt.

![Megabolt integrated single shear rig (MISSTR)](image)

The MISSTR is a horizontally aligned integrated system consisting of a shearing rig and an integrated 120 t capacity compression machine. The 3.6 m long concrete shearing cylinder consists of
two sections, each containing 1.8 m long concrete cylinders. The concrete cylinders are covered by steel clamps, which provide confinement during the shearing process. Either a hand pump or a power pack of a suitable capacity applies the hydraulic pressure for the compression machine legs. The pressure in the manifold was monitored with a digital pressure transducer (Type Measure X, range 0-800 Bar) in conjunction with an analogue pressure gauge (0-700 bar). The rate of loading was applied manually, which was not constant, however the aim was to apply a constant load at the rate of around 1 mm/min (0.018 mm/sec), in line with BS7861-2 standards. The displacement at the shearing plane was measured using a Linear Variable Differential Transformer (LVDT) as shown in Figure 5. Two other LVDTs were also mounted on the cable ends to enable monitoring of cable debonding. A data taker recorder was used to collect data during the tests.

When preparing, two 1800 mm concrete cylinders, two 900 mm cylinders were butt-glued together in a specially built tensioning frame. The cable bolt is then inserted through the centre rifled hole of the concrete cylinder. The cable bolt was pre-tensioned. The whole concrete cylinder loaded frame with cylinder was then tilted for 65 degree and grout was pumped from the bottom up the hole to remove any air bubbles remaining inside the grout annulus area and to ensure full cable encapsulation. Stratabinder HS grout was used to encapsulate all tested cables in this programme of study. The strength properties of the grout have been reported by Majoor et al. (2017), and Mirza, et al. (2016).

After a grout curing, each concrete sample with encapsulated cable bolt was disassembled from the frame and lifted out to be mounted on to the shearing rig. Once the concrete cylinder was correctly placed in the shearing machine, steel clamps were placed around the concrete blocks to provide a confining pressure to the sample. When sheared one side of 1.8 m of the 3.6 m concrete column remains fixed on the rig, while the other half is subjected to shearing. The applied shear load was recorded in a data taker and the displacement of cable ends and sheared cable strand wires were monitored by LVDTs, which were all logged by computer. Initially 16 single shear tests were conducted on the 3.6 m concrete blocks to study the effect of cable type, surface profile type, pre-tension load, birdcage structure, bonding and debonding, and the failure mode of cable bolts as shown in Table 3.

Plain wire cable bolts were found to have higher peak shear load compared with indented cable bolts. Figure 6 shows the cross section views of both MW9 Spiral and MW10 Plain wires, with both wires of equal diameter of 7 mm. However some minor strength reduction may occur because of the rifling or spiralling process during manufacture, but no weight loss. Figures 7 (a and b) show the load-displacement profile variations of other manufactures cables strand wires, where up to 10 % weight and strength loss can occur during indentation process.

Figure 6: MW10 plain and MW 9 spiral wires, with both wires of 7 mm in diameter (McKenzie and King, 2014)
### Table 3: Summary of single shear test results

<table>
<thead>
<tr>
<th>Test No.</th>
<th>Product Name</th>
<th>Cable dia. (mm)</th>
<th>UTS (t)</th>
<th>Cable geometry</th>
<th>Pt (t)</th>
<th>Peak Shear load (t)</th>
<th>Shear Displacement (t)</th>
<th>Peak shear load / UTS (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>MW 10-P</td>
<td>31</td>
<td>70</td>
<td>Un-bulbed</td>
<td>15</td>
<td>68.3</td>
<td>68.2</td>
<td>97.6</td>
</tr>
<tr>
<td>2</td>
<td>MW 10-P</td>
<td>31</td>
<td>70</td>
<td>6 bulbs</td>
<td>0</td>
<td>63.8</td>
<td>62.6</td>
<td>91.1</td>
</tr>
<tr>
<td>3</td>
<td>MW 10-P</td>
<td>31</td>
<td>70</td>
<td>6 bulbs</td>
<td>15</td>
<td>60.4</td>
<td>56.0</td>
<td>86.3</td>
</tr>
<tr>
<td>4</td>
<td>MW9-S</td>
<td>31</td>
<td>62</td>
<td>6 bulbs</td>
<td>0</td>
<td>47.7</td>
<td>43.5</td>
<td>76.9</td>
</tr>
<tr>
<td>5</td>
<td>MW9-S</td>
<td>31</td>
<td>62</td>
<td>6 bulbs</td>
<td>15</td>
<td>43.9</td>
<td>47.4</td>
<td>69.9</td>
</tr>
<tr>
<td>6</td>
<td>MW9-S</td>
<td>31</td>
<td>62</td>
<td>Un-bulbed</td>
<td>15</td>
<td>49.7</td>
<td>41.7</td>
<td>67.3</td>
</tr>
<tr>
<td>7</td>
<td>Secura HGC</td>
<td>31</td>
<td>68</td>
<td>6 bulbs</td>
<td>0</td>
<td>64.7</td>
<td>51.8</td>
<td>95.2</td>
</tr>
<tr>
<td>8</td>
<td>Secura HGC</td>
<td>31</td>
<td>68</td>
<td>6 bulbs</td>
<td>15</td>
<td>55.9</td>
<td>45.9</td>
<td>82.2</td>
</tr>
<tr>
<td>9</td>
<td>SUMO-P</td>
<td>28</td>
<td>65</td>
<td>6 bulbs</td>
<td>15</td>
<td>55.8</td>
<td>71.8</td>
<td>86.8</td>
</tr>
<tr>
<td>10</td>
<td>SUMO-P</td>
<td>28</td>
<td>65</td>
<td>6 bulbs</td>
<td>15</td>
<td>68.4</td>
<td>78.2</td>
<td>106.5</td>
</tr>
<tr>
<td>11</td>
<td>ID-SUMO</td>
<td>28</td>
<td>63</td>
<td>6 bulbs</td>
<td>0</td>
<td>40.4</td>
<td>40.9</td>
<td>73.7</td>
</tr>
<tr>
<td>12</td>
<td>ID-SUMO</td>
<td>28</td>
<td>63</td>
<td>6 bulbs</td>
<td>15</td>
<td>37.4</td>
<td>30.9</td>
<td>59.4</td>
</tr>
<tr>
<td>13</td>
<td>ID-TG</td>
<td>28</td>
<td>60</td>
<td>Un-bulbed</td>
<td>0</td>
<td>44.9</td>
<td>51.3</td>
<td>69.8</td>
</tr>
<tr>
<td>14</td>
<td>ID-TG</td>
<td>28</td>
<td>60</td>
<td>Un-bulbed</td>
<td>15</td>
<td>36.3</td>
<td>30.9</td>
<td>57.6</td>
</tr>
<tr>
<td>15</td>
<td>Superstrand-P</td>
<td>21.7</td>
<td>60</td>
<td>Un-bulbed</td>
<td>15</td>
<td>52.4</td>
<td>90.2</td>
<td>85.7</td>
</tr>
<tr>
<td>16</td>
<td>Garford-P</td>
<td>2 <strong>15</strong></td>
<td>54**</td>
<td>Bulbed</td>
<td>0</td>
<td>44.6</td>
<td>46.8</td>
<td>80.9</td>
</tr>
</tbody>
</table>

**Figure 7 a:** Tensile load / elongation profiles of both MW plain and MW indented 7 mm wires of equal length
COMPARISON OF THE TESTING RESULTS BETWEEN SINGLE AND DOUBLE SHEAR TESTING METHODS

Realistically, the methodology of tendon shear testing should not influence the test outcome, as long as various parameters are the same. Factors of particular interest include:

1) Cable ends anchorage: In the single shear test, reliance is made on securing the optimum cable length encapsulation in the concrete cylinder. The optimum length of encapsulation was found to vary between indented surface and plain/smooth wires. In double shear testing methods the Barrel and Wedge (B&W) system provides positive anchorage irrespective of cable strand wire surface. Cable debonding in single shear testing may occur if the encapsulated cable length is insufficient; resulting in increased cable wire failure mostly in tension rather than the tensile/shear, because of the increased shear load displacement.

2) Competence of concrete medium confinement: Poor medium confinement may result in premature concrete radial cracking causing a reduction in cable stiffness with higher shear load travel. The increased cable shear displacement would cause the cable to fail with an increased number of cable wires failing in tension rather than in tensile/shear. In other words the cable strand shear failure load will be closer to failure in tension rather than is shear. Thus effective confinement of the concrete reduces the chances of radial crack occurrence, with less vertical shear travel. This is clearly observed when testing samples in cylindrical concrete with effective and high torqued steel clamps.

3) Similar situation as in point 2 can also occur in single shear testing of cables and particularly plain cables that have failed with increased debonding, because of the lack of effective encapsulation length. All plain wired cables were found to debond readily in comparison with indented and cables with increased shear displacement with strand wire failure occurring in tensile shear combination as reported by Yang et al., (2018).

4) Pure shear in cable wires occurs when the cable is guillotined, with wires being squeezed with lower shear load as reported by McTyre and Evans (2017). In double shear testing it is impossible to observe cable debonding because of barrel and wedge influence.

5) It should be recognised that a realistic way of evaluating cable debonding per encapsulated length can best be determined by pull-out testing and not by shearing. As various tests demonstrated excessive displacement of a cable during the shearing process makes it behave as if the strand wires fail in tension rather than in shear. This is demonstrated by the fact that most wires fail in tension with snapped surfaces typically characterised by cone and cup failure, as shown in Figure 8.

6) The rate of shear loading should be at least 4 mm /min that will produce consistent results irrespective of the methodology of testing with other factors being the same.
Figure 8: Cross sectional view of debonded Plain MW10. Note the extent of wire failure in tension with failed wires surface being mostly in cone and cup.

CONCLUSIONS

The choice of the method of testing cable bolts in shear is governed by the relevant factors that influence the outcome of the testing, and irrespective of the methodology of testing. Effective medium confinement in shear testing would prevent radial cracking of the concrete and effective confinement of the concrete reduces the chances of radial crack occurrence, with less vertical shear load travel resulting in cable failures closer to shear rather than failure in tensile. Under the same testing conditions, plain cable bolts appears to debond much more readily than indented cable bolts when tested in the single shear testing machine with equal length of encapsulation. Pure shearing of cable bolt wires is possible if the cable strand confinement is strong enough so that the cable bolt is shear in pseudo-guillotined, with the with wires being squeezed surface area with lower shear load. Finally, the use of the single shear test rig method in not appropriate for evaluating true cable debonding for a given encapsulation length, rather the pull-out test method should be used.

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POTASH MINES: RISKS AND CONSEQUENCES OF HAZARDOUS FLOODING

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ABSTRACT

Salt deposits development is always associated with the risk of waterproof stratum failure resulting in hazardous fresh water inflow in mine openings. An intensification of mined bed deformations and sinkhole formation on earth surface at the site of water breakthrough due to salt rocks dissolution are the most common negative consequences of potash mine flooding. On the basis of 3D retrospective mathematical modelling of processes that accompanied First Berezniki Potash mine flooding prerequisites for sinkhole formation on earth surface are established. Range estimation for sinkhole moment of appearance is obtained. Approaches to analysis of intensity of room-and-pillar load-bearing elements degradation due to salt rock dissolution are proposed. Procedure for forecast of earth surface deformation due to mining and dissolution of load-bearing elements of underground mining is developed. Results of geomechanical calculations are used for management decisions on ensuring safety of industrial and civil objects that are situated within boundaries of hazardous area.

KEYWORDS

Mine flooding, Sinkholes, Salt rock dissolution, Mathematical modelling, Deformations, Failure

INTRODUCTION

Potash mining is one of the most actively growing branches of world mining industry. In resent years alone mines were designed, built and put into operation in Russia, Belarus, Canada, Uzbekistan, Turkmenistan and other countries. At the same time salt deposits development is prone to accidents due to waterproof stratum (WPS) failure and inflow of fresh or brackish waters into mine openings. As a rule, this leads to flooding and collapse of mine (Shiman, 1992). Well-known Canadian mining engineer F.Prugger noted “…out of all potash mines that have been commissioned, there are more lost to flooding than operating ones” (Prugger F & Prugger A, 1991).

In 2006 at Verkhnekamskoe potash deposit the largest accident in world practice of water-soluble minerals mining took place: First Berezniki Potash mine (mine BKPRU-1) flooding (Baryakh et al., 2013). Volume of filled mined space was around 80 million m$^3$, duration – over 2 years. The negative consequences were aggravated by the fact that Berezniki town, which is a large industrial center in Perm Krai with population around 150 thousand people, is situated over the flooded mine.

Due to dissolution of salt rocks fresh water breakthrough in stopes leads to intensification of earth surface deformations (Baryakh et al., 2010; Shokin, 1997) up to their realization in dynamic form with sinkholes formation which depth could reach hundreds of meters (Whyatt & Varley, 2008; Rauche, 2000). This creates real threats of destruction of buildings and engineering infrastructure objects situated on earth surface. That makes the prediction of earth surface deformation increase and range estimation of preconditions for sinkhole formation the most important problem.

METHODS

Mined rock mass was considered as viscoelastic plastic continuum. Its alteration in time is mostly defined by rheological deformation of room-and-pillar method load-bearing elements during exploitation and emergency situation – flooding. For analysis of mined rock mass deformation in time including the period of flooding a rheological approach based on mathematical description of graphs of earth surface subsidence increase and using modified transient modules method proposed in (Baryakh & Samodelkina, 2005) was applied.

The forecast of earth surface deformation after hazardous flooding of mine is based on consideration of complex of factors connected with hazardous fresh water breakthrough in mine openings: the rib pillars width decrease and chambers height increase due to salt rock dissolution; additional backfilling due to leaching processes; hydrostatic pressure of the brines after the complete filling of mined space (Baryakh & Samodelkina, 2018).

The forecast of earth surface deformation after hazardous flooding of mine is based on consideration of complex of factors connected with hazardous fresh water breakthrough in mine openings: the rib pillars width decrease and chambers height increase due to salt rock dissolution; additional backfilling due to leaching processes; hydrostatic pressure of the brines after the complete filling of mined space (Baryakh & Samodelkina, 2018).

During the period before the accident earth surface subsidence increase in time $\eta(t)$ is entirely defined by actual mining parameters (curve 1 on Figure 1). From the moment of dissolution beginning $t_d$ until the complete filling of mined space there is an increase of subsidence rate (curve 2). After the complete filling of mine with brines ($t > t_p$) due to hydrostatic pressure an “unloading” of pillars takes place which leads to decrease of earth surface subsidence rate (curve 3). In following period of time both an increase (curve 4) and stabilization of earth surface deformation rate (curve 3) could happen depending on intensity of dissolution.
In geomechanical calculations the influence of brines hydrostatic pressure $p$ in filled mining openings on mined rock mass was introduced according to full drainage scheme, for which the pressure in the process of rock mass deformation remains constant and is equal $p = \gamma_b H$, where $\gamma_b$ — specific brine weight; $H$ — mining depth.

The problem solution was carried out using finite elements method (FEM) (Zienkiewicz, 1971; Fadeev, 1987). Numerical implementation of mathematical modelling in three-dimensional statement was based on semi-analytical FEM procedure that allows to reduce three-dimensional problem to a combination of two-dimensional ones by Fourier-series expansion of displacement vector (Zienkiewicz, 1971).

Analysis of mined rock mass stress-strain state at every time step was carried out using elastoplastic approach. That made it possible to locate plastic deformation areas that from the physical point of view were treated as zones of anthropogenic failure due to shear and tensile cracks appearance (Baryakh & Samodelkina, 2012).

For the strength criterion reflecting three-dimensional stress-strain state of rock mass the following expression was used (Baryakh & Samodelkina, 2017)

$$F = \frac{1}{2}[(\sigma_1 - \sigma_2)^2 + (\sigma_2 - \sigma_3)^2 + (\sigma_3 - \sigma_1)^2] + (\sigma_c - \sigma_t)(\sigma_1 + \sigma_2 + \sigma_3) - \sigma_c \sigma_t \leq 0$$  \hspace{1cm} (1)

where $\sigma_c$ — uniaxial compression strength; $\sigma_t$ — uniaxial tension strength. Main stress values $\sigma_1$, $\sigma_2$, $\sigma_3$ were obtained from characteristic equation solution.

For the estimation of actual dissolution scope mining field was divided in areas with same mining parameters and “residual” height of rib pillars. For every area by varying the decrease of rib pillars width and if necessary — their height, through multivariant modelling the correspondence between calculated and actual earth surface subsidence was secured. It worth mentioning that for conditions of BKPRU-1 mine after it complete flooding the process of sylvinitic beds dissolution was practically finished. Mainly, the following degradation of rib pillars was related to dissolution peculiarities of mined carnallite bed (Baryakh et al., 2010).
Quantitatively the scopes of room-and-pillar method load-bearing elements dissolution were defined as a volume of dissolved salt rock \( V \) per mined area unit \( S \): \( V_e = V/S \). Year average rates of carnallite bed specific volume dissolution are presented on Figure 2. This source information was used as a parametric support for assurance of calculated estimations of earth surface deformation caused by mining and following salt rock dissolution.

**RESULTS**

**Earth surface deformations**

Due to complexity of processes associated with salt rocks dissolution it is appropriate to limit the forecast of earth surface deformation by short-term period of 3–5 years. Two approaches could be used in this case. First is based on an extrapolation of earth surface subsidence graphs at fully undermined area with subsequent construction of creep function for transient deformation modules calculations (Baryakh & Samodelkina, 2012). Second approach uses estimation of rib pillars loading degree increase due to their degradation, roof rock failure and, probably, dissolution of superincumbent carnallite bed.

Calculation of rib pillars loading degree for conditions of flooded mine could be modified as following

\[
C = \xi \frac{\gamma^{(a+b)}H_m}{k_f \sigma_m}
\]

(2)

where \( \xi \) — coefficient that takes into account pillar loading degree alteration due to influence of various mine technical factors (additional load from salt dump pile, bearing pressure, additional pillars and so on); \( \gamma \) — rock volume weight; \( H_m \) — maximal distant from earth surface to pillars top; \( k_f \) — pillars form factor coefficient; \( \sigma_m \) — in-situ rock strength.

Figure 3 shows an example of rib pillar loading degree alteration at carnallite bed calculated according to (2). On the basis of analysis of displacement process development at Verkhnekamskoe potash deposit numerical relation between earth surface subsidence and pillars loading degree that allows parametrical provision for creep function estimation is established.

![Figure 3 - Alteration in time of rib pillars loading degree at carnallite bed](image)

In case of stationary development of dissolution processes (curve 2 on Figure 3) both approaches give similar values of earth surface deformation. At the same time using loading degree looks more appealing provided there is an additional information – geophysical or seismological – on a state of flooded mined space. In particular, on the basis of research data it is possible to indirectly determine a stage of roof mass failure and involving of superincumbent carnallite bed in the process of dissolution. This in turn leads to relatively abrupt increase of rib pillars height and to decrease of their form factor, which according to (2) boosts their loading degree increase (curve 1 on Figure 3) and subsequent acceleration of earth surface subsidence. The mere extrapolation of subsidence increase graphs is hardly able to reflect such effects.

The Figure 4 depicts a fragment of distribution of earth surface subsidence and horizontal strains within residential area of Berezniki town obtained by methods of three-dimensional mathematical modelling. The results presented are used as a basis for safety measures development for civil and industrial objects, systematic survey arrangements and management decisions to minimize negative aftermaths of the accident within the town.
Sinkhole formation preconditions

Sinkhole formation at the site of water breakthrough is a common consequence and accompanies accidents at every potash and salt mine. As a phenomenon sinkhole could be described as a transition from static rock mass deformation, manifested in earth surface subsidence, to dynamic phase – failure. Sinkhole sizes could reach hundreds of meters and their formation presents a real threat to vital activity in the area, leads to considerable financial losses and negative socio-economic and ecological consequences. That defines the question of determining the moment of sinkhole appearance and the conditions for its formation at the site of fresh water inflow in mine openings as a major geomechanical problem directed at the minimization of potash mine flooding negative aftermaths.

In case of WPS integrity violation and appearance of straight-through failure zone stretching from mined beds to water-bearing horizons the formation of channel of overburden fresh water intrusion in mine openings takes place. In the course of time due to upper part of salt stratum dissolution an intensification of earth surface subsidence at a relatively small area is registered. In those cases, displacement trough with high subsidence gradients that increase with time is formed. According to performed geomechanical calculations (Baryakh et al., 2016), those gradients sometimes couldn’t be achieved due to room-and-pillar structural components destruction only and caused by formation of local zones with “weakened” strength and deformation properties in overburden. It is significant that the existence of those “weakened” zones is proved by engineering seismic prospecting and results of engineering-geological well-boring.

On the first stage of mathematical modelling estimation of limit sizes of water conducting channel preceding sinkhole formation was carried out. According to the calculations when the channel radius reaches 5 m and mechanical properties in overburden are decreased by factor of 6 a straight-through anthropogenic failure zone that stretches up to earth surface is formed. For assumed model that means that the preconditions for sinkhole formation appears. Verification of those estimations for conditions of BKPRU-1 mine flooding allows to define the water conducting channel growth rate to be around 2 cm/day. The obtained rate is in range of theoretical (about 20 cm/day) and laboratory (about a centimeter per day) estimations of salt rock dissolution rate.

Taking into consideration the obtained water conducting channel growth rate retrospective mathematical modelling of BKPRU-1 mine hazardous flooding for the moments of time from the beginning of overburden water
inflow into the mine openings (17th of October 2006) up to the moment of sinkhole formation on earth surface (28th of July 2007) was sequentially carried out.

Calculations were held in two stages. At the first stage stress-strain state of undermined rock mass at the moment preceding the beginning of hazardous water inflow was modelled. At that time “weakened” zones in overburden were used to achieve better correspondence between calculated and actual values of earth surface subsidence.

At the second stage a water conducting channel was added into calculation scheme at the site of obtained WPS integrity violation and the process of its lateral growth with 2 cm/day rate due to rock salt dissolution and washing out was modelled. Sizes of “weakened” zones and level of mechanical properties decrease in them at that stage were regulated by actual earth surface subsidence.

According to the obtained results at the moment of overburden water breakthrough (Figure 5 a) straight-through anthropogenic failure zone is registered in WPS, within which a channel of overburden water intrusion in mine openings could be formed. This zone is located at the subsidence trough edge where high subsidence gradients are observed.

![Figure 5 - Results of mathematical modelling of sinkhole formation: October 2006 (a), March 2007 (b), July 2007 (c)](image)

In the course of time (Figure 5 b) the upward propagation of failure zone is observed in overburden. In July 2007 the anthropogenic failure zone already stretches from mined openings up to earth surface (Figure 5 c) which could be interpreted as the possibility of sinkhole formation. At that moment according to the results of mathematical modelling the mechanical properties at the site of maximum subsidence decrease by factor of 6. It seems that that level of “softening” could be used as a quantitative criterion of transition of static deformation into dynamic phase with further sinkhole formation in case a dissolution void of enough capacity exists in WPS.

Initial sizes of sinkhole at the site of fresh water breakthrough into mine openings of BKPRU-1 mine were (Figure 6): crater – 55x80 m, central part (associated with water conducting channel sizes) – 5-10 m, depth – 15 m.

According to the results of modelling the volume of water conducting channel at the moment of sinkhole formation is \( V_0 = 7850 \text{ m}^3 \). Volume of caved rocks with consideration of their decompaction factor \( k_p = 1.1 - 1.2 \) is estimated 27400-29900 \( \text{m}^3 \). This volume is enough for complete filling of channel free space and sinkhole formation on earth surface with depth of 16 – 30 m.

![Figure 6 - Sinkhole at the site of water breakthrough (28.07.2007)](image)
It is worth mentioning that the results of mathematical modelling in whole are in accordance with the place (at the site of the highest subsidence gradient) and sizes (central part lateral size is of 5-10 m with initial sinkhole depth – 30 m) of the sinkhole that appeared at BKPRU-1 mine.

So, according to the calculation results it is possible to predict sinkhole formation when channel radius reaches 5-10 m with average growth rate 2 cm/day. On the basis of those assumptions that are in accordance with the sinkhole observations at the moment of its appearance and with experimentally obtained values of salt rock dissolution rate it is possible to estimate the period of time from the water breakthrough to the sinkhole formation at 250-500 days.

CONCLUSIONS

Synthesized geomechanical model of flooded mine that reflects geological and technical mining conditions, room-and-pillar load bearing elements degradation due to salt rock dissolution, undermined rock mass elastoplastic pattern of deformation and failure in the course of time is constructed.

Based on a correlation analysis between calculated subsidence obtained as a result of mathematical modelling and an actual surveyor observations data an approach to estimation of salt rock dissolution actual scope for specific districts of flooded mine is proposed.

On the basis of three-dimensional mathematical modelling of rock mass stress-strain state alteration due to mining and load bearing elements dissolution a procedure for earth surface subsidence forecast is developed.

The level of overburden rocks mechanical properties decrease (by factor of 6 and more) due to their deformation and typical sizes (5-10 m) of water conducting channel formed in salt stratum due to salt rock dissolution were proposed as a quantitative criteria, determining possibility of sinkhole formation at the site of fresh water breakthrough into mine openings. Taking into account the obtained water conducting channel growth rate (2 cm/day) period of time of sinkhole formation in this case is estimated in 250-500 days.

Obtained results are used as a basis for timely management decisions about security of vital activity of urbanized territories situated over the flooded mine.

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PROBLEMATIC ASPECTS AND WAYS OF DEVELOPMENT OF THE REPEATED UNDERGROUND GEOTECHNOLOGY IN THE CONDITIONS OF CAVED DEPOSITS OF THE ZHEZKAZGAN DEPOSIT

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Mining Institute after D.A. Kunayev.
Abstract. The article analyzes conditions and problematic aspects of development of natural and anthropogenic reserves in caved pillars of the Zhezkazgan field. It contains the review of operating practice at Zhezkazgan underground mines and ways of advancement of repeated geotechnology in the area of caved deposits of the field. Possible options of developing flat and inclined deposits are proposed. Design and process parameters of developing field drilling and haulage workings are substantiated for the draw level and level of ore delivery. Operating procedures for the application of the resource-replenishing technology of repeated underground mining of natural and anthropogenic reserves from the subsoil are developed by the example of the Zhezkazgan field. Technical and economic efficiency of the proposed repeated underground geotechnology was estimated.

Keywords: Rib pillars, overlapping and caved ore deposits, repeated underground geotechnology, design and process parameters, operating procedures, loading, haulage and discharge workings.

Introduction. In recent years, there has been a trend of decreasing productivity and increasing cost of copper production at operating mining companies of Zhezkazgan. This contributed to the reduced efficiency of applied conventional technology of ore mining using the room-and-pillar system and required the transition to advanced mining technologies, involvement of temporarily inactive recoverable and unrecoverable ore reserves into production with the view of prolongation of the field’s operation life.

In this connection, currently the repeated development of these reserves is essential for solving the issues of resource replenishment without additional capital investments and for prolongation of operation life of Zhezkazgan mining companies.

The repeated underground mining of natural and anthropogenic ore reserves from the subsoil is characterized by unfavorable geomechanical and mining technical conditions in caved deposits of the fields. Therefore, one of the urgent issues is the substantiation of process charts and parameters of the mining technical system that ensure its smooth operation at the repeated underground development of copper fields.

The analysis of current state of underground development of Zhezkazgan mines reveals a number of problems, among which are worsening geomechanical and mining technical conditions of Zhezkazgan caved deposits resulting in high cost of ore production.

Thus, development of remaining reserves of the Zhezkazgan field takes place in complicated mining technical and geomechanical conditions. It requires finding effective and industrially safe ore mining geotechnologies ensuring repeated integrated subsoil management.

The peculiarity of geomechanical substantiation of the technology of repeated mining of reserves of the caved area is the new approach to the calculation of strength in the recommended process chart [1-3].

One of the most important factors both from the standpoint of technology and safety of mining operations at stoping is the selection and substantiation of the place of drilling and haulage workings for proper establishment of the discharge haulage level.

The methodology of the proposed technology. It is designed to extract pillars from fringe drifts driven in bed rocks under the sections subject to repeated mining. The section is prepared and developed in close connection with the overall chart of preparation and development of the considered area.

The development involves the reserves of isolated, coupled and massive pillars (which are not destroyed completely) and their parts located under the caved rock mass [4, 5].

The proposed technology with uncontrolled caving in the conditions of the caved area represents a design where the load-haul drift is located at the 20-meter high sublevel at safe depth \(h_m\) in the rock mass at the stope sill along the strike and along the axis of the chamber. The cross section of the drift is 18.6 m\(^2\) (height: 4.5 m and width: 4.3 m) and it connects the 10\(^{\circ}\)-12\(^{\circ}\) inclined drilling and loading drives, the section of which is 13.8 m\(^2\) (height: 3.5 m and width: 4.5 m), and lies at an angle of 45\(^{\circ}\) in two directions to the axis of the load-haul working under each rib pillar.
Then, from each end of the drilling and loading drives along the axis of the rib pillar upwards from both sides of the load-haul working, as a result of drilling of fan holes, funnels are formed up to the contact of the inclined area of the rib pillar for complete caving of rib pillars.

Based on structural elements for the creation of the proposed design of caved ore drawing from the rib pillar, it is necessary to lay the load-haul working at the stope sill in the rock mass with drives under the rib pillar able to bear high loads upwards from both sides of the load-haul working with the formation of funnels up to the contact of the inclined area of the rib pillars.

The blocks are prepared in the workings of the fringe drift by driving the load-haul drifts in the underlying rocks under the rib pillar’s layout (Figure 1). Drilling and hauling drives are developed from the load-haul drift under each pillar (Figure 2).

In the conditions of flat occurrence with overlapping deposits, for the purpose of repeated development of reserves of destroyed rib pillars with filled chamber of the caved mass, the proposed technology of ore drawing after uncontrolled caving in the conditions of the caved area represents the following process chart. The load-haul workings lie at the safe depth \(h_m\) in the rock mass at the stope sill under the chamber (5) located between the rib pillar (6) and continue on both sides of the drive (2) under each rib pillar up to their boundary sidewall contacts. Then, from the drive face, the working is developed upward forming the funnel (4) as a result of drilling the fan holes and re-drilling up to the rib pillar’s roof.

The pillars are broken according to data sheets (projects) drawn up by the mine in line with the Unified Explosive Safety Regulations.

Figure - 1. The proposed design of the caved ore drawing from the rib pillar at the stope sill in the rock mass of inclined overlapping deposits.
Figure 2 - The proposed design of the caved ore drawing from the rib pillar at the stope sill in the rock mass of flat overlapping deposits.

The major processes determining the selection of self-propelled equipment are the drilling and load-haul operations.

The mined rib pillars are drilled off using diesel hydraulic drilling rigs of the Solo DH 07-7S-type by the company Sandvik Mining and Construction (Finland).

The load-haul operations provide for the use of the following complex of self-propelled equipment: Cat-980G LHD machine with a scoop capacity of 5.2 m$^3$ (USA) for loading; TORO-40D dump truck with body capacity of 22 m$^3$ (Finland) for ore delivery.

The methodical basics for the determination of geomechanical parameters of the recommended process chart imply that the load on the support of the horizontal drilling and haulage working is formed due to its rocks’ weight within the natural self-supporting arch. In addition, this allows solving the problem of determining the value or height of the roof arch. One of the hypotheses of calculating the loads on the support is M.M. Protodyakonov’s hypothesis based on the assumption that the natural self-supporting arch is formed over the mine working [6].

The practice shows that the natural self-supporting arch or the roof arch is of an elliptic form. Using simple considerations and calculations, we find that the specific load on the unit of the roof area equals:

$$q_{30} = \frac{\pi \gamma h_c}{4}$$  \hspace{1cm} (1)

Here, $\gamma$ is the volume of the superincumbent rock.

The methodical scheme for the calculation of the natural self-supporting arch $h_{cs}$ does not account for the structural particularities of the mass and the depth of the mine working (drilling and haulage working). We have developed the method for determination of $h_{cs}$ based on the value of the plastic range of stress formed around the drilling and haulage working depending on its depth and fracturing of the mass. Based on the proposed method, the analytical scheme was developed for the calculation of loads on the support of the drilling and haulage working driven in the mass under the caved area. In this case, the average estimated pressure in the sectional plane perpendicular to the mine working’s axis is taken as

$$q = \gamma H k_n h$$  \hspace{1cm} (2)

relation

$$q = \left( \frac{\gamma h_{cs}}{k_{ynz}} - \gamma h_M \right) k_n$$  \hspace{1cm} (3)

Here, $h_{cs}$ is the height of caved rocks;

$h_{ynz}$ is the compaction factor;

$h_M$ is the thickness of the mass under the caved area up to the drilling and haulage working.

By analyzing the above-given equations, it is easy to note that the formation of loads depends on the height of the layer of caved rocks and volume weight of loosened rocks.

The depth of the drilling and haulage workings relative to the roof arch depends on the value of the plastic range of stress in the mass above the drilling and haulage working that forms the natural self-supporting arch $h_{cs}$.

In this case, based on the practice of rib pillar development at the repeated mining of the field, the recommended value $h_M$ is $h_M \geq 6$ m [7].

**Discussion.** Any process chart requires geomechanical substantiation, ensuring the stability of all structural elements, calculation of the form and cross-section of load-bearing elements of the workings, substantiation of selection of the cross-sectional parameters of the workings,
determination of the load on the mine workings, calculation of the load factor of the workings located in the rock mass at the bottom under the rib pillars.

Based on the charts considered above, an important indicator is the calculation of the value of hydraulic radius of the rock mass capability as a MRMR rating according to Prof. D. Laubscher's rating classification [8].

The determination of rock mass rating according to D. Laubscher's rating classification using the proposed development technology of uncontrolled ore caving.

The hydraulic radius is the major indicator of the rock mass capability and is determined using the diagram (Figure 3) as a relation of the exposure area to its perimeter at which the self-destruction process starts. It equals 25 m (Table 1).

Figure 3 – The diagram for determining the MRMR hydraulic radius according to Prof. D. Laubscher's rating classification

Table 2 – Dependence of the hydraulic radius on the MRMR rating

<table>
<thead>
<tr>
<th>Indicators</th>
<th>Class of the rock mass according to MRMR</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>1</td>
</tr>
<tr>
<td>Hydraulic radius</td>
<td>-</td>
</tr>
<tr>
<td>Cavability</td>
<td>-</td>
</tr>
</tbody>
</table>

The considered sections of deposits Ann-3-I top, Ann-3-I bottom, Ann-3-II, Ann-4-I, Ann-4-II of the block 12bis are of irregular forms, the dimensions are: width varies from 0 to 100m and length can be up to 400m. This section of repeated development of deposits of the block 12bis is located in the caved area of Zhezkazgan field where reserves are mined at the depth of 375 – 410m using the room-and-pillar development system within the project.

The proposed technology of repeated mining of the part of block 12bis consists of flat and flexural sections.

The early accepted project ZhezkazganNIIsvetmet for initial mining of reserves of block 12bis Ann-3-I top, Ann-3-I bottom, Ann-3-II, Ann-4-I, Ann-4-II has applied the room-and-pillar development system with descending order and left the rib pillars under protection of barrier pillars with the cleaning-up of the block reserves.
The volume of recoverable reserves for initial mining in this block in the caved conditions amounts to 616.5 thous.t of ore (Table 3) [9]:

Table 3 - The volume of recoverable reserves for initial mining of the block

<table>
<thead>
<tr>
<th>Block</th>
<th>Kind of reserves</th>
<th>Ore, thous.t</th>
<th>%</th>
<th>Metal, t</th>
</tr>
</thead>
<tbody>
<tr>
<td>12 bis 3-II</td>
<td>Recoverable reserves</td>
<td>616.5</td>
<td>0.99</td>
<td>6132</td>
</tr>
</tbody>
</table>

In line with the process chart, with the purpose of designing, preparation and development of the block 12 bis with field preparation, at safe depth in the rock mass there is a load-haul drift at the stope sill along the strike and along the axis under the chambers in two directions toward the axis of the load-haul working with the drive into cutouts and funnel for bottom outlet of caved mass under each rib pillar.

To draw up a local project of the proposed process chart of repeated mining of the block 12 bis, we represent charts of preparation and development of reserves in individual block panels based on the actual mining technical condition after the initial development of the deposits Ann-3-I-в, Ann-3-I-н, Ann-3-II, Ann-4-I, Ann-4-II (Figure 4).

The research was conducted by MI after D.A. Kunayev in the framework of the program “Grant Financing of Scientific Research” in the years 2015-2017 on the subject: “Development of the Method and New Options of Resource-Replenishing Technology of Repeated Underground Mining with Rational and Integrated Mining of Natural and Anthropogenic Reserves”.
Results of the discussion. Based on the conducted research it is possible to state that the geomechanical situation at underground mines of Zhezkazgan has worsened and this resulted in the reduction in efficiency of employed ore mining technology at the transition to inclined deposits and reserves located at peripheral areas of mine fields.

Consideration of systematized remained natural and anthropogenic reserves in the form of rib pillars, barrier pillars and panel pillars revealed that during the last years the productivity had reduced and copper production cost had increased at operating Zhezkazgan mining companies. This contributes to the decrease in efficiency of applied conventional technology of ore mining using the room-and-pillar system and requires the transition to advanced mining technologies, involvement in mining of temporarily inactive recoverable and unrecoverable reserves with the view of prolongation of the field operation life.

Conclusions. Based on the foregoing factors, we have proposed the methodology of development of new options of resource-replenishing technology of repeated underground mining with rational and integrated mining of natural and anthropogenic reserves from the ore fields.

To accept concrete process solutions for repeated mining, it is necessary to test the method of calculation of major processes and parameters of selected development systems.

Efficiency evaluation of the repeated underground development system with uncontrolled caving revealed that the depth of drilling and haulage workings relative to the roof arch depends on the value of plastic range of stress in the rock mass over the drilling and haulage working that forms the natural self-supporting arch.

The evaluation of the strength of load-haul workings and drilling and loading drives situated in the rock mass of the stope sill revealed that the load factor for load-haul workings is \( n_p = 1.9 \); for drilling and loading drives \( n_p = 2.0 \), and this corresponds to their safety factor (at \( n \geq 1.4 \)).

The use of GIS simulation revealed that the general process chart (depth of drifts, angles of ore passes etc.) was selected correctly.

The most optimal for these mining geological conditions is the option of uncontrolled caving system and superincumbent stratum of caved rocks with field preparation of reserves with two-directional drives from the load-haul drift under each rib pillar. The major part of the operating procedure proposes the local project of repeated underground development of caved Annensk area of Zhezkazgan field, and block 12 bis of Annensk East Zhezkazgan mine was selected.

The research resulted in the economic calculation of profit indicators from the product sale, where the profitability within option 1 (State Design Institute ZhezkazganTsvetmet) equaled 2.18 %, within option 2 (MI after D.A.Kunayev) the profitability amounted to 5.0 %, i.e. the profit is 2.3 times higher compared to the option of the State Design Institute ZhezkazganTsvetmet. Due to the decrease in stope development in the proposed option, the saving equals 281 million 500 tenge.

List of references


SOME DIRECTIONS FOR THE DEVELOPMENT OF MINING TECHNOLOGIES ENSURING THEIR EFFICIENCY AND SAFETY

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ABSTRACT

The contact of a person with a massive of subsoil in a state of natural equilibrium, whether on the surface or in the bowels of the earth, triggers forces that must be controlled in order to be able to extract minerals without extraordinary incidents, economically and environmentally. With the deepening of mining operations, the fulfillment of these tasks becomes more and more difficult, more dangerous and more costly. But the mind of a man armed with modern technology and carrying out his activity creatively, with an understanding of the mechanism of the forthcoming difficulties, makes it possible, in the future, to provide the vital activity of society with the necessary raw materials.

KEY WORDS

Safety, cost, labor productivity, losses, dilution, level caving, combined development, electrochemical chlorination.

INTRODUCTION

The use of minerals extracted from the bowels of the earth will be the source of human needs for the foreseeable future. Depletion of reserves, especially ores, the content of metals corresponding to the profitability of production at the current level of technologies for their extraction and processing, deterioration of geological conditions in connection with the increase in the depth of mining operations and other negative factors require the creation of new technologies.

For our state and all other leading countries in the commodity sector of the world, the task number one among the top ten tasks, which President N.A. Nazarbayev said in his Address to the people of Kazakhstan, is absolutely fair: "The world of the XXI century continues to need natural resources, which in the future will have a special place in the development of the global economy and economy of our country. However, it is necessary to critically rethink the organization of the raw materials industry, approaches to the management of natural resources".

METHODS

First of all, the organization of raw materials industries and the management of natural resources in a situation of constant growth in demand for almost all types of minerals, taking into account factors that have a clearly pronounced tendency to increase, it is necessary to create and use new technologies that neutralize emerging problems and even improve the results of the use of raw materials.

At the same time, the tasks of ensuring safety, increasing labor productivity, reducing production costs and saving resources, should be at the forefront, while at the same time economically justifiably reducing losses and dilution during extraction and recovery in the processing stages.

Such a statement of the question is not new in itself, but it acquires decisive importance in the modern interpretation, when the discovery of new deposits and outstripping replenishment of mineral resources is costing subsoil users and the state is increasingly more expensive.

In accordance with the development strategy of “Kazakhstanys Corporation” LLP, we carry out consistent work in this fundamental direction, some of which are presented to the attention of the high-level meeting.

1. Combined development of ore deposits.

The theory and practical application of the combined development of ore deposits have developed as an important and integral part of the complex of mining sciences, the ultimate goal of development of which is the solution of the dual task of using mineral resources. The first direction was born as the accumulation of knowledge...
about the extraction of minerals taking into account the patterns of subsurface development, which included the combined technology of simultaneous or sequential field development by open and underground methods, as well as by two or more fundamentally different technologies.

The second direction, which should be interfaced with the first one, includes the solution of the tasks of a single geotechnological project for the maximum useful use of all geological resources in the mining development zone with the minimum possible environmental impact, while at the same time making it possible to work safely with high labor productivity and with a lower cost of production.

RESULTS

Combined development is an independent way of extracting minerals from subsoil within one field.

The principles of spatial and temporal combination of open, underground and physicochemical methods of ore extraction are taken as the basis for isolating individual schemes of combined deposit.

The principles underlying the classification of varieties of combined development presuppose compliance with the most important features of this method of developing the ore reserves of the deposit, namely: the use of different technologies for mining operations in zones of their mutual geomechanical influence and the availability of a unified field discovery scheme.

The classification that reflects the marked features is shown in Fig. 1.

**Note:** Physicochemical technologies can be present at any scale in each of these types of combined development.

As criteria for the economic efficiency of the options, we choose, for example, the maximum value of IRR for total income based on the algorithm (Figure 2), taking into account the minimization of the environmental impact of technological processes on the indicators of the selected option.
The curves in Fig. 2 abbreviations mean:

- $A_k$ – total production capacity for joint mining with the use of (OMO) and underground (UMO) mining operations;
- NPV – Net present value;
- $T_{pp}$ – The period for which the income from the project is offset by the investment $K$, i.e. Payback Period-PP;
- $I_{pr}$ – Profitability Index, PI – ratio of the amount of positive discounted flows to the sum of negative;
- IRR – Internal Rate of Return;
- $Q_o$ and $Q_u$ – reserves, respectively, for the quarry and underground mine.

Figure 2 - Algorithm for optimizing the ratio of open and underground technologies for combined field development

1. Based on the historically developed situation in the technological support of mining operations, in which the principles of combined development of ore deposits have taken their place, its individual elements, simultaneous or sequential mining by open and underground methods are widely used, it is necessary to streamline and legalize the obligatory universal use of combined development. The implementation of such a decision must be organized on the basis of the following main directions:
1) The principles and methods of combined development should be laid in each project for the establishment of a mining enterprise that has reserves for both open and underground works.

2) In this project, the issues of simultaneity or the sequence of open and the underground mining operations, the opening of the underground part with the use of excavations of the quarry(s), the determination of the maximum possible depth of that part of the quarry in which the extraction will be produced without overburden work, and, finally, the issues of ventilation underground works and use of the worked out space;

3) The project should be accompanied by a separate section in which the possibilities of using methods and methods for the explosion-free destruction of an massif of minerals with bringing it to a transportable state using physicochemical technologies;

4) Research and development of physical and chemical mining technologies as an integral part of combined development are the optimal solution ensuring the efficiency and safety of mining operations.

2. As one of the important components of the combined method of mining of solid minerals, physicochemical technologies based on dissolution, melting, leaching, roasting and other physical and chemical processes that convert solid ore and nonmetallic minerals into a liquid or gaseous state, extract them in this form from rock massifs and transport them to processing points.

Everyone knows the technology of heap leaching of gold from oxidized ores with the use of sodium cyanide solution or the technology of underground borehole leaching with sulfuric acid solution of uranium directly from its occurrence in the depths.

From the point of view of the requirements of the "green economy", safe and complete excavation, it would be economically and environmentally beneficial to extract directly from the depths of many other metals using options of this technology.

For example, in Kazakhstan we thought with our colleagues from the Russian Federation over underground leaching of copper from sulphide deposits. We theoretically substantiate the technology of underground electrochemical chlorination, the parameters of which have been studied in laboratory conditions. Laboratory tests gave the following results: for a duration of 24 hours, we extracted a copper product from 55% to 80% with a mass fraction of copper from 60% to 30%, i.e. having received a ready rich concentrate. In the precipitation stage of zinc, a zinc concentrate with a mass fraction of 50% was obtained when zinc is recovered to about 70%.

The liquid phase (gold-containing solution) after precipitation of copper and zinc was directed to the sorption of gold and silver. Activated carbon was used as the sorbent. A gold-containing coal concentrate with a gold mass fraction of 2 kg/t was obtained with gold recovery in relation to the original tailings of 68%, with a silver mass fraction of 5.1 kg/t when silver was extracted with respect to the original tailings of 43%.

The achieved results are obtained when working in an open cycle. The implementation of the electrochemical chlorination technology with circulation and continuous addition of sodium chloride to restore the required concentration will provide a significant increase in the extraction of components into concentrates by returning to the head the process of solutions with a residual concentration of dissolved metals.

Based on the foregoing, for the implementation of mine hydrometallurgy, a process flow diagram, shown in Figure 3.

The technological scheme includes the preparation of the blasted massif, its electrochemical chlorination with the dissolution of sulphide minerals and precious metals. The productive solution obtained during electrochemical chlorination is directed to oxidation and precipitation of iron, subsequent precipitation of copper and zinc and sorption of precious metals.
As a result of the implementation of the proposed scheme, an iron-containing precipitate, copper and zinc concentrates and a gold-containing product can be obtained. A possible circuit diagram of the apparatus for mine electrochemical chlorination is shown in Figure 4.

The scheme includes the blasted mass in the ore massif 1 in which electrodes 2 of the electrochemical chlorinator are installed to collect a saturated solution 3 with a pump 4.

Figure 3 - Technological scheme of mine electrochemical chlorination of copper and copper-zinc ores with precipitation of iron, copper and zinc and sorption of gold and silver
When voltage is applied to electrodes 2 of the electrochemical chlorinator, the material of the blown mass is polarized, anodic-cathodic electrochemical reactions occur on the surface of electrically conductive sulphides and noble metal particles.

Each electrically conductive particle (piece) becomes a bipolar electrode. Anodic reactions occur on the anodic side of the bipolar particle in NaCl solution with the formation of atomic and gaseous chlorine, hypochloric and hydrochloric acids. All compounds formed on the surface are powerful solvents and actively enter into oxidation reactions of sulphides and noble metals. Due to anodic electrochemical reactions, the solution is saturated with metal ions and complex ions of gold and silver.

To collect a saturated solution in the bottom, a collector of saturated solution 3 with pump 4.

The pump 4 pushes through the pipeline 5 the saturated solution from the shaft to the surface and sends it to the sump 6. For the oxidation of ferrous iron, a special oxidant.

In sump 6 carried out precipitation of iron-containing product.

The deferrized solution from the sump 6 flows into the sump, 7 into which the catholyte obtained in the electrolyzer 12 is supplied to adjust the pH of the solution to 5.5-6.0. In sump 7 carried out the sedimentation of the copper concentrate. The solution from the sump 7 flows into the sump 8, where the catholyte from the electrolyzer 12 is also supplied for adjust the pH of the solution to 8. In sump 8 takes place the sedimentation of the zinc concentrate. To intensify the precipitation of zinc to sump 8 can be fed soda Na$_2$CO$_3$.

The solution from the sump 8 is pumped by pump 9 to the sorption columns 10, in which sorption of gold and silver from the solution takes place. The solution flows through the sorption columns sequentially. From the last sorption column, the spent solution is mixed with the NaCl solution from the solvent tank 11 and with the anolyte from the electrolyzer 12 and sent to the bottom into the irrigators 13, from which the solution enters the mass of the blown mass. The solutions circulate through the shaft electrochlorination unit until dissolution of copper and zinc sulphides, gold and silver occurs. At the end of the electrochlorination process, the current is disconnected from the electrodes 2, the dewatering of the blown mass, the separation from the sumps 6,7,8 of obtained products and the discharge from the sorption columns 10 of the sorbent, rich in gold and silver. The obtained products of electrochemical chlorination are sent for further processing. The spent blasted mass can be flooded with a cement solution and play the role of a bookmark of the worked out space.
The proposed technology of mine hydrometallurgy is a fundamentally new and promising direction of mining metals from ores. It requires a large amount of experimental research to develop effective technological solutions, optimize process regimes, pilot industrial and industrial development.

3. An illustrative example of the application of new technological solutions based on the implementation in production conditions are the technologies of pre-concentration of mined ore prior to its shipment to the beneficiation.

Our proposed XRF separators have established themselves as a reliable and efficient technological equipment. This equipment, created by JSC “Technogen” (RF), when sold on copper deposits allows:

- expand raw materials base of mining enterprises of Kazakhstan;
- reduce transport costs by 25-30%, increase the content of copper, gold and silver in the ore of origin, supplied to the beneficiation plant by 15-25%;
- reduce material, energy and labor costs for enrichment by 12-17%, while simultaneously increasing the extraction of metals by 2-7%.

Based on the conducted pilot-industrial tests on the enrichment of copper-zinc and copper ore deposits of “Kazakhmys Corporation” LLP using the X-ray radiometric separation (XRRS) method for the underground mining of technological regulations and the design of ore-sorting complexes, we recommend the principle technological scheme of preliminary enrichment at the ore preparation stage of mineral raw materials (Figure 5.).

![Figure 5 - Principal technological scheme of enrichment](image-url)
For copper and copper-zinc ore of each particular deposit at the stage of the process regulation, equipment will be selected, its quantity determined, and a preliminary estimate of capital and operating costs.

Table 1 presents the results of pilot industrial tests on the enrichment of copper and copper-zinc ores of a number of fields of “KazakhmysCorporation” LLP.

Table 1 - The results of pilot industrial tests on the enrichment of copper and copper-zinc ores of a number of fields of “KazakhmysCorporation” LLP.

<table>
<thead>
<tr>
<th>Field</th>
<th>Separation products</th>
<th>Output, %</th>
<th>Content, %</th>
<th>Extraction, %</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>Cu</td>
<td>Zn</td>
</tr>
<tr>
<td>№1</td>
<td>An enriched product</td>
<td>74,1</td>
<td>2.05</td>
<td>3.18</td>
</tr>
<tr>
<td></td>
<td>Separation tails</td>
<td>25.9</td>
<td>0.28</td>
<td>0.24</td>
</tr>
<tr>
<td></td>
<td>An initial ore</td>
<td>100,0</td>
<td>1.59</td>
<td>2.42</td>
</tr>
<tr>
<td>№2</td>
<td>An enriched product</td>
<td>72.0</td>
<td>1.90</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>Separation tails</td>
<td>28.0</td>
<td>0.22</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>An initial ore</td>
<td>100.0</td>
<td>1.43</td>
<td>-</td>
</tr>
<tr>
<td>№3</td>
<td>An enriched product</td>
<td>63.2</td>
<td>3.08</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>Separation tails</td>
<td>36.8</td>
<td>0.23</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>An initial ore</td>
<td>100.0</td>
<td>2.03</td>
<td>-</td>
</tr>
<tr>
<td>№4</td>
<td>An enriched product</td>
<td>73.7</td>
<td>1.94</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>Separation tails</td>
<td>26.2</td>
<td>0.20</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>An initial ore</td>
<td>100.0</td>
<td>1.48</td>
<td>-</td>
</tr>
<tr>
<td>№5</td>
<td>An enriched product</td>
<td>68.0</td>
<td>0.78</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>Separation tails</td>
<td>32.0</td>
<td>0.15</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>An initial ore</td>
<td>100.0</td>
<td>0.57</td>
<td>-</td>
</tr>
</tbody>
</table>

In the implementation of the underground mining new technologies in the production of the requirements of the "green economy", we entered the boundaries of further improvement of the pre-enrichment with the use of X-ray radiometric separation (XRRS). The problem was that with the use of the XRRS, ore fraction of +30 m was subjected to preliminary enrichment, and the fraction -30 mm left after the screening was combined with the concentrate obtained as a result of the XRRS and, naturally, depleted it a little. We carried out an additional separation of the remaining class of -30 mm on the newly created separators, where a product with a copper content of 12-15% was obtained with 20-25% recovery, i.e. The obtained concentrate can be sent to metallurgical processing, bypassing the concentrating.

The test results give confidence in the creation of an additional resource to reduce the specific consumption of materials and energy, with the possible exception, in some cases, from the processing technology of the concentration process for obtaining concentrate from the fraction -30mm.

4. The underground mining systems used in ore mining are also not structures that cannot and do not need to create new ones. For example, we particularly seriously consider a development system such as a subterranean cavity with caving, the use of which involves breaking the ore with vertical fans of wells with an end outlet, now widely used throughout the world.

With all its obvious advantages, starting from high labor productivity and the possibility of applying under the most diverse geological and mining conditions, it also has its drawbacks. The main disadvantage is the dilution of the caving rock, which is mixed with the broken ore, starting with the breakdown of the first fans of the wells, coming from both the level caving and the sides of the hanging and recumbent sides. A deterrent that prevents the continuity of the production process is the need for cyclical breakage of ore on the principle of "breaking-off-breaking". It should be noted that the breakage of ore "in the clamp" also harbors negative surprises, including difficulties with airing after a mass explosion.
Due to the above disadvantages, we have developed a new version of this development system that combines the structural elements of both well boring, but to the entire height of the floor, and the creation of a "chipped ore shop", which is formed due to the breakage of the ore by horizontal fans of the wells, and playing simultaneously the role of an element that supports the sides of the camera from premature collapse.

Figure 6 - Development systems with a level extraction with breakage of horizontal (slightly inclined) fans of wells and subsequent cavity

The preliminary conditional name of the version developed by us sounds like this: "Development systems with a level cavity with breakage of horizontal (slightly inclined) fans of wells and subsequent cavity" (Fig. 6). Release of the chopped ore from the block for loading and delivery is carried out through the system of outlet funnels from the ore gangway or the mouth, depending on the thickness of the ore body. For the preparation of the chamber, it is necessary to pass parallel level delivery ore and transport field drifts (orts) and congresses between them, parallel to them to a height, ensuring ore release taking into account the angle of the natural slope, the cut-off line. Outlet holes with their subsequent spreading pass in a staggered order from the ore gangway in both directions to the drift of the crossing.

The cut-off or cut-off drift is extended to the area of the breakout chamber to create a compensation space, and the mined ore is magazined.

Prior to the preparation from the lower floor to the upper one, using the monorail tunneling complex KPU or Alimak, the boroventional uprising is passed in the lying side in the alignment of the ore floor drift, from which the fans of the wells of the estimated diameter, length, distance between the ends of the wells and the line of the least resistance.

The fan of the wells after the compensation space is designed and the part of the broken ore is released, taking into account the loosening factor, is charged and exploded, thus discouraging ore to the compensation space created in accordance with the calculated drilling and blasting certificate.
The length of the wells is determined based on the quality of the blasting operations and taking into account the minimization of well deviation under the influence of the own weight of the drill string and the centrifugal force of the drilling torque.

The estimated length of the well determines the area of the chamber, which is also limited by the thickness of the ore body.

The advantages of this version of the underground mining system are:
- high labor productivity and almost non-stop work to produce chipped ore from the block;
- production costs at the lowest comparative level;
- reducing dilution and loss due to the complete absence of leakage of the collapsed rock during the breakage of the chamber (the rock enters the ore only after connecting the chamber to the collapsed upper chamber), and also due to the constant backing of the ore in the store on the hanging and recumbent side;
- saving on the volume of tunneling works, because there are no costs for the penetration of sub-stages (ore and rock) and slopes with congresses for each sub-floor to the entire height of the floor;
- improvement of ventilation conditions in the compartment during the work due to general depression without using local ventilation.

DISCUSSION

Dear colleagues! The contact of a person with a massif of subsoil in a state of natural equilibrium, whether on the surface or in the bowels of the earth, triggers forces that must be controlled in order to be able to extract minerals without extraordinary incidents, economically and environmentally.

With the deepening of mining operations, the fulfillment of these tasks becomes more and more difficult, more dangerous and more costly. But the mind of a man armed with modern technology and carrying out his activity creatively, with an understanding of the mechanism of the forthcoming difficulties, makes it possible in the future to provide the vital activity of society with the necessary raw materials.

CONCLUSIONS

Due to the above disadvantages, we have developed a new version of this development system that combines the structural elements of both well boring, but to the entire height of the floor, and the creation of a "chipped ore shop", which is formed due to the breakage of the ore by horizontal fans of the wells, and playing simultaneously the role of an element that supports the sides of the camera from premature collapse.

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THE STRATEGY OF COAL MINING DEVELOPMENT OF DTEK ENERGY MINES AS FOUNDATION OF UKRAINE ENERGY BASE

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ABSTRACT

The analysis of the current state of Ukraine coal mining industry, as well as the electricity consumption and production in the country, are given in the articles. The production results of DTEK Energy mines are shown and the priority directions for their further development are formulated.

KEYWORDS: DTEK Energy, coal production, electricity output, production results.

INTRODUCTION

Currently, the coal mining industry of Ukraine is in the conditions of rapid changes in the external environment and constant challenges. Inside the country, destabilization in the east of Ukraine is affecting. Military operations in the east of Ukraine have complicated the work of a significant part of domestic enterprises. In the world energy arena, hydrocarbons are gradually displaced and replaced by renewable sources of energy. Even the term “decarbonization” has appeared, as a tendency to avoid traditional types of thermal generation using coal, oil and natural gas. In these conditions, it is important to ensure the effective functioning of DTEK Energy mines to preserve Ukraine energy base.

WORLD TENDENCIES IN COAL MINING DEVELOPMENT

Despite the current trends in the maximum displacement of hydrocarbons in the energy sector, coal production in the world remains significant. Annually, in the world produces 7.68 bill t of coal (EIA, 2017). At the same time, it is expected that the volume of world coal production will increase to 9.23 bill t by 2040, whereof 64% will be provided by the main producing countries as China, India, and Australia. China share in world coal production will decrease from 47% (2016) to 44% in 2040. Stable growth in coal production is projected in India (an average +100 mill t for every five years). The forecast of world coal production in major coal mining countries and in the world is shown in Figure 1.

Figure 1 – The forecast of world coal production in major coal mining countries and in the world to 2040

Similar conclusions can be drawn about the cost of energy commodities. In the medium term, world prices for thermal coal will grow to 112 USD per ton (6000 kcal). Previously, analysts predicted that after the “bottom” in 2015, prices for thermal coal stabilized at 65 USD. However, as of today prices for coal have risen to 106 USD. The above data, in general, show the presence of a positive external background for the development of the coal industry.

ENERGY RESOURCE AND ELECTRICITY PRODUCTION IN UKRAINE

Most countries have own coal reserves. However, 90% of these reserves are buried on the territory of 10th countries. Ukraine is among them and takes 7th place (34 bill t) in the world by coal reserves (Baker Tilly, 2018).

For Ukraine, coal is the only natural raw material which potential is sufficient to ensure its energy security. In the structure of world fossil fuel reserves coal has 67%. In Ukraine, it is 95%. At the same time, Ukraine has hostile climatic conditions, a relatively small extent of the seacoast, and developed heavy industry. In this situation, the potential of alternative energy is small, and thermal generation along with nuclear power will occupy the main place in the internal energy balance in the near future (Snihur, 2016).

In Ukraine, thermal power plants (TPPs) produce 31% and nuclear power plants (NNPs) generate 55% of domestic electricity. The share of electricity generation from alternative sources as wind power stations (WPSs), solar
power stations (SPSs), and biomass is up to 1% (UA Energy, 2017). The structure of energy resources and electricity production in Ukraine is shown in Figure 2.

Ukraine electricity production was 154.5 bill kWh in 2017, whereof 80.9 bill kWh produced at NPPs and almost 50 bill kWh at TPPs. Over the past 4 years, Ukraine power industry has been experiencing a tendency to reduce electricity consumption (-19%), which is explained by the shutdown of a number of TPPs (reequipment of power generating units on gas-coals due to loss of control on anthracite mines), an increase in the energy market debt to energy generating companies (DTEK annual report, 2014). However, even despite the reduction in electricity consumption the anthracite coal deficit was 4.5 mill t for the past year.

THE ANALYSIS OF UKRAINE COAL INDUSTRY

Ukraine has 63 mines whereof 30 are private and 33 are state mines. The regional distribution of mines is as follows: Lviv – 9 mines (8 – state and one private); Volyn – 4 state mines; Dnipro – 10 mines of DTEK Energy, Donetsk – 24 mines (15 – state, 9 – private mines, Lugansk – 16 mines (6 – state, 10 private) (DTEK annual report, 2016).

The mines of the Donetsk region for the past year produced 11438 thou t of coal (-26.9% by 2016), Lugansk – 1689 thou t (-66.5%), Dnipro – 20142 thou t (+9.4%), Lviv – 1545 thou t (-3.1%), Volyn – 101 t (-45.4%) (Ministry of Energy and Coal Industry of Ukraine, 2018) The dynamics of coal production and consumption in Ukraine is shown in Figure 3.

In 2017, the total volume of produced coal in Ukraine was 34.9 mill t (-14.4% compared to 2016). Coal consumption remains at the level of 30 mill t per year. In 2017, DTEK announced the loss of control over Sverdlovanthracite, Rovenkyantracite and Komsomolets Donbassa due to military activities at the East of Ukraine. These mine administrations produced about 17 – 19 mill t of coal annually.
THE ANALYSIS OF PRODUCTION RESULTS OF DTEK ENERGY MINES

The mines of DTEK Energy are the flagship of Ukraine coal industry. They produce 71% of coal in Ukraine. In the total production structure, DTEK mines make 24.8 mill t whereof 22.9 mill is steam coal (+8.2% compared to 2016, the highest rate in the company history by the results of the year), and 1.9% is anthracite (-76.8% to 2016). The dynamics of coal production in DTEK mines in the general structure of Ukraine coal mining is shown in Figure 4.

![Figure 4](image-url)

Thanks to the efforts of Pavlohradvuhillia and Dobropilliavuhillia DTEK enterprises were able to increase extraction of gas-coal types, and therefore TPPs were fully provided by fuel and could bear in 1.5 times more output than in normally. In the future, the company will continue to increase the extraction gas-coals, first due to the expansion of investment program of Dobropilliavuhillia. The company will allot 150 mill USD for the enterprise development until 2020.

In 2017, DTEK Pavlohradvuhillia mines have produced 20.1 mill t. Coal production of Dobropilliavuhillia mines has composed 2.8 mill t, and by 2030 it is planned to increase it to 5.8 mill t by introducing new longwall faces on Almazna, Novodonetska, and Bilozerska mines.

THE PROJECTS OF TECHNOLOGICAL DEVELOPMENT OF DTEK ENERGY MINES

The project “concentration of mining operations”. To ensure an economically acceptable level of production in the DTEK mines, in the recent years a thorough modernization of the reserves preparation schemes for the excavation has been carried out, and a complete transition to the use of combined systems of development of longwall faces with reuse of mine working has been conducted. The panel length has increased from 1140 to 1600 m, the panel width has risen from 200 to 245 m, and longwall panel has expended by 74%. The average daily coal output per face has increased by 76%, while the number of longwall installation decreased from 34 to 26 (-23%).

In general, the trend of increasing the parameters of the excavation panels of operating longwall faces in the mines of DTEK allowed to balance the process of preparation and working out the coal faces taking into account the available technical and financial capabilities, which ensured the achievement of the planned DTEK development strategy of technical and economic figures of mining.

The project “reuse of mining workings”. The technology of reuse of excavations is the basis for the efficiency and competitiveness of the work of DTEK mines. Thus, only at the mines of Pavlohradvuhillia, up to 36 new longwalls are introducing annually to maintain 26 operating ones. A similar situation is typical for other DTEK coal mining enterprises.
The project “Technology of selective very thin seam coal mining”. The National Mining University together with DTEK company have developed the technology of selective very thin seam coal mining with goaf backfilling to reduce the ash content of mined coal and decrease the volume of rock hoisting from mines (Bondarenko, 2014; Byzylo, 2015). The technology provides the coal seam mining a thickness of 0.55 – 0.80 m, the subsequent excavation of the wall rock undercut a thickness of 0.40 – 0.65 m and worked-out space backfilling. The technological scheme of selective very thin coal seam mining with goaf backfilling is shown in Figure 5.

Figure – 5 The technological scheme of selective very thin coal seam mining with goaf backfilling: 1 – coal seam; 2 – wall rock undercut; 3 – shearer; 4 – mechanized roof support; 5 – reverse cantilever of mechanized support; 6 and 7 – corresponding face and backfilling lines of horizontally-closed scraper conveyor

The process of coal extraction is carried out in the following way: in the initial position the support and the conveyor stand close to the face, the section of the mechanized support are unfastened, the conveyor drive heads are shift, the shearer is cut into the coal seam at the ventilation drift. The forward drum is installed in the upper position and maintained by a seam roof, the back drum is maintained on the extraction of leaving rock bench. When the shearer moves from ventilation to conveyor drift the coal is removed without wall rock undercut. When the shearer moves in opposite direction the section of the support are alternately moved at the same time the face and backfilled conveyor lines are not moved. The position of longwall equipment while coal extraction is shown in Figure 6.

Figure – 6 The position of longwall equipment while coal extraction

The process of rock extraction and backfilling it in the worked-out space is realized as follows: after the extraction of coal and its transportation to the conveyor drift, the shearer is reversed. Further, the rock bench is extracted in the direction from the conveyor to the ventilation drift. The rock from the face is transported to the conveyor drift, and then it goes around the drift and, in the opposite direction, enters on the backfilling conveyor line of the horizontally closed scraper conveyor.

The backfilling line of the conveyor is installed at an angle to the seam floor plane. As the crushed rock moves along the inclined pans, it is self-unloading into the worked-out space with its subsequent packing by a tamping device. The laying of the backfilled massif is carried out continuously along the length of the worked out space. The position of longwall equipment while wall rock extraction and goaf backfilling is shown in Figure 7.
The main technical and economic indexes of the selective coal mining technology were carried out for the conditions of mining 861 longwall C8t seam of Zakhidno-Donbaska mine (Table 1). At the same time, annual coal output per longwall face is expected to be 400 thou t while 21% of ash content (compare to traditional technology 46 – 48%).

<table>
<thead>
<tr>
<th>Parameter name</th>
<th>Indexes</th>
<th>Parameter name</th>
<th>Indexes</th>
</tr>
</thead>
<tbody>
<tr>
<td>Seam thickness</td>
<td>0.8 m</td>
<td>CAPEX</td>
<td>57.48 mill EUR</td>
</tr>
<tr>
<td>Annual coal production</td>
<td>400 thou t</td>
<td>Working capital</td>
<td>6.18 mill EUR</td>
</tr>
<tr>
<td>Daily coal output per face</td>
<td>1200 t</td>
<td>IC</td>
<td>71.5 mill EUR</td>
</tr>
<tr>
<td>Coal ash content</td>
<td>21%</td>
<td>NPV</td>
<td>58.6 mill EUR</td>
</tr>
<tr>
<td>Revenues</td>
<td>17.01 mill EUR</td>
<td>PLC</td>
<td>10 years</td>
</tr>
<tr>
<td>EBITDA</td>
<td>11.73 mill EUR</td>
<td>DPP</td>
<td>4.3 years</td>
</tr>
<tr>
<td>EBIT</td>
<td>9.46 mill EUR</td>
<td>IRR</td>
<td>15.6%</td>
</tr>
<tr>
<td>Net profit</td>
<td>7.78 mill EUR</td>
<td>ROI</td>
<td>15%</td>
</tr>
</tbody>
</table>

The project “Technology of auger coal extraction”. Coal reserves in the protective pillars of DTEK Pavlohradvuhillia mines are 12.3 mill t. For their development, the technology of auger extraction by the KSB complex is proposed. The general view of the KSB auger complex is shown in Figure 8 and the main technical parameters are given in Table 2.
Table 2 – The main technical parameters of KSB auger complex

<table>
<thead>
<tr>
<th>Parameter name</th>
<th>Indicator</th>
</tr>
</thead>
<tbody>
<tr>
<td>Technical productivity, t/min</td>
<td>2.0</td>
</tr>
<tr>
<td>The diameter of drilling bit, mm</td>
<td>625; 750</td>
</tr>
<tr>
<td>Well width, mm</td>
<td>1905</td>
</tr>
<tr>
<td>Well length, m</td>
<td>100 – 120</td>
</tr>
<tr>
<td>Motor power, kW</td>
<td>2×355 kW</td>
</tr>
<tr>
<td>Forward pressure on cutting member, kN</td>
<td>600</td>
</tr>
<tr>
<td>Productivity while coal seam extraction with a thickness 0.6 – 0.9 м, t/day</td>
<td>400 – 600</td>
</tr>
</tbody>
</table>

The KSB auger complex works according to the following technological scheme. The technological cycle in the extraction of coal is composed of a series of parallel and sequential operations. The visualization of the coal extraction process is shown in Fig. 9.

At the same time, two auger machines are working simultaneously in the KSB complex. The distance between the machines is set equal to 12150 mm. The first machine drills a well using auger bit from a previously drilled well. The second machine removes the auger, leaving the cutting member on it. The machine with the auger bit moves on 2430 mm and starts the drilling process using a screw auger removed from a previously drilled well. The cycle is repeated four times. The machine with cutting, which is ready for drilling, moves on 12150 mm.

The auger machines are operated alternately (one provides drilling, the other removing auger bit from the well, then the functions are reversed). Transportation of extracted coal is carried out by scraper conveyors, then by belt conveyor or rail transport, depending on the technological scheme of coal delivery accepted at the mine.

The main advantage of the KSB auger complex and its analogs is the exclusion of the intermediate operation of storage auger bit elements. This ensures continuity of drilling and increases the productivity of the complex. The main technical and economic indexes of the auger technology are carried out for the conditions of DTEK Pavlohradska mine and are presented in Table 3.

Table 3 The main technical and economic indexes of auger technology

<table>
<thead>
<tr>
<th>Parameter name</th>
<th>Indicators</th>
<th>Parameter name</th>
<th>Indicators</th>
</tr>
</thead>
<tbody>
<tr>
<td>EBITDA</td>
<td>83,57 mill EUR</td>
<td>NPV</td>
<td>235,1 mill EUR</td>
</tr>
<tr>
<td>EBIT</td>
<td>82,68 mill EUR</td>
<td>PLC</td>
<td>5 year</td>
</tr>
<tr>
<td>Net profit</td>
<td>67,27 mill EUR</td>
<td>DPP</td>
<td>0,42 year</td>
</tr>
<tr>
<td>CAPEX</td>
<td>15,36 mill EUR</td>
<td>IRR</td>
<td>208%</td>
</tr>
<tr>
<td>Working capital</td>
<td>26,14 mill EUR</td>
<td>ROI</td>
<td>15%</td>
</tr>
</tbody>
</table>
DEVELOPMENT OF GREEN ENERGY

The structure of production and consumption of electricity undergoes radical changes in the world. Ukraine can’t remain aloof from these processes. Many countries are trying to completely switch to electricity from renewable sources. In the future, this trend will only increase. Therefore, DTEK is actively developing green energy as one of the most promising directions. In 2017, DTEK wind power stations generate 637 mill kWh of electricity. This is on 4.8% more than in 2016. Currently, the total capacity of the wind farm of DTEK is 257 MW. It is on 53% more than in 2016.

DTEK has started construction of the first stage of the Prymorskyi wind park with the capacity of 200 MW in the Zaporizhzhia region. General Electric in 2018 will supply and install 26 wind turbines with a capacity of 3.8 MW each. Also in 2018, the construction of a solar power plant with a capacity of 200 MW in the Dnipropetrovsk region is planned.

By 2020, DTEK total renewable energy capacity will be 1,300 MW. According to the Energy Strategy and international obligations, the installed renewable energy capacities in Ukraine should be 5 GW by 2020. This is in 3.5 times more than nowadays.

CONCLUSIONS

1. In the near future, the role of coal in power generation will remain significant. Ukraine produces more than 30% of its electricity from coal. This means that annually, TPPs need about 30 mill t, while gas-coal types become highly-demanded. DTEK mines produced 24.8 mill t or 71% of total Ukraine coal production in 2017.
2. The panel length has increased from 1140 to 1600 m (+42%), the panel width has risen from 200 m to 245 m (+ 23%), the longwall panel square has expanded to 76%, and the number of longwall faces has decreased from 36 to 26 or less on 13% for the last years. The technologies of reused mine workings with an application of anchor systems are applied widespread. The total volume of reused mine workings has reached 77%.
3. The project of development technological model of selective coal seam mining has been completed. The auger coal pillar extraction technology is being implemented to introduce at Pavlohradska mine.
4. The company is actively implementing projects in renewable energy. The production capacity of DTEK wind farms has been doubled from 121 MW to 257 MW over the past year. Total generation of electricity from renewable sources amounted to 637.8 mill kWh in 2017.
5. By 2020, the electricity generation from renewable energy sources will consist 11% of the total generation in Ukraine.

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NATURAL HAZARD CONDITIONS RESULTING IN MAJOR ACCIDENTS IN THE COAL-MINING SECTOR IN POLAND

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ABSTRACT

Underground mining operations undertaken by the extractive sector in Poland are threatened by nearly all potential natural hazards. Despite the use of a variety of preventive measures and control strategies, the scale and intensity of hazardous incidents registered in mine excavations is high, leading to a number of accidents, some of them involving the loss of life.

This paper summarises the key aspects of mining activities having relevance to mines in the Upper Silesia Coal Basin, which have a direct impact on intensity and scale of mining hazards. The scale and intensity of potential hazards, including methane emissions, self-heating of coal seams, seismic activity of the rock strata, adverse climatic conditions and water inflows and inrush is investigated alongside the study of accident rates and fatality statistics registered in the last ten years (2008-2017), fully evidencing their negative impacts on safety features in the Polish mining sector.

KEYWORDS

Underground mining, natural hazards, work safety in the coal mining sector

INTRODUCTION

Providing safe working conditions should be a top priority for employers, as a vital aspect of effective management of their business operations. Due to the specific conditions in the coal-mining sector in Poland, the safety features in currently operated mines are affected by the natural hazard manifestations, such as: serious accidents (methane emissions, coal dust explosions, endogenous fires, rockbursts, rock and gas outbursts) and/or non-catastrophic hazardous conditions (dust emissions, adverse climatic conditions, radiation). Hard coal mined in a majority of Polish collieries within the Upper Silesia Coal Basin is deposited in most complex geological and mining settings. Coal is mined in multiple bed operations at increasing depths (also by sub-level mining), in geologically and tectonically disturbed areas, in zones affected by old excavations, in residual sections of the coal deposits (including previously mined rocks) and in regions where the rock strata are rich in methane. Most coalbeds are now mined by the longwall system with caving-in, which guarantees high concentration of coal production due to full mechanisation of processes (cutting, loading, hauling, transport). These as well as other conditions have a bearing on the scale and intensity of natural hazards and the occurrence of catastrophic events, particularly methane emissions, coal dust explosions, rockbursts, rock and gas outbursts (Fig. 1). One has to bear in mind that the distinctive feature of the mining sector in Poland is the coincidence of several natural hazards which tend to occur simultaneously, which further increases the number of dangerous incidents during the mining operations (Burtan, 2016; Kabiesz, 2016).

GEOLOGICAL AND MINING CONDITIONS TRIGGERING THE NATURAL HAZARDS IN POLISH COLLIERIES

Depth of the coal deposit

Mining operations in the collieries within the Upper Silesia Coal Basin have been continued at increasing depths. Presently coal is being mined at the depths below 1200 m whilst in 6 collieries coal...
mining is continued below 1000 m. The average mining depth now approaches 770 m and tends to increase at the rate of about 10.9 yds per year (Patyńska, 2017).

Coal mining at increasing depths leads to temperature and stress increase in the rock strata, whilst the strength of the surrounding rock strata is enhanced and methane contents tends to rise. For example, at 1000 m the rock temperature in collieries within the JSW Mining Corporation can be as high as 50 °C, vertical stresses approach 25 MPa whilst the methane contents at certain points will exceed 10 m³/Mg. Not only will elevated temperatures of the rock strata, including that of undisturbed coal body, produce adverse climatic conditions, but will accelerate the rate of coal self-heating as well, thus increasing the risk of an endogenous fire.

![Figure 1 – Factors triggering the natural hazard conditions in the coal mining sector](image)

At increased mining depth, the strength parameters of rock strata surrounding the coalbeds being mined will be enhanced, which can be attributed to gravity-induced compaction of rock material (Goszcz, 1999). Large values of stress components in coalbeds and surrounding rock strata (stress concentration zones), particularly those registered in strata featuring high compression or tensile strength, are responsible for seismic hazard conditions in collieries, and are likely to trigger the rockburst occurrence (de-stressing behaviour).

Obviously, the depth-related methane bearing capacity and release of large amounts of free methane under elevated pressures at increased depths further adds to the methane hazard being present there (in the context of a potential explosion), and can trigger the occurrence of a rock and gas outburst. The scale of the rock and gas outburst hazard is adversely affected by declining strength of rock strata with depth, registered in a majority of coalbeds; another negative factor being the lower proportion of fissures in the coal/rock structure reducing their gas permeability, at the same time pressure increase will prompt the release of significant amounts of methane in its free state (Konopko, 2013).

**Structure of rock strata and geo-mechanical properties of the rock medium**

Most coal in Polish collieries is mined from Carbon deposits which developed as silts- mudstone and sandstone formation combined with coal seams. In the case of most coal ranks and types, their strength tends to decrease with depth while resistance of the surrounding rock strata tends to increase irregularly, this increase being more apparent in sandstones than in silts (Patyńska, 2017). Furthermore, the thick-layer structure of the rock strata at greater depths is manifested via the presence of thick and compact (highly resistant) strata, including those abutting on several- meter thick highly-resistant roof strata of sandstone or silts mudstone capable of accumulating elastic energy, and therefore often referred to as burst-prone. Mining thick seams leads to more serious deformation of overlying strata (due to undercutting) which tend to break thus triggering the occurrence of high-energy seismic events. That in turn is the major determinant of the scale of rockburst hazard in the area. On the other hand, when thick beds are being mined whilst coal has to be left in the roof strata or in coal shelves in the floor strata (in the case of multi-layer exploitation), the risk of an endogenous fire in old excavations tends to increase.

Characteristic properties of coalbeds being mined and overlying and underlying rock strata in collieries within the Upper Silesia Coal Basin include their methane contents, volatile matter contents,
tendency to self-heating, moisture contents, sorption capacity, gas permeability, sparking tendency or water bearing capacity, will either trigger or control the scale of natural hazards: methane emissions, coal dust explosions, fires, rock and gas outbursts, water conditions. Mechanical properties of rock, such as strength parameters, compactness and elasticity determine the scale of natural hazards due to dynamic or gas-geodynamic behaviour of strata (rockbursts, rock and gas outbursts).

**Geological disturbances**

Coal mining operations continued in geologically disturbed strata are fraught with difficulties whilst the presence of sediments and seismic disturbances create the most adverse conditions since they give rise to negative changes in the state of stress in rock strata alongside the elevated risk of rockbursts and rock and gas outbursts. In certain circumstances they can further enhance the risk of methane emissions (Krause & Dziurzyński, 2015). Sediments typically occur in the form of thinning, leaching, and decay of coal deposits, also revealed by anomalies in coalbed thickness or distance between the coalbeds. Obviously, they determine the choice of the mining method or parameters of the mining systems and its adaptations, in certain cases make it necessary to leave unmined coal residues (for specified or unspecified periods of time), which can give rise to or reveal hitherto undetected fire and rockburst hazard conditions. In most cases the necessity to leave coal residues is due to the scale of geological disturbances, particularly the faults with large thrust amplitudes, which often determine the extent and boundaries of mining operations, the panel geometry and mining regions. Coal mining in the neighbourhood of large-thrust faults can trigger their activity and generate high-energy seismic events. Most seismic shocks registered to date and accompanying major bursts have typically occurred in the neighbourhood of large-thrust faults (Zorychta & Burtan, 2008). While the longwall operations have to pass through the small-thrust faults, the longwall advance is restricted, a certain amount of coal has to be left in the roof (or gob area) or floor strata, which can lead to coal self-heating and finally result in a fire (Krause & Dziurzyński, 2015). Mining coalbeds with disturbed original structure, with fissures and voids in the faulting zones poses additional fire hazard whilst methane present in those regions in its free state further enhances the risk of methane emission and rock and gas outbursts. These findings are corroborated by reported data on rock and gas outbursts in collieries of the JSW Mining Corporation, confirming that the majority of outbursts were registered in the faulting zones.

**Mining methods and systems**

The employed mining method and system may either enhance or reduce the risk of natural hazard occurrence. As regards the longwall mining system widely employed in Polish collieries, the system components such as mining method, the method to handle old excavations, the configuration and geometry of longwall galleries or the ventilation system are major determinants of the scale and intensity of natural hazards: methane emissions, coal dust explosions, fires and rockbursts. In many cases this aspect has to be considered in while developing the effective prevention measures and strategies (Burtan, 2016).

Compared to blasting methods, mechanical mining widely employed in currently operated longwalls yields the output in the form of smaller rock chunks yet produces more dust in the mine excavations, thus enhancing the risk of coal dust explosion. The roof control strategy adopted in the majority of longwall operations involving the caving-in leads to elevated levels of methane emissions to the excavations (compared to the roof stabilisation methods), thus increasing the risk of methane emissions behind the longwall face. Furthermore, caving-in of roof strata enhances the air migration towards the goaf area, prompting the self-heating of coal in the area and increasing the risk of a fire (Krause & Dziurzyński, 2015). A large number of endogenous fires registered to date took place in caved-in wall sections, in some cases these were accompanied by methane explosions (Wyższy Urząd Górnictwy, 2009-2018).

A practice is currently adopted whereby the longwall galleries behind the working face should not be maintained, hence the need to drive new headings to prepare for subsequent longwall operations, leaving the stretch of undisturbed coal body (coal fences). At larger depths, however, their structure tends to disintegrate. These processes seem to produce positive results in the context of potential dynamic phenomena, yet air inflow is possible into undisturbed coal sections under the critical stress through the systems of cracks and fissures, thus enhancing the risk of an endogenous fire. On the other hand, control
measures taken to leave higher-capacity wider pillars will reduce the risk of a fire, yet they prompt the accumulation of elastic energy, thus enhancing the risk of a rockburst (Burtan, 2016).

The airing conditions in the longwall regions also affect the hazard conditions associated with air quality and ventilation parameters. The most widely employed reversed ventilation system alongside the coal body enhances the risk of gas emissions in methane-rich beds at the outlet from the longwall region, whilst the fire risk is reduced. The ventilation system involving the air removal alongside the mined workings reduces the methane concentrations at the outlet from the face zone, yet the airflow in the vicinity of mined-out workings and air migration through the goaf area enhance the risk of an endogenous fire (Zorychta & Burtan, 2008). Furthermore, in all variants of the ventilation system where air is supplied in the direction opposite to the haulage paths, there is an increased amount of coal dust and an additional risk of coal dust explosion (Burtan, 2016).

Previous mining activities

Because of the multi-bed structure of coal deposits in Polish collieries, previous mining activities and residues (mining remnants) are of paramount importance. These residues include goafs, pillars or edges and they are responsible for changing the primary state of stress in the rock strata as a result of mining operations that were undertaken and later abandoned. Interactions between goaf areas, edges with different strain parameters, and residual pillars give rise to non-uniform state of stress in the strata. As a result the stress tensor components in the vicinity of goaf areas tend to decrease and then increase, forming the stress concentration zones in the proximity of edges and unmined coal body parts. Thus produced disturbances might reduce or enhance the impacts of geomechanical behaviours, which in turns determine the scale of natural hazards: rockbursts, gas and rock outbursts (Chlebowski et al., 2006; Burtan, 2016). Mining operations in the vicinity of stress concentration zones also impact on the ventilation conditions and involved risk factors. Since the rock medium structure can get damaged due to excessive stress, thus produced fissures enhance the risk of a fire in methane rich (gassy) panels.

A major problem encountered in a majority of Polish collieries, particularly those in which the coal deposits have been mostly mined-out, is that they are forced to mine irregular coal residues in protective and supporting pillars in shafts or workings being closed, including the coalbed sections which were left unmined following the decisions to stop the mining operations (Zorychta et al., 2008). The coalbed being mined out or mining operations in residue zones enhance the risk of rockbursts and rock and gas outbursts, mostly due to stress concentrations further exacerbated by interactions between mining edges and residues of the neighbouring beds. In the context of potential formation of fissure zones within the undisturbed coal body, and hence air inflow and migration in fissured medium, identical factors are responsible for increased levels of fire risk (Zorychta & Burtan, 2008).

Sub-level mining

As the expenditures on development works have dwindled in the recent years, numerous collieries resort to sub-level mining, whereby air needs to be supplied to mine workings via the descending air streams. Because of the need to provide or extend the ventilation networks, sub-level mining operations at deep levels negatively impact on the safety features, particularly with reference to methane emissions and fire hazards (Krause & Dziurzyński, 2015). On the other hazard, an occurrence of a sub-level fire, rockburst, methane or coal dust explosion will disturb the operation of the ventilation network, produce still more adverse effects, hinder the effective rescue actions or interfere with the evacuation of miners from the area.

Concentration of coal production

For economic reasons, mining companies in Poland tend to ensure the maximal coal production from the smallest possible number of operated longwalls, aiming to operate longer the working faces and to maintain the relatively high rate of the face advance. Thus understood concentration of coal production, however, tends to enhance the risk of methane emission, leads to elevated coal dust contents and triggers the seismic activity of the rock strata, which consequently results in natural hazard conditions, mostly with
reference to rockburst hazard, and methane emissions and coal dust explosions (Patyńska, 2017).

Assuming the given mining height, operation of a longer working face affects the actual amounts of methane released from the coal body being mined, and the inflow of methane from the neighbouring strata and coalbeds is intensified as it enters the active workings and the goaf areas. For example, the amount of methane released to a 300 m long working face is by 100% higher than the amount released to the 200 m long face (Konopko, 2013; Krause & Dzierżyński, 2015), hence operation of long working faces results in an enhanced risk of methane emission. A faster rate of face advance will intensify the methane release and enhance the risk of methane explosion. Even though the amount of methane released from the coal body being mined varied proportionally to the face advance, the more intensive inflow of methane being a result of methane drainage from the surrounding strata involves a certain inertia in the process. That is why rapid restrictions of the rate of face advance may not result in significant reduction of methane emissions (Konopko, 2013).

As regards the rate of face advance in the context of geo-mechanical processes, it is worthwhile to mention that mining engineers and practitioners tend to vary in their opinions on how the face advance actually impacts on the level of rockburst risk. The mine operators’ expertise has shown that high-energy seismic events are mostly registered when the rates of face advance are higher (Goszcz, 1999), that is why the rate of face advance is reduced as part of the rockburst control strategy in the conditions of elevated levels of natural risks.

Concentration of coal production understood as reducing the number of currently operated longwalls results in reducing the length of potential working faces in which the fire hazard can arise, whilst the fast rate of face advance positively affects the fire hazard levels, as it shortens the time of air inflow into the mine galleries near the working face.


Coal dust explosions

The risk of a coal dust explosion is present in all collieries. Though the popular opinion is that identification of factors that may trigger a coal explosion and implementation of effective dust control strategies should help prevent their occurrence, yet these explosions still do occur. Even though they may not be frequent, they are often responsible for loss of life and major damage. Over the last 10 years 2 coal dust explosions have been registered, both triggered by methane explosions. The previous coal dust explosion triggered by the blasting operations took place in 2002 in the colliery Jas-Mos, causing the loss of 23 lives (Patyńska, 2017). Major determinants of the coal dust explosion hazard levels include the concentration of coal production from working faces (longwall operations with the daily output of the order of several thousand Mg) and the combination of hazardous conditions (methane, rockbursts, rock and gas outbursts).

Methane emissions and explosions

16 out of 21 operational collieries in 2017 are threatened by methane emissions during the mining operations whilst the proportion of coal production from gassy seams tends to increase steadily, approaching 78% in 2016. In the consequence of mining operations in gassy seams, the amount of methane released in 2016 reached the level 933,8 million m³, which amounts to 13,3 m³/Mg whereas the figures for ‘gassy’ coalbeds only are 17,05 m³/Mg. In spite of a general tendency to limit the coal production levels and reduce the number of operational mines, these figures are the highest registered in the Polish coal mining sector (Patyńska, 2017).

In the period 2008-2017 31 accidents have been reported involving methane fires or/and explosion, causing the loss of 37 lives among the miners (Zorychta & Burtan, 2008). During those accidents other hazards were manifested, in some cases methane fires and explosions were triggered by endogenous fires or rockbursts, whilst methane explosions brought coal dust explosions in their wake. The most tragic accident took place in 2009 in the colliery Wujek in the Upper Silesia where an explosion of methane and coal dust occurred during longwall mining operations, causing 20 fatalities among miners whilst other 25 miners were seriously injured (Wyższy Urząd Górniczy, 2009-2018).
Rockburst hazard and de-stressing events

Dynamic phenomena, understood as the potential occurrence of a seismic event leading to the loss of stability of the mine excavations, are experienced now in a growing number of collieries. In 17 out of 21 operational mines (2017) coal is mined from burst-prone seams, and coal production from burst-prone seams accounted for 52.4% of total output in 2016. The rockburst hazard levels are high, too, the registered total seismic energy ratings being the highest over the years, approaching $9.7 \times 10^9$ J (2015). This value is associated with the strongest seismic event ($4.0 \times 10^9$ J) registered so far in the Upper Silesia Coal Basin, which triggered a major rockburst in the coal seam 409 in the colliery Wujek (Wyższy Urząd Górniczy, 2009-2018).

Over the last few years the rockburst hazard has been manifested by over 1,5 thousand high-energy events (energy ratings above $10^5$ J, being the determinant of the scale of the risk) and by several rockbursts and de-stressing events (3 rockbursts and 1 de-stressing event registered in 2017). 32 events were registered from 2008 to 2017, producing major damage in mine workings and causing the loss of 7 lives among miners underground (Patyńska, 2017). It is worthwhile to mention that rockbursts and other dynamic phenomena are often followed by increased methane inflows to the working zone (Kabiesz, 2016). As cracks and fissures appear in barren rocks and in undisturbed coal body, air migration becomes more intensive, the face advance is interrupted and thus the risk of an endogenous fire will enhance, too (Burtan, 2016).

Endogenous fires

Fire hazard results from the coal’s natural tendency to self-heating and is experienced in nearly all coal mines where numerous geological and mining factors play a part, further enhancing the risk of a fire.

In the period 2008-2017 62 endogenous fires were registered (8 of them occurred in 2017), including two major accidents in the colliery Mysłowice-Wesoła in 2008 and 2014, which triggered the self-heating or explosion of methane and causing 7 fatalities among miners (Wyższy Urząd Górniczy, 2009-2018). Endogenous fires, particularly in gobs left following the longwall operations, may result in methane fires and explosions, or coal dust explosions. No fatalities were reported in the investigated period as the result of fires and no further events occurred in their wake, which seems commendable in the context of miners’ safety, confirming the adequacy of fire-detection methods applied in mines.

Rock and gas outbursts

At the moment rock and gas outburst hazard is present during mining operations continued in 5 collieries (2017), including 3 mines within the structures of the JSW Mining Corporation, which experienced major rock and gas outbursts (in colliery Pniówek in 2002, in colliery Zofiówka in 2005 and in colliery Budryk in 2012). The rock and gas outburst in the colliery Zofiówka in 2005 caused the loss of 5 lives among miners (Wyższy Urząd Górniczy, 2009-2018). In the context of a burst-control strategy widely adopted in collieries within the JSW Mining Corporation this event can be regarded as incidental (compared to other major accidents), yet the energy involved in the process is getting higher and higher, which affects the miners’ safety levels. One has to bear in mind that all registered outbursts occurred in development headings, at the stage of heading driving in the proximity of faults (Burtan, 2016).

Rock and gas outbursts can results in enhanced levels of methane emissions and coal dust explosion risk and can trigger further hazardous conditions being the consequence of these events.

Mine water inrush and inflow

Unlike previously listed hazards, water hazard tends to decrease with depth as water-bearing capacity of rocks becomes lower and the volume of water reservoirs formed in mine workings is restricted due to tightening of the gob zones. Many years of mining in the carbon strata results in the level of water inflow risk becoming lower following the years-long drainage of the rock strata. Actually, to a certain extent the water hazard is present in nearly all coal mines. In the investigated period of time (2008-2017) 1
water inrush incident was reported (in the colliery Knurów-Szczygłowice in 2013), causing 1 fatality (Wyższy Urząd Górniczy, 2009-2018).

WORK SAFETY CONDITIONS IN MINES

Statistics of accidents

Natural hazards are present during the underground coal mining operations, giving rise to accidents and catastrophic events, posing a threat to miners working underground.

In the period from 2008 to 2017 128 accidents were registered that were caused by natural hazards, in most cases endogenous fires. The largest number of accidents, coinciding with the largest number of reported fires, took place in 2009 (20) and in 2017 (15). Actually in the last three years of the analysed time interval the number of accidents attributable to natural hazard conditions remained on a relatively high level (11-15 accidents a year) despite coal production cutbacks. When analysing individual hazards and accidents triggered by them, the effect of a sequence of events should be interpreted as the effect of a final incident. Hence the concurrent occurrence of a fire and methane fire or/and explosion should be interpreted as a methane explosion, whilst a fire and a concurrent methane fire or explosion and a coal dust explosion should be counted as a coal dust explosion.

![Frequency of various natural hazards](image)

Figure 2 – Proportion of natural hazards resulting in major accidents (2008-2017)

The analysis based on those criteria (Fig. 2) reveals that the largest number of incidents were caused by endogenous fires (47.6%) whilst rockbursts and de-stressing events accounted for 25.0% of incidents whilst methane fires and explosions accounted for 24.2% of their total number. Proportionally, the smallest number of incidents were caused by coal dust explosions (1.6%), gas outbursts (0.8%) and water inflows (0.8%).

Statistics of fatalities

Catastrophic incidents triggered by natural hazards lead to numerous accidents at work, including also loss of life. In the period 2008-2017 natural hazard conditions and resulting accidents were responsible for 45 fatalities. The most serious accident took place in 2009, causing the loss of 20 lives in the colliery Wujek, following an explosion of methane and coal dust. A large number of fatalities was registered also in 2008 (8 fatalities including 6 miners who lost their lives as a result of methane explosion in the colliery Borynia). In contrast, in 2017 no loss of life was reported despite a relatively high number of accidents (15). In the analysed period of time the largest proportion of fatalities related to natural hazard conditions were caused by coal dust explosions (48.9% of their entire population), in most cases these were triggered...
by methane fires of explosions (Fig. 3). A significant though a slightly lower number of accidents were caused by fires and explosions of methane (33,3\%) including those triggered by fires; a considerable number of accidents were triggered by rockbursts and rock de-stressing events (15,6\%).

![Figure 3 – Proportion of natural hazards resulting in fatal accidents (2008-2017)](image)

The smallest fraction of incidents were caused by water inrush (2,2\%). Rock and gas outbursts and endogenous fires which did not bring methane or coal dust explosions in their wake did not result in any fatal accidents.

**CONCLUSIONS**

Mining and geological conditions characteristic of coal deposits in collieries within the Upper Silesia Coal Basin and the employed coal extraction method determine the scale and intensity of emerging natural hazards which may lead to serious accidents, in particular methane emissions, coal dust explosions, endogenous fires, rockbursts, rock and gas outbursts and water inflows. These hazard conditions can trigger dangerous situations resulting in major accidents, even involving the loss of life. The analysis of hazard conditions and accidents reported between 2008 and 2017 leads us to the following conclusions:

- Among the natural hazard conditions, those that caused the majority of dangerous accidents included endogenous fires, rockbursts, de-stressing events, fires and explosions of methane, whilst coal dust explosions, rock and gas outbursts and water inflows caused a smaller number of incidents.
- The largest number of fatal accidents being the consequence of natural hazard conditions seem to have been triggered by explosions of methane and of coal dust, another cause of numerous accidents were rockbursts. The smallest number of fatal accidents were caused by water inrush, endogenous fires and rock and gas outbursts.
- Hazard conditions associated with methane and coal dust explosions are regarded as the most dangerous in the context of potential accidents, even though the registered explosions of methane and coal dust were few and far between. Rockbursts seem to produce the most hazardous conditions, often resulting in accidents.

Natural hazard conditions not only lead to fatal accidents, but can also result in interruption, temporary ceasing or abandoning of mining operations, thus affecting the overall coal production figures. The decision to leave some coalbed sections unmined because of natural hazard conditions can reduce the mining company’s production capacity, in certain cases it can even result in shortening the working life of a mine.

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POLISH EXPERIENCE IN SHAFT DEEPENING AND MINING SHAFT HOISTS EXTENDING ON THE EXAMPLE OF THE LEON SHAFT IV IN THE RYDUŁTOWY MINE

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ABSTRACT

The Leon IV shaft in Rydułtowy Mine is one of the last mining shaft deepened from the surface in Polish hard coal mining industry, extended and finally equipped with mine shaft hoist installation. This investment will allow the construction of the exploitation level 1150 m and availability of next hard coal seams to the level 1200 m, what will guarantee possibility of the exploitation of over 65 mln ton of coal, as well as continuous operation of the coal mine up to 2040. This shaft was an object, in which a number of innovative solutions were implemented during its construction, due to actual level of technical and material technologies. Elastic guiding of mining cages was applied for the shaft furniture as a one of the first time in Polish mining industry, what in comparison with rigid shaft furniture is considerably cheaper and has many other advantages.

A number of innovative solutions, which have not been used before in Polish mining shaft building were applied during the Leon IV shaft building period – basal deepening in the years 1990-1998 and additional deepening in the years 2013–2017.

Deepening of active mine shaft comprises number of specific and very difficult operations, because it calls for use of untypical devices securing hoist operation in the shaft, as well as special technology tailored to actual technology of the deepened shaft face.

In case of the Leon IV shaft building, the special attention was paid for works related with the shaft deepening and extension of the shaft hoists, where elastic guiding of mining cages was applied for the first time.

Finally, the shaft deepening and extension of the shaft hoists will allow development of the new transport level at the depth of 1150 m, what in turn will improve condition of the mine ventilation and it will result in personnel transport time shortening.

KEYWORDS

shaft construction, deepening mining shafts, shaft hoist elongation,

INTRODUCTION

Hard coal mine Rydultowy is one of the oldest Polish mine in Rybnik Coal District (Retrived from: http://pl.wikipedia.org/). Its predecessor named Charlotte started production in the year 1806 and it was one of the greatest mines at that time. The mine as the first was equipped with steam engine already in the year 1855 and it was connected with the rest of the country via railway line what facilitated coal sales. At the beginning of twentieth century, the mine in question passed trough numerous crisis phases what resulted in the employment reductions, and in the year 1932 the mine was even closed for the period of four years. However the mine developed during the Second World War – because Germans needed big amount of good quality coal. In the period 1940–1944 the employment was increased threefold up to 3582 workers (Retrived from: http://pl.wikipedia.org/). After the Second World War the Charlotta mine was renamed as Rydultowy mine, which belonged to various structures of Polish mining industry. In the year the KWK Rydułtowy was joined with Anna mine forming mining plant named as KWK Rydułtowy–Anna.

In the period 1990–1998 a new shaft Leon IV of rarely occurring in Polish mining industry, with diameter amounting for 8,5 m. In this period it was decided that the shaft depth of 1076,2 m would allow development of the new level 1050 m, what would fully satisfy the mine operational needs. Reach resource basis of the mine on the depth over 1000 m and necessity of avoiding of over levelled exploitation allowed implementation of the next investment comprising the shaft Leon IV deepening to the depth of 1210,7 allowing development of next exploitation level 1150 m. In the year 2013 design works were started and the process of shaft pipe deepening and extension of two shaft hoists (main and auxiliary) to the depth of 1000 m, have been started (Retrived from: http://wodzislawslaski.naszemiasto.pl/).

Flexible -ropes guiding of the shaft hoist cages in the Leon IV were implemented for the first time in Polish hard coal mining industry. Shaft deepening and necessity of extension of shaft hoists to the depth of 1150 m constituted great challenge both for designers and the unit realizing building Works. It should be noted that like in each active coal mine, deepening of the active shaft is related with necessity of its continuous and undisturbed operation. In case of such technological restriction, works related with shaft deepening call for special securities, among other leaving of the rock shelf called as natural bottom, or building in the shaft so called artificial bottom.

In such case, transport works in deepened shaft section call for building of auxiliary hoist device with underground hoist machine of special turnstile adapter for personnel transport. Big-diameter whole used for winning transport, water drainage and fresh air bringing can greatly facilitate the works related with shaft deepening. However, in such case excavation on the level to which the shaft is deepened, is needed. Shaft Leon IV can be a good example of application of new technical and technological solutions. Three of such solutions will be discussed in the present study:
• single-layer waterproof lining within the section 782.0–932.0 m,
• shaft deepening technology within the section 1076.2–1210.7
• extending of shaft hoists from the level 1000.0 (960 m) to the mining level 150 m and auxiliary level 1200 m used for the mine water drainage.

SINGLE-LAYER SULPHATE-PROOF LINING OF THE SHAFT LEON IV

In original project of the shaft IV, two-layered lining with hydro–insulating shield made of PE foil was foreseen for the shaft section 782.0–932.0 characterizing with occurrence of sulphate and magnesium waters. Such linings were commonly used by KOPEX – Shaft Building Company S.A. Sinking technology within the section in question foresee the following works (Kostrz et al, 2000):

a) between ordinates 784.5–786.0 m – building of the B15 class concrete lining (currently C12/15 concrete class),
b) between ordinates 786.0–932.0 m – building of preliminary shaft lining (in direction from top to the bottom) in form of 0.56 m thick shaft concrete block wall,
c) between ordinates 930.5–932.0 – building of shaft brick made of B30 class concrete B30 (at present C25/30),
d) making of internal layer of the lining made of B25 and B30 class concrete (at present C20/25 and C25/30) wet concreting from the top, after former putting and sealing of 2 mm thick foil shield.

Because at this time one of the Polish cement factories produced special Portland cement called as bridge portland cement CP 45(M) marked with symbol CP 45(M) resistant to strong sulphate and magnesium aggression, after research works executed in the AGH University of Science and Technology, modification of the construction and technology of completion of shaft lining section, has been proposed. New project of single–layered lining lied from the top to the bottom following the shaft face advance comprised making of 0.65 m thick single-layered concrete lining lied in wet system. In dependence of calculated pressure onto the shaft lining, its bearing capacity was controlled due to concrete strength with use of two receipts (Kostrz et al, 2000) for concrete of class B25 marked as R25/1/2 and for concrete B30 marked as R30/1/4. All concretes were prepared on the basis of Bridge Portland cement CP 45(M).

Receipts developed in the AGH University of Science and Technology and verified by laboratory tests guarantied suitable strength of the concrete and suitable bearing capacity of the lining of targeted thickness, as well as suitable watertightness of the level W8. Concrete lining was made in 4 m long sections in direction from the top to the bottom. In such technology, waterproofness depends mainly on technological joints between upper (old) section and bottom (new) section. In the project in question, re–sealing of these joints was made first time in Polish shaft building with use of injection hoses of the type FUKO 2 (Fig. 1.), having former opinion from Higher Mining Office concerning their security due to methane presence, after special tests conducted in Experimental Mine Barbara in Poland. The injection hoses FUKO 2 were mounted to upper section of the shaft with use of metal connectors fitted with screwed joints. After the next lining, the section was concreted, the fissure was filled with a binding mixture on its whole length (see Fig. 1 and Fig. 2) obtaining satisfactory sealing of the neuralgic element of the shaft lining.

Figure 1 – Sealing system of the concrete shaft lining with use of hoses FUKO 2 (Kostrz et al, 2000)
a) Injection hose FUKO 2; Markings: 1 – injection channel $\phi 10$ mm, 2 – hose core; 3 – injection holes; 4 – neoprene ribbons playing role of non-return valves.


Next problem solved during the shaft building comprised drainage of the rock body behind the lining. It is commonly known that water swell being the waterproof lining is dangerous with respect to possibility of the occurrence of high hydrostatic pressures into the shaft lining, in result of joining different water-bearing horizons by the shaft hole. This problem was solved by laying vertical drainage pipelines along the shaft lining 100 mm distributed on four azimuths (see Fig. 2.)

Figure 2 – Injection system via hoses FUKO 2 and rock body drainage system behind the shaft lining (Kostrz et al, 2017).

In case of used waterproof single-layered shaft lining, the key problem comprises waterproof concrete resistant to sulphate aggression. The concrete mixture was prepared in professional concrete plant located in the distance of 40 km from the shaft. Thus the concrete mixture had to be designed in such manner that after bringing it to the shaft it kept suitable consistence measured by the cone fall amounting for about 12 cm. Additionally, admixtures of slag binder, plasticizer SK–1 and polypropylene fibre called as “Fibermesh”, have been introduced to the concrete mixture receipt.

The elaborated concrete mixture receipts based on sulphate-resistant bridge cements CP45(M), as well as constant author supervision of concreting works in the shaft were successful. Replacing of the two-layered lining with waterproof single-layered lining allowed reduction of the cost of this shaft section by about 30%. This solution can be positively assessed after almost 20 years of exploitation, in particular in case of the aggressive waters occurrence.

**Chosen elements of the shaft Leon IV deepening**

This about 80 mln PLN investment of the Leon IV shaft deepening by next 140 m resulted from necessity of the production processes modification in the Rydultowy mine (Olszewski et al, 2017). This modification comprised first of all shortening of the time of personnel transport to the shift face, as well as facilitation of the needed materials delivery and considerable improvement of the ventilation of this part of the mine.

The investment task related with development of mine infrastructure in Leon IV region comprised the following activities:

- making the shaft deepening technical project,
- physical shaft deepening and reinforcing,
- making the two-way inlet to pit bottom of exploitation level at the depth of 150 m,
- making the inlet to single-way pit bottom at the depth of 1200 m destined for needs of the mine main drainage system,
- elongation of mining hoists: main to the level of 1150 and auxiliary to the depth of 1200 m,
- installation of needed elements of mechanical equipment of the inlets to pit bottoms of both built levels.
Targeted depth of 1210.7 m was reached in August 2016. After completion of the reinforcing of the deepened shaft part in the year 2017, pioneer in Polish mining industry works related with elongation of shaft hoists, including untypical and difficult elongation of guiding hoists accompanied by exchange of all leading ropes, have been executed.

**Technology of the Leon IV shaft deepening**

The Leon IV shaft was deepened to level 960 m, keeping full exploitation ability. Works related with shaft deepening were conducted with artificial bottom of special construction of two platforms joined with vertical partition. Mining works were conducted by standard method with use of explosives. Because winning haulage is the greatest problem in deepened shafts, in the case in question, at first a dike was made on level 1200, in order to make great diameter hole of the length of 115 m reaching the shaft bottom before deepening. The hole was located in such manner that its axis was located in a distance of 2.2 m towards East from the shaft axis, what allowed collision-free operation of the basket covering the hole inlet in the shaft face. However, the 1200 mm diameter hole was exposed to danger of development of the winning jam. In order to remove the jam in the hole, a rope of 25 mm diameter with conveyor scrappers was installed in the hole. The rope vertical movements stimulated by low-speed winches KUBA–5 installed in ditches on the level 1076 and 1200 m. (see Fig. 4) provoke winning fall down and the hole clearance.

Transport in the whole deepening process was handled by special devices located in ditch on level 1076 m (Fig. 4):

- hoist machine B–1500 for bucket handling,
- two low–speed windlass KUBA–10 for adjustable formwork handling,
- windlass KUBA–5 for rope handling,
- windlass KCH–9 for basket hanging protecting hole in shaft face,
- supporting construction for assemblage of wheels during shaft deepening.

In the ditch, on the level 1200 m, low–speed windlass KUBA–5 with track wheel for the hole clearing rope (see Fig. 4) was installed. Single–layered lining of C30/37 concrete lied in wet system with use of steel moveable formwork of the height 2.15 m, has been applied. Calculated and consulted with the Investor lining has thickness from 0.5 to 0.6 m. With respect to expected small and ephemeral water inflows into the shaft, none special waterproof precautions were designed.
Start–up of levels 1150 m and 1200 m

Thanks to the Ruch Rydultowy investment, it will allow of exploitation from coal seams No 713/1–2 and 712/1–2, which belong to the most promising mining assets within mining areas belonging to this part of Rybnik Mining District.

Development of this part of the deposit will allow building the new level at the depth of 1150 m. Two–way inlet is equipped with full set of the wheel transport handling, with special platform for material re–loading from wheel into lifted gondola transport.

The inlet performances are as follow:
- excavation founding depth – 1143,7 m,
- height – 7,3 m,
- width: W side – 8,11 m , E side 7,0 m,
- basement depth – 2,30 m.

The pit–bottom geometry with use of 3D visualisation is shown in Fig. 5.

Universality is characteristic feature of the shaft pit–bottom 1150 m – main transport level. Within this level there is a possibility of using three shaft hoists, what will considerably accelerate process of material lifting, as well as it will allow fast and fluent personnel transport. Using transport platforms forced equipping shaft pit–bottom basement with devices and machines needed for pushing mine trucks into large–size mining cages, as well as into standard cages. By the occasion of the Leon IV shaft deepening, water management in this region was ordered. For this purpose, excavations needed for the main water drainage handling were localized in one–way pit–bottom of the level 1200 m, and the elongation of this level to auxiliary hoist was necessary, and it was re–qualified from auxiliary hoist into “small” hoist.
Geometry of the shaft inlet on level 1200 m:
- depth of the excavation founding – 1195.7 m,
- height – 4.2 m,
- width – 6.1 m.

The shaft inlet has anchor–concrete–steel lining and is equipped with level guidance construction with oscillatory platform in the inlet basement.

Furnishing of the shaft Leon IV

As the main transport shaft, the shaft Leon IV is equipped with three compartments: one for main hoist with large-size cage, second for ordinary three-deck cage and auxiliary hoist. The shaft cages are suspended on two 48 mm diameter rope carriers driven by drive wheel Koeppe. In order to balance masses of rope carriers, two equalizing ropes of diameter \( \phi 53 \) mm are installed.

Shaft Leon IV is the first shaft in Polish hard coal mining industry, in which flexible guiding of shaft cages or skips has been applied. Guiding and defender ropes are suspended on wedge–shaped spreader beams located over beams of the shaft tower. The guiding and defender ropes hang down freely and are tensed by the attached weights of such mass that each 100 m of the shaft depth corresponds to tensing power of the value at least 8 kN. The guiding and defender ropes are mounted in special baskets located below lower guiding rope frame. In case of the Leon IV shaft, the shaft furniture consists of the following elements: (Fig. 6 and 7).
Elastic guiding of the shaft cages comprises 12 guiding ropes, 4 defender ropes and 3 rope carriers and equalizing ropes between large size–cage and three–deck cage (Fig. 7).

![Diagram of shaft Leon IV](image)

Figure 7 – Elements of the elastic guiding of shaft Leon IV (Olszewski et al., 2017)

Profits resulting from application of rope guiding of hoist cages ore skips are great. The profits are as follow:

a) low cost of used materials,
b) easy handling,
c) long durability,
d) short assembling time in new shaft,
e) guiding of the cages ore skips in the shaft is soft, without of tremors and side hits,
f) possibility of fast shaft hoist operation,
g) quiet run cages ore skips results in elongation of the rope carrier,
h) ventilation resistances are almost 10 times lower than in case of shafts with rigid guides.

Linear guiding of dishes has also some disadvantage, like:

a) possibility of transverse movements, which are vertical to the running direction, what calls for longer operation intervals than in case of rigid guides,
b) serious difficulties in exploitation result – rock body movements,
c) need of great size shaft diameters and difficulties related with relocation of shaft hoist devices in numerous of transport sections,
d) tensions of guiding ropes reaching values of 2000 kN causing additional loading of the shaft tower and application of suitable construction needed,
e) rope tensioning advices need suitably adapter – sump,
f) in case of application of two–cages hoists there application of additional ropes called as “defender ropes” protecting against collisions of dishes moving in two different directions, are needed,
g) rope carrier must be made of non–rotating rope,

The main advantages of the application of such type of guiding result from the fact that the use of tens shaft fastening frames is not necessary.

The other element of the shaft furniture is related with main hoist cages braking system localized under level 1150 m and in the shaft tower. This system is composed of thickened wood guides. The other elements of the shaft furniture comprise:

a) platform of the return station of equalizing rope,
b) platform of equalizing ropes control system,
c) positioning frame of weighs of guiding ropes and bumper ropes,
d) control platforms of guiding and bumper ropes,
e) protective platform.
Necessity of equipping the level guidance with special corner guiding of dishes is essential element of shaft dishes elastic guiding on individual levels. In case of the shaft Leon IV it refers to levels 800 m, 1067 m, 1150 and 1200 m.

**ELONGATION OF MINE SHAFT HOISTS**

Shaft deepening is strictly related with necessity of mine shaft hoists. In case of the Leon IV shaft, in construction of elastic guiding of hoist dishes to level 1067 m, the ropes longer than the exploited shaft, were applied. Excess of ropes was stored on special drums located in pit–bottom of the level 1076 m. Thus elongation of these ropes required only suitable control of the rope destruction degree and then lowering them to the level 1150 m. Works related with elongation of the mine shaft hoist were started from auxiliary hoist and then lowering two guiding ropes of diameter Ø 32 to level 1200 m. Weight of the rope of diameter 32 mm amounts for around 5.3 t, whereas for ropes of diameter Ø 54 mm this weight reaches value over 25 ton. Thus every rope manoeuvres, i.e. their raising or requires execution of assembling works with use of special hoist machine having high lifting capacity, as well as special guiding wheels mounted on special platform. Low–speed lifting machine EWP–35 was used for rope lifting and lowering. After making welded clamps and taking rope weight by lifting machine, disassembling was made in order to check condition of ropes by suitable expert. After acceptance of the ropes for exploitation they were lowered to level 1200 m, where weighting baskets were mounted. Next stage of elongation of the shaft hoist comprised elongation of guiding and bumper ropes of main hoist i.e. unravelling rope reserve accumulated on the level 1067 m. After installation of additional constructions of rope wheels on the platform on lower rope wheel of the Leon IV shaft, i.e. directing technological rope Ø 40 mm in place of wedge–shaped spreader beam, stage of basal works related with taking and lowering ropes in targeted place, have been started. When the welded clamps were installed and the weight was taken by low–speed lifting machine (40 t), the rope was lifted to level of the foundation in order to be checked by the expert, and then the rope was lowered again to the level of weights control platform. After completion of the operations of the first rope the guiding wheel was relocated on the platform in such manner that operations of the next ropes would be possible. This operation was repeated eight times for guiding lines of the main hoist dishes, and four times for bumper ropes located between both dishes of the same hoist. Scheme of devices needed for hoist elongation and visualization of the guiding wheels on the tower, are show in Fig. 8.

![Sliding sheave wheel](image)

Figure 8 – Visualization of sheave wheel location on hoist tower (Olszewski et al., 2017)

Technology of the rope carriers and equalizing ropes in the shaft Leon IV are similar to standard technologies of ropes operated in mine shafts. In this case, at first some preparatory works related with construction of foundation of the lift EPR–1000 and installation of sheave wheels, have been executed in the shaft foundation. Carrying ropes were lowered to the shaft after placing shaft dishes on special platform and taking the weight of ropes by portable lift EPR – 1000.

After relocation of the great–size cage to the level 1150 m and installation of spreader beams, the equalizing ropes were elongated. In this case, the rising up large–size cage pulled new equalizing ropes. After relocation of three floor cage to the level 1150 m, construction of equalizing ropes under foot of three floor cage, was completed. All operations were successful and actually the final works are continued, before validation the shaft for exploitation.

**SUMMARY**

Technical problems related with the shaft Leon IV sinking and deepening presented in this can be considered as an example of continuous innovation and development of the shaft building technology. Although nowadays shafts are rarely deepened, high level of modern technique and mechanization of both sinking and equipping the shafts indicate for potential possibilities of further development of this building branch.
Application of new bridge cements M45 and modification of the philosophy of assuring the shaft lining tightness even during building the shaft Leon IV allowed implementation of very profitable replacement of multi-layered lining with single-layer lining, which is less time consuming and much more cheaper.

Shaft deepening during its exploitation was possible only in result of application of modern construction of „artificial bottom”, which tightly separated the shaft from the area of works conducted by the company named as KOPEX – Shaft Building Company S.A.

Application of elastic system in the shaft Leon IV in hard coal mining industry and use of much more longer ropes and storage of the rope surplus on the level 1078 m can be classified as the uniquely far-sighted project. This in turn allowed implementation of much more easy technologies of shaft hoists elongation.

Designed by Shaft Sinking Company elongation of the shaft hoists, which was realized in possibly shortest stoppage of the shaft operation, was pioneer and innovative venture. Total works comprised elongation of 20 ropes.

Modernization of the shaft Leon IV was a key element of the restructuring plan and development of the joint-venture mine ROW gathering mines: Jankowice, Chwałowice, Marcel and Rydultowy. Elongation of shaft hoists, development of main transport horizon on the level 1150 m and development of the main drainage system at level 1200 m will allow considerable shortening of the time of personnel transport to exploitation excavations, what is related with considerable improvement and elongation of the personnel working time, i.e. improvement of financial results of mine operation and whole mine ROW.

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MODEL TO DETERMINE THE OPTIMAL VARIANT OF THE FACE TECHNOLOGY WHEN THE CONDITIONS NATURAL ONES DESCRIBING A DEPOSIT ARE KNOWN

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MODEL TO DETERMINE THE OPTIMAL VARIANT OF THE FACE TECHNOLOGY WHEN THE CONDITIONS NATURAL ONES DESCRIBING A DEPOSIT ARE KNOWN

ABSTRACT

Regardless of the historical period, mankind, to survive, needs useful mineral substances extracted from nature. These are in the form of deposits in Earth bark at different depths. In order to be exploited, it is necessary to know the following information:
- the geological-mining conditions in which the deposit is located;
- the form of deposit;
- the geometric parameters describing the reserve units of the deposit;
- the technical-economic parameters describing each reserve unit.

Depending on this information, the research activity in the field has developed models that offer solutions for the exploitation of mineral substances deposits useful in conditions of economic efficiency. The model that will be presented allows to determine the optimal variant of the face technology that can be used by an underground method of exploitation.

KEYWORDS
Deposit, useful mineral substance, exploitation method, face technology, production cost.

GENERALITIES

The exploitation of a useful mineral substances deposit is based on mining decision processes with a high degree of complexity, given by the following information:
- the natural conditions in which a useful mineral substances deposit is found;
- quantitative parameters (geometrical, mechanical, tectonical, gazo-dynamics, hydrodynamics, physical and chemical) and qualitative parameters (the mineral substances content of the deposit etc.) of the deposit and of the rock masses surrounding the deposit;
- the resources available for capitalizing the deposit (financial, human, material);
- the technics and technology available;
- the infrastructure in the region where the deposit is found.

The process of knowing a useful mineral substances deposit, being a complex process is carried out in stages, namely:
1. the detection, through various methods of research, of a deposit of useful mineral substances;
2. Collecting information regarding the environment around the deposit;
   - the qualitative and quantitative parameters of the rock masses in the roof and the floor of the field;
   - tectonics of rock massifs in the roof and the floor of the deposit;
   - hydrogeology of rock masses in the roof and the floor of the deposit;
   - gazo-dynamics of the rock masses in the roof and the floor of the deposit.
   This information allows for the development of a grafo-analytical model, a model that allows the generation of natural conditions in which a deposit or a reserve unit belonging to the deposit can be found.
3. collecting information about the deposit:
   - collecting quantitative information about the deposit:
     - geometric parameters of the deposit (number of deposit units, direction and incline lengths, inclination, normal, horizontal and vertical thickness);
     - the physico-mechanical parameters of the useful mineral substance;
- the general tectonics of the deposit and the deposit units;
- collecting information, of a qualitative nature, about the useful mineral substance belonging to the reserves of the deposit.

With this information, a grafo-analytical model is being developed, which allows the generation of solutions for the capitalization of the useful mineral substances within each unit of reserves.

By coupling the two graph-analytical models (the graph of the conditions and the graph of the solutions) results a great-complex graph-analytical model, a model that allows the generation of possible solutions for the exploitation of the useful mineral substances belonging to the reserves units of the deposit.

The graph-analytical model resulting from the combination of the two models (conditions-solutions-solutions) comprises two parts:

1. The graphic part (a graphical construction fig.1) that allows the generation of solutions variants for the exploitation of a deposit depending on the natural conditions in which it is found. This graphical construct only realizes the links between the values of two adjacent variables, links presented in the form of left-right oriented arcs.

Analyzing fig. 2 it is observed that in this figure two columns of nodes with one node were inserted; these nodes represent the start time of a graph path (START) and the end of the road path (STOP). The two nodes (Start and Stop) are compatible with each value belonging to categories and solutions variable;

2. The analytical part (the analytical transposition in the form of table, of the graphical construction, fig. 2), allows to generate the solutions for the capitalization of a deposit according to a set of natural conditions in which the deposit or a reserve unit of the deposit is found. This part of model allows for the removal of the variants impossible to achieve and will retain only those solutions variants that can be achieved.

<table>
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<tr>
<th>Nr. crt</th>
<th>Specifications</th>
<th>Parameter values</th>
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<th>Columns of solution matrix</th>
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<tbody>
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<td>Lines of condition matrix</td>
<td>CONDITION MATRIX</td>
<td>MODULE OF CONNECTION BETWEEN CONDITION MATRIX AND SOLUTION MATRIX</td>
<td>SOLUTION MATRIX</td>
<td></td>
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Fig. 2. Connections matrix
The analytical part called the "Connections Matrix" realizes the following:
- through the matrix "conditions" realizes the connections between the values of a variable "natural categories", with the values of all the variables "natural categories" (the matrix of connections natural-categories-natural-categories);
- through the solutions connections matrix make the connections between the values of a "solution categories" variable, with the values of all "solution categories" variables, in part (solutions-solutions matrix);
- through the linking module between the matrix "conditions" and the matrix "solutions" realizes the links between the values of matrix "natural categories - natural categories" and the matrix "solutions-solutions".

Having known: the system of opening and preparation of the deposit, respectively of a unit of reserves; the method or methods of exploitation of a reserves unit, it is the question of the establishment of the technology, respectively of the technological variant of face, used for the extraction of the useful mineral substance from a reserve field of face belonging to a reserve unit.

The solutions of opening, preparation and exploitation of a deposit or a unit of deposit were established using a graph-analytical model.

Establishing the most appropriate technology, or technological variant of face for cutting of a face field, is also carried out using graph-analytical models.

**GRAPH-ANALYTICAL MODEL OF GENERATING, EVALUATING AND ESTABLISHING THE TECHNOLOGY, RESPECTIVELY THE TECHNOLOGICAL VARIANT OF EXPLOITATION OF THE RESERVES OF USEFUL MINERAL SUBSTANCES BELONGING TO A FACE FIELD**

**Generalities on the classification of information used to determine the variants of capitalization solutions of a deposit**

1. **The classification of the information according to the nature of the state variables**
   Any complex system can be characterized by a determined number of parameters that at one point take different values and define a certain state of the system, which is why status variables are also called. Status variables describe the system from two points of view: qualitative and quantitative. Depending on the domains described, state variables can be grouped into: category variables, continuous variables and discrete variables.

   The **category variables** represent the group of status variables describing the complex system from a qualitative point of view, e.g.: the nature of the rocks in the stope, the type of coal, the methane emanation regime, the type of support in a face, the type of underground transport, etc.

   The **continuous variables** represent that group of state variables describing the complex system from a quantitative point of view (mass, volume, length, and time). Continuous state variables take any value from a given range, such as: seam thickness, length of the face, length of a hole of mine, etc.

   The **discrete variables** describe the quantitative part of the system, but discreetly (number of pieces, elements or number of component parts). Discrete variables can only take certain values within a given range, without passing through intermediate values, e.g.: the number of seams, the number of face fields, the number of support sections, the number of workers placed on a work stope, etc.

2. **The classification of the variables according to the time variation**
   Changes in time of the value of one or more continuous or discrete variables lead to a change in the state of the complex system only in terms of quantity. This change does not lead to a qualitative change of the system.

   It is known that any complex system evolves over time, changes both in qualitative and quantitative point of view.

   The evolutions over time of the variables describing the state of a complex system can be: dynamic, quasi-static and static.

   **Dynamic variables of state** are those variables that change continuously over time and are therefore time functions. The trajectories of these variables are not constant over the timeframe analyzed.
**Quasi-static variables of state** are those variables that, over a period of time, only know discreet changes. Trajectories of these variables look like a gradient graph over the analyzed time period.

**Static variables of state** are those status variables that remain constant over a period of time. The trajectories of these variables are constant over the analyzed time period.

3. **The classification of variables of state according to their relation to the decision-maker**

During the existence of a complex system, frequently or periodically, it intervenes to modify its status, both qualitatively and quantitatively, according to the aims pursued. There are status variables that can or may not be altered by human intervention. Depending on this, state variables can be grouped into: variable stimuli and reactions variables.

- **The stimuli variables** are the set of status variables that can not be changed by human intervention, for example: the number of layers, the nature of the rocks in the stop, the depth of the deposit, from the surface etc.

- **The reactions variables** are the set of state variables that can be modified by human intervention, for example: the number of punches that are used concurrently to puncture holes in the stope, placement of work force in work site, etc.

4. **Indicators variables of result**

The leadership of any system aims to transform the set of stimuli $S$ into admissible reactions $R$. Any operator that transforms objective stimuli into reactions is called determinant. The role of the determinant is played by the man or group of people who transforms, under certain established rules, the "$S$" stimuli into "$R$" reactions. In transforming stimuli-reactions, written in the form: $S \rightarrow M \rightarrow R$ it tends towards achieving the most useful transformation.

The variables that characterize the results are dependent on the sets $S$ and $R$ and are called outcome indicators.

**The database required to design the grap "conditions"**

The information needed to elaborate the model with graph structure to describe the conditions in which a deposit or a reserve unit is located are grouped as follows:

1. **The group of information "stimuli, statics and categories"**, comprising:
   - the shape of the relief from the surface of the massif in which the useful mineral substances deposit are found;
   - type of depth from the surface to the useful mineral substances deposit;
   - type of stability of the massif;
   - the type of rocks' stability in the rock strata located in the roof or in the floor of the deposit unit;
   - the presence of groundwater in the rocks massif situated in the roof or in the floor of the deposit unit;
   - the groundwater accumulation form;
   - the presence of mine gases in the rocks massif located in the roof or the floor of the deposit of useful mineral substances;
   - the form of mine gas accumulation;
   - type of gas emissions from the massif;
   - the presence of tectonic accidents in the rocks massif located in the roof or the floor of the reserve unit;
   - type of tectonic accidents;
   - the type of slope of the rocks layers that make up the rocks massif located in the roof or the floor of the deposit of useful mineral substances;
   - type of variation of the thickness of the reserve unit.

2. **The group of information "stimuli, statics and continuous"** including:
   - average value of depth from the surface to the backup unit;
   - relative flow of the groundwater from the massif surrounding the deposit;
   - relative flow of mine gases from the reserve unit or surrounding rocks massif;
   - the values of the strength coefficient of useful mineral substance;
   - the values of the abrasiveness coefficient of the useful mineral substance.
3. The group of information "stimuli, statics and discreet" includes:
- the number of layers of rock that make up the massif of rocks;
- the number of tectonic accidents of the same type affecting the reserve unit;
- the number of aquifer formations in the roof and the floor of the reserve unit;
- the number of geological blocks in which the reserve unit is fragmented.
4. The group of group of information "stimuli, dynamic and categories" including:
- category of variation of stability of massif on direction and slope;
- category of variation of tectonic accidents on direction and slope;
- the variation of rock layers on vertical;
- the variation of the type of gas emissions from the massif;
- the variation of the groundwater accumulation type on direction and slope;
- the variation of the type of mine gas accumulation in the direction and slope;
- the variation of the relief type from surface on direction and inclination.
5. The group of group of information "stimuli, dynamic and continuous" includes:
- the variation of the absolute water flow of aquifers;
- the variation of water pressure from an aquifer from the rocks massif;
- the variation of the absolute flow of the mine gas emissions from the reserve unit or the surrounding rocks massif;
- the variation of the gas pressure cantonated in the rocks massif;
- the variation of the water circulation speed of the aquifer formations;
- the variation of the geothermal stage;
- the variation of mining pressure.
6. The group of group of information "stimuli, dynamic and discreet" includes:
- the variation of the number of the same type of tectonic accidents on the direction and inclination of the reserve unit;
- the variation of the number of water deposits on the inclination and direction of the rocks massif;
- the variation of the number of rocks layers of the same type on the direction and inclination of the rocks massif.

The database required to design the graph "solutions"

The information needed to elaborate the model with graph structure for generating technological variants that can be used to exploit a face field, when known the natural conditions in which the reserve unit are located, are of the "Reaction categories" type, e.g.:
- the exploitation method and the variants of the exploitation method;
- the type of rocks cutting in the stope;
- type of cutting the front rocks in the niche area;
- the type of rocks evacuation from the stope;
- the type of support of the space resulting from cutting and removing useful mineral substances from the stope;
- the type of support of the space in the intersection area;
- the type of pressure guiding of the exploited space;
- type of supplying the work stope.

Graph-analytical model of technology generation and technological variants of face

By knowing the method or methods of exploitation of a reserve unit and the production process that takes place within each exploitation method, it is necessary to prepare a list of the following information for the production process operations:
- types of machines (simple and complex), installations, devices, hand tools that can be used;
- types and specific consumption of raw materials, consumed materials;
- types of energy used;
- the standard work teams, required in a face.
The database required to prepare the above lists includes:
- the geometric parameters describing the reserve unit;
- the geometric parameters describing a face field;
- the qualitative parameters of the useful mining substance from reserve unit;
- the physical-mechanical parameters of the useful mining substance belonging to the face field;
- the physico-mechanical parameters of the rocks in the roof, respectively the floor of the reserve unit.

In Romania, in order to be able to use a method of exploitation or a technological variant of exploitation, it is necessary for the research and design institutes in the mining field to elaborate "framework projects for the exploitation methods used in Romania", which then are subjected a process of endorsement by specialized bodies (bodies within the ministries of economy, labor, environment and health).

The necessity of elaborating such documents was imposed by the law in force and by the unwanted events appeared in the practical activity.

The existence of these documents obliges users at:
- creating normal working conditions in the face;
- prevention of mechanical, technical and human accidents;
- observance of the order of exploitation of the face fields within a reserve unit;
- protection of unexploded reserve units.

The information used in the elaboration of the grafo-analytical model for the generation of technologies or technological variants of face, are taken from the following data sources:
- "Indicators of norms and unit prices for mining construction works and fitting underground works C.M 1982";
- "Framework projects for the exploitation methods used in Romania";
- the specialty literature in the field;
- the technical sheets of the equipment, respectively of the installations, showing the technical and economic characteristics of using;
- work reports edited during the exploitation of the various face fields.

The following groups of information are taken from these documents:
1. The natural conditions in which each method of exploitation is applied, in part;
2. The technologies, respectively technological face variants that can be used by a method, or a variant of a method of exploitation.

**PRACTICAL EXAMPLE**

Having connections matrices of the conditions type for each mining field belonging to the Jiu Valley Carboniferous Basin, it is possible to generate sets of conditions in which a reserve unit belonging to a mining field can be found (the carboniferous deposit of the Jiu Valley is administratively divided into 13 mining fields).

Attaching to the connections matrix of the conditions type, the solutions matrix, depending on a set of known conditions, can be generated exploitation solutions for any of the unit of reserves belonging to a mining field, and based on a decision criterion, can be identified the most favorable action variant.

Knowing the exploitation method, to capitalize a reserve unit, it is necessary to choose the technology and the technological abatement variant that can be used.

Next, there is presented a graph-analytical model for the choice of technology, respectively of technological variant used by the methods of exploitation of the coal from the Valea Jiului coal deposit, Romania.

The elaboration of the graph-analytical model was based on information taken from the database presented at point 2.3.

Due to the complexity of the condition-solutions graph, it does not presented. Only the analytical transposition of the graph is presented in Annex 1, as a table called the "The connections matrix conditions-solutions".

The information used in elaboration of the graph-analytical model is:
- the thickness of the layer, m;
- the average slope of the layer °;
- specific admissible pressure on the floor, daN/cm²;
- the variation of inclination in the feed direction of the face, °;
- the variation of the slope of the face line, °;
- type of support;
- type of the machine used for cutting coal;
- the type of conveyor in the face.

In the matrix of connections presented in Annex 1, values for each variable (stimuli and reactions) are assigned in part.

Construction of matrix "conditions-solutions" for the set of established conditions

Knowing the following data:
- direction length of the abatement field, Lca = 750m
- the length of the face Lf = 120m
- front work height Hf = 3.5 m;
- the average slope of the layer \( \alpha = 15^\circ \);
- specific pressure accepted on the floor area, Pc, 35 daN / cm²
- the variation of inclination on the feed direction of the face, ±10°;
- the variation of the inclination of the face line, \( \alpha \leq 15^\circ \).

is required:
- type of support;
- type of the machine used for cutting coal;
- the type of conveyor in the face.

In the matrix from Annex 1, the following codings are identified for the above parameter values:
- \( C_{0106} \), corresponding to the thickness of the layer \( \leq 3.5 \) m;
- \( C_{0201} \), corresponding to the average inclination of the layer \( \alpha_{med} < 15^\circ \);
- \( C_{0304} \), corresponding to the specific pressure on the floor \( \leq 35 \) daN/cm²;
- \( C_{0402} \), corresponding to the inclination of the front forward direction \( \pm 10^\circ \);
- \( C_{0502} \), corresponding to the inclination variation of the front line \( \alpha \leq 15^\circ \).

The construction of the matrix "conditions-solutions" for set of established conditions, Annex 2, requires the following steps:

In Annex 1, for each variable "categories" are identified the values that are appropriate to the natural conditions in which the face field is found, for which the technology is chosen, respectively the variant of the face technology. The process of constructing the matrix "solution-conditions" goes through the following steps:

- step 1. For the first variable "categories", is identified its line, \( C_{0106} \), it goes to the right and for the group of columns encountered (\( C_{01}, C_{02}, ..., C_{05} \)) all columns that do not describe the unit of deposit are deleted, leaving the columns: \( C_{0106}, C_{0201}, C_{0304}, C_{0402} \) and \( C_{0502} \).

We move further horizontally and we reach the area of the variables "solutions" (\( V_{06}, V_{07}, V_{08} \)), where, all the columns that have in the boxes, the value "0", will be deleted.

- step 2. Go to the "variable code" column, identify for each variable "categories" the values "\( C_{0106}, C_{0201}, C_{0304}, C_{0402} \) and \( C_{0502} \)”, which remain, and the rest of the lines are deleted.

Descending in the area of variables "solutions" (\( V_{06}, V_{07}, V_{08} \)), retains only those lines that have the codes according to the not deleted columns in the area of the variables "solutions", and the rest of the lines are deleted.

Remark. Codes of the columns that have not been deleted must be the same as the codes of the remaining lines.

Identification of the technology or technological variant for the exploitation of a field of face

There is the question of identifying the technology or the optimal technological variant of exploiting a face field, when we know: the natural conditions describing the reserve unit in which the face field is located and geometric parameters of the face field.
In order to be able to determine the optimal exploitation technology of the face field, the values of the function of specific costs for each node in the graph must be established, values that are dependent on the set of conditions and the values taken by each node.

The values of the functions of specific costs, set for each node from the area of variables "solutions" are determined by the statistical processing of the data obtained in practice. Specific costs are determined for the following operations: supporting the ceiling and directing pressures, cutting and charging the coal, and transporting coal to the working stope. These specific costs include:

- the specific costs with the raw materials, the materials used for the accomplishment of each complex of operations, in part;
- specific labor costs;
- the specific costs with the damping of the equipments used in the execution of each complex of operations, in part.

In Annex 3, for each node belonging to columns of nodes from the area of variables "solutions" that form the graph of solutions, was added the values of the specific costs.

It can be seen that for column of nodes "9" ("STOP") the value of the node "01" has the value of the specific cost equal to "0".

There are situations when for a given set of conditions, there is no compatibility between the nodes of two adjacent columns.

In order to build the graph, for a given set of conditions, a fictitious node is inserted for the column that does not match the nodes of the neighboring column, the node having the value of the unit costs function "0". The fictitious node will be compatible with all the nodes in the graph.

Remark. Determining the optimal technological variant to exploit a face field can also be done depending on other types of specific functions of evaluating the execution variants (time, productivity, production, etc.)

The Optimal Solution Algorithm

As the matrix in Appendix 4 and the graph in Appendix 3 are known, the critical path, which may have a maximum or minimum value, (in this case the evaluation functions refer to casts, resulting in a problem of minimum) values given by the evaluation functions of solutions variants generated by the matrix presented in Annex 4.

The algorithm used is taken from the critical path analysis method and complemented with the verification of the existence of compatibility between the nodes of the non-adjacent columns describing a technological variant of face used to extract a face field.

The algorithm for determining the optimal variance after an evaluation function involves the following situations:

- situation 1: we identify the first column of the nodes belonging to the matrix "conditions" (in this case, the matrix is the column $C_{0106}$), we descend to line $C_{0001}$ and we find the value "0" in the box from this position; it goes to the right and in the box next to the 00 column the value "01" is inserted, representing the value of the variable $C_0$; we continue moving right to the column "Value of critical road" where we enter the value "0" in the respective box.

We return to column $C_{0106}$ and descend to line $C_{0106}$; we continue to the right until to column 01. In box $C_{0106-00}$ the value in box $C_{0001-00}$ is copied and in box $C_{0106-01}$ the code 06 is writed, representing the first line from the lines group $C_0$. We move to right until to the column "Value of critical road" and enter the value from box $C_{0001-C_{0106}}$ in the respective box, i.e. the value "0".

Then we identify column $C_{0201}$, we move on column until to the $C_{0106}$ line, where we identify "0"; we continue descending to line $C_{0201}$, then we follow this line until the column 02; in boxes $C_{0201-00}$, $C_{0201-01}$, we enroll the values above, in box $C_{0201-02}$. We enter the code of the first line belonging to the $C_2$ lines group, that is 01. We continue moving to the right until to the column "Value of critical road", and in that box we insert the value from the box $C_{0106-C_{0201}}$, which is "0".

The same goes for the columns: $C_{0304}$, $C_{0402}$ and $C_{0506}$.

Remarks.

1. The path of the critical road in the portion of matrix "conditions" is $C_{0001-C_{0106-C_{0201}-0304}-C_{0402-C_{0506}}}$, and the value of the critical road on this section is "0".
2. On this critical road section there is no question of the existence of compatibility between the nodes of road, because it was built in this way.

-situation 2. It refers to determining the critical road section of the "solution-solution" connections matrix.

It identifies the first column in the column group belonging to variable \( V_{06} \), namely column \( V_{0601} \). It descends on column \( V_{0601} \) to line \( C_{006} \), where is found the value 7. It continues the lowering until to line \( V_{0601} \); it moves on this line until to column 06. In the boxes located at the intersection of line \( V_{0601} \) with columns 00..05 it copies the codes located above them, i.e. 01-06-01-04-02-06; in the next box is written the value "01", i.e. the code of the first column from the group of columns of the \( V_{06} \) variable. It moves further until to the column "Value of critical road", and in the respective box is written the result of the sum between the value from the box \( C_{0506} \cdot V_{0601} \), which is "7" and the value from box \( C_{0506} \). "Value of critical road", which is "0", i.e. \( 7 + 0 = 7 \).

Checking the compatibility of the \( V_{0601} \) node (value) with the nodes (values) of the road \( C_{0001} \) - \( C_{0106} \cdot C_{0201} \cdot C_{0304} \cdot C_{0402} \cdot C_{0506} \).

To verify these compatibilities, it will proceed as follows:

- it descends on the column \( V_{0601} \) and it checks if in the boxes resulting from its intersection with the lines \( C_{0106} \), \( C_{0206} \), \( C_{0304} \), \( C_{0402} \), there is the value "1"; if the answer is yes, it results that \( V_{0106} \) is compatible with all the nodes of the road.

After applying the working mode from the column \( V_{0601} \) and on the rest of the columns belonging to the column group \( V_{06} \) we obtain the values enrolled in the lines of the variable \( V_{06} \).

After completing columns in column group \( V_{06} \), it moves to column group \( V_{07} \).

It identifies the first column, \( V_{0701} \), it descends until to lines of group of lines \( V_{06} \), where we encounter values 3, 3, 3, 3, 3, ∞. Each of these values it adds to the values corresponding to the boxes located at the intersection of these lines with the column "Value of critical road", namely 3 + 7 = 10, 3 + 8.5 = 11.5, 3 + 9 = 12, 3 + 10 = 13, 3 + 12 = 15, ∞ + 15 = ∞.

From these values, it choose the lowest value, i.e. "10", the value of the amount obtained on line \( V_{0601} \).

It checks that the value of the node "$V_{0701}$" is compatible with the nodes of the road enrolled on the line of variable \( V_{0601} \), which is \( C_{0001} \cdot C_{0106} \cdot C_{0201} \cdot C_{0304} \cdot C_{0402} \cdot C_{0506} \cdot V_{0601} \).

The check can also be done by going down on the column \( V_{0701} \) and it checks if the value in the cassettes encountered at the intersection of this column with the variables \( C_{06} \), \( C_{01} \), ..., \( C_{05} \) is 1, (variable \( V_{0601} \) is compatible with \( V_{0701} \) by constructing the graph); if the answer is "yes", then the road is admitted.

Value "$10$" is written in the box \( V_{0701} \) - "Value of critical road". In the boxes resulting from the intersection of line \( V_{0701} \) with columns 00-06, the route of the road corresponding to the line giving the minimum amount calculated above is entered, this being located in the line \( V_{0612} \), namely 01-06-01-04-02-06-01, and in the the next box is written the end of code of the line \( V_{0701} \), i.e. "$01".

In the same mode, it continues also, with the rest of the columns of the \( V_{07} \) column group.

After completion of column group \( V_{07} \), it moves to column group \( V_{08} \).

It goes down on the column \( V_{0801} \) until to the \( V_{07} \) lines group, where we meet the values 2.5, 2.5, 2.5, 2.5, ∞; these values are added with the values written in the boxes resulting from the intersection of the column "Value of critical road" and the lines of the \( V_{08} \) variable, namely: 2.5 + 10 = 12.5, 2.5 + 11.5 = 14, 2.5 + 11 = 13.5, 2.5 + 12 = 14.5, ∞ + 13.5 = ∞. Now, the minimum value that is given by line \( V_{0704} \) and is equal to 12.5, is chosen.

It verifies the compatibility of node \( V_{0801} \) with the rest of nodes of the admitted road, identified on line \( V_{0701} \). It descends on column \( V_{0801} \), up to line \( V_{0601} \), it is noticed that in box \( V_{0601} \cdot V_{0801} \) the value is 0, it results that the node \( V_{0801} \) is not compatible with node \( V_{0601} \), and it results that the road is non-admitted.

The following value by size is searched, this being 13.5 and belonging to line \( V_{0704} \). It checks that \( V_{0801} \) is compatible with the nodes of the enrolled road on line \( V_{0704} \); it observes that they are compatible, and it results that the road is admitted.

It copies the path enrolled on line \( V_{0704} \), on line \( V_{0801} \), to which code sequence 01 is added, and in box \( V_{0801} \) - "Value of critical road" the value 13.5 is entered.

It continues the same, for the rest of the columns the compatibilities it checks, and the results are written in the matrix of Annex 4.

The results are presented in the matrix in Annex 4.
After the column groups belongings to the variables “solutions” have been ended, the “STOP” column is reached with a single value of "01". It goes down on this column until to the lines group of $V_{08}$ variable, where the value "0", in all the boxes resulting from the intersection of the $V_{09}$ column with the lines of $V_{08}$ variable, is found. We add these values to the values of admitted road written on the lines of $V_{08}$ variable, and resulting the values 13.5, 12.75, 14, 15.55, 12.25. From these values, it chose, the minimum value, which is 12.25, corresponding to the $V_{0805}$ line, and resulting in the critical road in the graph, admitted road, which gives the optimal technological variant of the face field exploitation. The critical road path and its value go over the line "Critical road value". The route of the critical road is 0001-0106-0201-0304-0402-0506-0601-0701-0805-0901, and its value is 12.5 monetary units.

CONCLUSIONS

The graphical part of the graph-analytical model allows the generation of all possible paths. This is possible because the graph only solves the compatibilities between the values of a variable with the values of the adjacent variables (neighbors).

The analytical part of the model allows the user to reduce at minimum the number of the roads generated by a graph. This is possible due to the fact that the matrix of connections allows verification of the compatibilities of values of a variable with all values of the non-adjacent variables.

The graph structure model allows:
- synthesis of the description of the natural conditions describing the surrounding environment of a deposit, respectively of a reserve unit belonging to the deposit;
- synthesis of the component elements of the technologies generation solutions, respectively the technological variants of exploitation;
- allows rapid updating with new information from both the "conditions-conditions" group and the "reaction-categories" group;
- allows the establishment of the technology and technological variant of face, when known: the natural conditions describing the rock massif around the deposit, respectively a reserve unit to be capitalized;
- the time to determine the optimal solution is relatively small.

The accuracy of the results obtained with this method is influenced by:
- the quality of research underlying the matrix "conditions-condition";
- the quality of the research underlying the "solution-solution" matrix development;
- the quality of the research underlying the realization of the links between the two matrices;
- the veracity of the database used to determine the cost functions attached to the node columns belonging to the "solution-solution" matrix.

The model is a very high precision instrument for the mining decision process in the mining design phases as well as in the practical work.

The presented model can be transposed into a computer application.

This type of model can be applied in any field of activity (politics, health, scientific research, economy, culture, etc.)

References

[6] Dobrițoiu, N., Mangu, I.S., Determining the optimal solution for the execution of underground mining constructions, given the geological and mining conditions, 24th World Mining Congress, Oct. 18-21, 2016, Rio de Janeiro, Brazil, PROCEEDINGS - UNDERGROUND MINING.


Annex no. 1

THE CONNECTIONS MATRIX FOR GENERATING THE FACE TECHNOLOGIES
"GENERAL CASE"

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THE GRAPH CONDITIONS-SOLUTIONS FOR GENERATING THE FACE TECHNOLOGIES
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SEVERAL CONSIDERATIONS ON APPLICATION OF UNDERGROUND COAL GASIFICATION (UCG) WITH HIGH EFFICIENT AND SAFETY

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Jiazuo, Henan, China, 454-003

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ABSTRACT

Underground coal gasification (UCG) is a technique to extract energy from coal in the form of heat energy and combustible gases through the chemical reactions in the underground gasifier. This technique enables to recover the coal energy resources abandoned in the underground for either technical or economic reasons by gasification in-situ. However, several considerations such as ignition process to start, control of gasification reaction and area, and extinguish system have to be needed to develop UCG system with high efficient and safety. This paper describes our activities for the development of a UCG system with high efficient and safety: an innovative ignition system by using semiconductor laser, a real-time monitoring system by using acoustic emission monitoring, and a rapid extinguish system by injecting inert gas.

KEYWORDS

Acoustic emission, Inert gas, Laser ignition, Underground coal gasification

INTRODUCTION

Underground Coal Gasification (UCG) is a technology to recover coal energy as the combustible gas from the coal seam in underground by means of boring excavation. UCG has several benefits such as recovering the coal energy from abandoned coal seam due to economic and technological reasons, lower capital/operating costs, no surface disposal ash, and no human labor underground. In addition, UCG has the possibility to combine with carbon capture storage (CCS). A conventional UCG systems is gasification of the coal in a gasification channel made by a well linking the injection and production wells (Kacur et al., 2014; Bhutto et al., 2013) as shown Figure 1 (a). This system is suitable for thin, flat, and deep coal seams. An alternative UCG system must therefore be developed in Japan because geological conditions are complicated, with the existence of faults and inclined coal seams. Given this background, we are developing a co-axial UCG system that is compact, safe, and highly efficient as shown Figure 1 (b) (Su et al., 2017; Hamanaka et al., 2016). The co-axial UCG system uses only well drilling and a double pipe. Gasification agents are injected from the inner pipe to expand the combustion zone. The production gas is recovered from the outer pipe.

![Figure 1](image_url) – Concept of UCG system.

The process of UCG consists of several steps: preparing the well along with coal seam as an injection and a production well, in-situ ignition of coal to promote the gasification reaction by increasing the temperature of coal, control of gasification reactions, and extinction of underground fires. For UCG process, the chemical reactions to produce the combustible gases mainly consisted of carbon monoxide (CO), hydrogen (H₂), and methane (CH₄) are promoted under high temperature. Temperature field for these reactions can be prepared by ignition of coal in underground and supplying enough oxidants. In laboratory experiment or field trial of UCG, numerous methods were
adopted for ignition of coal: gas burner, electric system, pyrotechnic charge, heated coal, and other chemicals (Mocek et al., 2016; Wiatkowski et al., 2015; Cui et al., 2014; Kapusta et al., 2013; Stanczyk et al., 2010; Prabu & Jayanti, 2011; Yang et al., 2008; Yang et al., 2007). After the ignition stage, the injection rate of oxidants have to be adjusted to heat up the coal seam and accumulate enough thermal energy to promote gasification reaction (Stanczyk et al., 2011). Increasing the injection of oxidants improve the gasification efficiency due to the increase of the reaction temperature and the expansion of the gasification area (Hamanaka et al., 2017). On the other hands, some of environmental issues have to be cared such as gas leakage, groundwater pollution, and surface subsidence (Imran et al., 2014; Bhutto et al., 2013; Kapusta et al., 2013; Kapusta & Stanczyk, 2011). Therefore, the monitoring and control of gasification area is another option that can be considered for the improvement of the overall process. From another point of view, the fire extinguishing technique should also be discussed to reduce the risk of environmental hazard due to not optimal operations. Additionally, it is also required to decrease the temperature in underground rapidly up to the temperature that coal is not reacted because the temperature of coal seam exceeds 1,000 degrees Celsius.

From these backgrounds, this paper describes our activities about ignition system, monitoring of gasification area, and fire extinguishing process for the development of a UCG system with high efficient and safety.

IGNITION PROCESS

Ignition is an important process to rise coal temperature to start UCG process since product gas can be generated due to the promotion of chemical reactions around the high temperature area. In this study, the laser ignition system with oxygen supply was adopted to ignite the coal with easy, safety, and quickly. Laser technologies are used in various industry fields. The idea of laser ignition of coal is from laser in material processing which is a technology to generated thermal energy of the material by emitting high energy density beam. The energy density of laser is increased with focusing light by a condensing lens. The processes are divided into three major classes, namely only heating (without melting/vaporizing), melting (no vaporizing) and vaporizing. In other words, it is possible to use this technology for coal ignition by adjusting the energy density of laser properly.

Laboratory Scale Experiment

The coal ignition experiments in a laboratory scale were carried out by emitting the laser to coal block under the several laser energy and distance between them. We used semiconductor laser equipment which laser emission wavelength was 808 nm (M710A45; Omron Laserfront Inc.). As shown in Table 1, the coal can be ignited by using laser under certain conditions. The duration needed for the coal ignition is the fastest under the short distance from the lens to coal block, meaning that only 0.5 second is enough to confirm the coal ignition when the distance is 100 mm. 20 W of the laser is not sufficient to ignite coal when the distance from lens to coal block is 200 mm, but the coal is ignited if the output is increased to 30 W. It can be observed that the intensity of laser beam increases with increasing the output of laser (Figure 2). This fact indicates that the coal can be ignited by increasing the output of laser even if the distance is long. Additionally, the duration of coal ignition is shortened by decreasing the distance from lens to the coal surface when the output of laser is same based on the results of 150 mm and 200 mm distance from the lens to the coal surface under 30 W of output. On the other hands, the diameter of beam is increased with an increase of the distance, meaning that it is possible to rise the temperature of coal surface over a wide range due to the large diameter of beam. From these results, the coal ignition with the laser can be conducted efficiently by adjusting the output of laser and the distance from the lens to the coal surface.

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Figure 2 – Laboratory experiment about laser ignition of coal under the different output (the distance is same in all figures).

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</table>

Large Scale Experiment and Field Trial

As next step, the laser ignition of coal was tried in a large scale UCG experiment (the length of experimental well is 2.5 m) and a field UCG trial (the length of experimental well is 18 m). Figures 3 (a) and (b) show the outline of laser ignition system. An endoscopic camera to check the coal ignition and an injection pipe to supply oxygen were inserted into the experimental well. The conditions for laser ignition were determined based on the pre-experiment as follows; output was 45 W, the distance from the lens to coal surface is 150 mm, and oxygen flow rate is 10 L/min (use 90% concentration of oxygen). As the results of the experiments, large scale UCG experiment succeeded the coal ignition by laser beam, but field UCG trial did not. The biggest different of both experiment was the moisture condition of coal surface. The conditions of coal seam in the field was wet due to the rainfall before the field UCG trial began, meaning that there is a possibility to consume the heat energy not for ignition of coal but for evaporation of water. Additionally, the coal surface that can be heated by laser beam is small compared with other ignition ways. In UCG process, the coal combustion reaction is promoted primary in order to increase the coal temperature, and then the gasification reaction is occurred under the high temperature field. Considering these mechanism, it is concluded that the coal combustion is difficult to propagate to the surrounding due to the endothermic process of water vaporization. Based on these considerations, the laser ignition of coal seam in the field UCG trial retried after the coal surface is dried by ventilation a hot air from the surface. Consequently, it was confirmed the coal ignition by endoscopic camera. To sum up, the laser ignition can be adopted efficiently when the coal seam is dry because the coal combustion process is difficult to propagate under the wet coal conditions.

Figure 3 – Laser ignition system
MONITORING OF GASIFICATION AREA

Gasification reaction in UCG process is promoted by enlargement of the oxidation surface around the gasification channel with crack initiation and development inside the coal seam. Fracturing activities inside the coal seam are accelerated with an increase of thermal stress caused by exothermic reactions and heat transfer, as a result, gasification reaction and cavity growth is promoted. Therefore, techniques to evaluate the fracture activity around the gasification area have to be developed for precise control of coal gasification in-situ and minimizing environmental impacts. Considering the UCG process is strongly related the fracturing activity, acoustic emission (AE) monitoring should be an effective method for the evaluation of fracturing activity around the gasification zone. We discussed the availability of AE monitoring to evaluate fracturing activity and cavity growth during UCG process.

Material and Methods

The model UCG experiment was conducted in an artificial coal seam. The typical dimensions and structure of the gasifier applied in the simulations of underground gasification are presented in Figure 4. The co-axial system is simulated in this experiment. The outlet position of gasification agents can be adjusted by the inner pipe with controlled movement. The artificial coal seam size was 0.55 × 0.60 × 2.74 m. During the experiment, temperature and acoustic emission (AE) were monitored to visualize inner part of the coal seam by using type K thermocouples (SUS310S; Chino Corp.) and piezoelectric acceleration transducers (620HT; Teac Corp.), respectively as shown in Figure 5.
Results and Discussion

Based on the temperature data, the two-dimensional maximum temperature profiles are plotted for several experimental periods in Figure 6 (a), representing the maximum temperature distribution in a cross-section of a horizontal well. Each figure shows results in the different position of an injection pipe. Additionally, the results of AE source locations are presented in Figure 6 (b). Using the onset time of AE waveforms recorded during UCG model experiments, AE source locations for each model were calculated using least-squares iteration algorithm. The extent of damage, i.e. the relative energy emitted from cracking, can be indicated by the sphere sizes. From the results in Figure 6 (a), the high temperature area is moved when the injection position is changed, meaning that the gasification area is moved. This fact means that the gasification reactions are activated around an injection pipe. Gasification reactions are promoted under the high temperature as a result of oxidation reaction. Therefore, the gasification area is expanded around an injection pipe because the most of oxidant is consumed near the injection pipe. From a different perspective, it is possible to control the gasification area with changing the position of the injection agents. The results of AE source locations agree with those obtained from temperature profiles, meaning that many AE events are occurred around the gasification area. These AE generations apparently result from crack initiation and extension around the gasification area under the influence of thermal stress. Additionally, AE sources can be obtained in real time during UCG process. Accordingly, monitoring the AE during the UCG process could be a useful technique to estimate the damage zone development in real time and inform the operators when the excessive damage occurs.
As the coal temperature during gasification process exceeds 1,000 degree Celsius, it needs to be considered to extinguish a fire and decrease the temperature to avoid gasification reaction more in coal seam as a countermeasure in order to reduce the risk of pollution to surrounding layer at the end of UCG operation. The fire triangle consists of three components viz., fuel, oxygen and source of ignition. If one of the components can be removed from the triangle it is impossible to ignite or sustain any fire (Ray, S. K. & Singh, R. P, 2007). Removal of fuel, coal, is impossible in the underground. Therefore, the use of inert gases to control underground mine fires is an effective way because its injection can be reduced the oxygen percentage and the gas can absorb the heat from the source of ignition by ventilation (Figure 7). Nitrogen (N₂) is the gas commonly used for extinguishing a fire. On the other hands, the use of carbon dioxide (CO₂) can be another option as an inert gas because abundant CO₂ is also produced from the UCG process. This concept is matched with the CCS with UCG, meaning that CO₂ it is possible to storage CO₂ in the gasification area together with extinguishing a fire and decreasing the temperature of coal seam. From these backgrounds, the effect of an inert gas injection is discussed as an extinguishing process of UCG by means of a laboratory scale experiment.
Material and Methods

Figure 8 shows image of the laboratory scale experiment to discuss an extinguishing process of UCG. In this experiment, $N_2$ and $CO_2$ gas were used as an inert gas. The flow rate was 0.5, 1.0, 2.0 L/min. At first, 25 g of crushed coal adjusting the particle size from 1 mm to 4 mm were filled into the central part of silica tube. In order to simulate the heated coal, the coal temperature was increased by the heater up to 700 degree Celsius with injecting the 1.0 L/min of $N_2$ gas to minimize the effect of oxidation reaction. The temperature was monitored in the three points of the sample; injection side (T1), central part (T2), and discharged side (T3). After the temperature reached 700 degree Celsius and condition became stable in T2, an inert gas of 20 degree Celsius was injected to the coal with turning off the heater. The injection process continued until the temperature was lowered to 100 degree Celsius.

Results and Discussion

Figure 9 (a), (b) shows the results of coal temperature with $N_2$ and $CO_2$ injection when the injection rate is 1.0 L/min. Based on the results of temperature, the coal temperature in injection side is decreased faster than that in discharged side. This result leads to decrease coal temperature along with the gas stream because the temperature of an inert gas is high in discharged side due to the absorption of heat in the injection side. Additionally, the decrease of coal temperature with $CO_2$ injection is faster than that of $N_2$ injection in any temperature monitoring point. Table 2 show the
decrease rate of coal temperature under the different range of coal temperature in T2. This is explained by the molar heat capacity at constant pressure of the gas. The molar heat capacity of CO₂ gas is higher in any temperature condition compared to N₂ gas (Table 3), meaning that the CO₂ gas absorb more heat from the heated coal to increase the gas temperature.

![Graph showing the decrease rate of coal temperature under different injection gases.]

**Table 2 – Decrease rate of coal temperature under the different range of coal temperature in T2**

<table>
<thead>
<tr>
<th>Coal temperature (°C)</th>
<th>N₂ injection (°C/min)</th>
<th>CO₂ injection (°C/min)</th>
</tr>
</thead>
<tbody>
<tr>
<td>700 - 600</td>
<td>30.63</td>
<td>36.78</td>
</tr>
<tr>
<td>600 - 500</td>
<td>29.88</td>
<td>35.54</td>
</tr>
<tr>
<td>500 - 400</td>
<td>25.96</td>
<td>31.52</td>
</tr>
<tr>
<td>400 - 300</td>
<td>20.59</td>
<td>24.67</td>
</tr>
<tr>
<td>300 - 200</td>
<td>15.28</td>
<td>17.65</td>
</tr>
<tr>
<td>200 - 100</td>
<td>9.41</td>
<td>9.96</td>
</tr>
</tbody>
</table>

**Table 3 – Molar heat capacity at constant pressure of the gas (Japanese thermophysical property academic meeting, 2008)**

<table>
<thead>
<tr>
<th>Gas temperature (°C)</th>
<th>Molar heat capacity at constant pressure (J/mol K)</th>
</tr>
</thead>
<tbody>
<tr>
<td>700 (800)</td>
<td>N₂</td>
</tr>
<tr>
<td>600</td>
<td>30.10</td>
</tr>
<tr>
<td>500</td>
<td>29.57</td>
</tr>
<tr>
<td>400</td>
<td>29.26</td>
</tr>
<tr>
<td>300</td>
<td>29.15</td>
</tr>
<tr>
<td>200</td>
<td>29.20</td>
</tr>
</tbody>
</table>

As next, the effect of injection flow rate on the extinguishing process. Figure 10 (a), (b) shows the results of monitoring temperature in T2 under the different injection gas flow rate. It can be said that the increase of injection rate accelerate the extinguishing process due to the rapid decrease of coal temperature. Additionally, it can be approximately estimated that the decrease rate of coal temperature is increased by 30~40% for CO₂ injection when the injection rate is two times while it is 20~30% for N₂ injection based on Figures 11 (a) and (b). This finding also shows the advantages to use CO₂ gas as an extinguishing process of UCG. Additionally, the boudoir reaction is expected to be attribute for the rapid extinguishing process if the coal temperature is more than 900 degree Celsius due to heat consumption by its endothermic reaction. In summarise, it can be considered that efficient extinguishing process can be achieved by utilizing CO₂ gas recovered from UCG process.
Figure 10 – Results of coal temperature in T2 under the different injection gas flow rate.

(a) N$_2$ injection                                     (b) CO$_2$ injection

Figure 11 – Results of decrease rate of coal temperature under the different temperature range.

(a) N$_2$ injection                                     (b) CO$_2$ injection

ACKNOWLEDGMENTS

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REFERENCES


DEVELOPMENT OF COMPACT AND HIGHER EFFICIENCY CO-AXIAL UCG SYSTEM AND ITS TRIAL EXPERIMENTS

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ABSTRACT

Underground coal gasification (UCG) is a technique used to recover coal energy by the \textit{in-situ} conversion of coal into gaseous products. The UCG enables recovery of coal energy from unused coal resources abandoned under the ground for either technical or economic reasons.

The technique of a conventional UCG system is to gasify the coal in a gasification channel made by a well linking injection and production wells. A conventional UCG system can be adopted efficiently for thick, flat, and deep coal seams. Considering these limitations, it might be difficult to adopt conventional UCG systems in Japan because of geological conditions that are complicated by the existence of faults and folds. Therefore, a co-axial UCG system that is compact, safe, and highly efficient is suggested as an alternative UCG system. The co-axial UCG system is flexible for adoption under complicated geological conditions because this system uses only well drilling with a double pipe: oxidants are injected from an inner pipe. Gaseous products generated in the coal seam were collected from an outer pipe.

This paper describes the process and the results of field trial experiments using the co-axial UCG system conducted at the Mikasa UCG coal mine in Japan.

KEYWORDS

Underground coal gasification, Co-axial UCG system, Environmental monitoring, Production gas

INTRODUCTION

Site experiments were conducted for the five-year research project named underground coal gasification (UCG). Starting from a coal seam outcrop investigation, a mining area was established and mining concession (exploration right) was acquired. A management plan, and security and management regulations were established to conduct small-scale co-axial type UCG experiments.

This project consists of [Advanced UCG system development] sub-project for developing a compact and efficient gasification method, a coal ignition system, and a method for estimating gasification and combustion zones, a [Highly efficient utilization of production gas and heat] sub-project for investigations of detoxification, separation, storage and utilization of the production gas, and a [Environment monitoring and assessment] sub-project for monitoring influences on atmosphere, land surface, and under the ground surface.

In the experiments, time for gasification is limited to 24 hr because of the short distance (12 m) between land surface and the position of gasification. As a result, experiments were performed and production gas was collected as scheduled. The production gas constituents were nearly the same as those of the past UCG experiments using lump coal. No influence of this gasification on the surrounding environments was noticed at all.

The following description presents an outline and accomplishments of this research project.

OUTLINE OF THE RESEARCH PROJECT

Object of Research and the Research System

The object of research set at the beginning was [To overcome remaining themes and to establish a compact, safe, highly efficient and environmentally friendly coal gas utilization technology as a local resource energy source, based on accumulated research outcomes of UCG and separation, recycling and transportation]
of the production gas. Later, [To propose regional regeneration project model for old coal mining area using UCG] was added. To accomplish these objectives, the following three sub-projects were established for this study:

1) Advanced UCG system development sub-project
2) Highly efficient production gas and heat utilization sub-project
3) Environment monitoring and assessment sub-project

With the [Advanced UCG system development sub-project], development of a method for improvement of gasification efficiency and development of visualization and monitoring technology of the gasification furnace and monitoring technology were promoted considering the perseverance of unused coal resources in Japan and devoting attention to a compact and co-axial type UCG system. Here, co-axial type UCG is the method shown in Figure 1 by which a double tube is inserted into one boring hole, an oxidizing agent such as air and oxygen is filled from the inner tube to collect the production gas between the inner tube and outer tube. An important shortcoming of this method is that the combustion and gasification zone in the coal seam expands, making gasification efficiency improvement difficult. Regarding the gasification efficiency of the co-axial type UCG and visualization and monitoring technology of the combustion and gasification zone, a method was established by UCG model experiments using a primarily artificial coal seam and obtained significant results (Su et al., 2017). In other words, results demonstrated that gasification efficiency is improved by a retreating type co-axial UCG method by which a co-axial tube is withdrawn while the oxidizing agent is being filled. For visualization of the combustion and gasification zone, it was confirmed that acoustic emission (AE) measurements for detection of the destruction in the coal seam in addition to temperature measurement is effective. At UCG site experiments, a co-axial type UCG system was used. Gasification of the unused coal seam was attempted.

With highly efficient production gas and the heat utilization sub-project, mainly separation and detoxification of the UCG production gas were performed. Discharge plasma that enables continuous processing was used. With this method, the production gas was introduced into an electrical discharge tube into which glass beads and alumina beads are filled to decompose the gas in part by discharging. Results show that this is effective for desulfurization of harmful hydrogen sulfide (Takahashi et al., 2017). At UCG site experiments, the production gas was sampled periodically to verify desulfurization effects (Takahashi et al., 2017).

For [Environments monitoring and assessment sub-project], a system for monitoring and assessment of influences of UCG on the environments was developed. At UCG site experiments, a continuous monitoring system for atmosphere around the co-axial hole, land surface, underground and groundwater was developed. Changes mainly before and after UCG were checked. Furthermore, influences on the area around UCG were checked before and after UCG site experiments.
Figure 2 presents the organization of this project. The research was promoted to attract cooperative participation by each institute, with the disused Mikasa coal energy research facility of Muroran Institute of Technology, located at Ikushunnbetu, Mikasa City serving as the hub.

![Research project cooperation system.](image)

**Status of UCG Experimental Site**

Figure 3 portrays the experiment site location. A coal outcrop is visible along the forest road at the south side of the state-owned forest of vertical shaft of Ikushunnbetu coal mine of the now defunct Hokkaido Colliery & Steamship company. Boring for geological investigation was performed here from the mountain side slope of the forest road. Finally, boring was performed at the nine locations depicted in Fig. 4. These boring holes were used later as the co-axial type UCG experimental holes and observation holes in which various sensors are provided. Results of this assessment revealed the coal seam inclination as about 70 degrees, with seam thickness of about 1 m. Weathering was significant including hanging wall and footwall. Many cracks were noticed. The highest heat generation in the coal seam was noticed at the area close to the boundary with the hanging wall and it was about 32 MJ/kg.

The groundwater level was checked in No. 8 pit and No. 9 pit, whereas the mouth of No. 1 boring hole shown in Fig. 4 is regarded as the 0 m level. The groundwater depths were, respectively, -23.55 m and -11.94 m. The water depths were 4.64 m and 0.39 m. Although the groundwater depth and water depth vary from season to season, the water level of No. 9 pit was 0 at the UCG experiment.

The coal seam permeability was measured later using No. 4 pit, which was used as the UCG co-axial hole. Figure 5 presents an example of chronological change of flow rate and pressure of the air injected. From this equilibrium pressure and equilibrium flow rate, average permeability was calculated using the following equation:

\[ K = \frac{f_0 P_{gs}^2 Q_{gs}}{\pi L (P_g^2 - P_b^2)} \ln \left( \frac{L + \sqrt{L^2 + 4r^2}}{2r} \right) \]

where \( K \) denotes permeability (m\(^2\)), \( T \) signifies the test section temperature (= 91°C = 282.25 K), \( T_s \) stands for the temperature in a standard condition (= 273.15 K), \( P_{gs} \) expresses the pressure in a standard condition.
Assumed outcrop line
Mikasa disused coal energy research facility
Experimental site
15 m × 30 m area along forest road (about 5 m wide)
Equipment is placed along the forest road

Forest road
UCG experiment site in state-owned forest (Ikushunbetu layer outcrop)

Vertical shaft of Ikushunbetu coal mine of Hokutan

(=101.325 Pa), $\mu$ represents the viscosity of air (=1.77×10^{-5} Pa•s (10°C)), $Q_{gs}$ expresses the air flow rate in standard condition (m³/s), $L$ denotes the test section length (= 5 m), $P_{g}$ denotes pressure at test section (Pa), $P_{0}$ represents atmospheric pressure (=101.325 Pa), and $r$ represents the boring hole radius (= 0.033 m).

Consequently, average permeability of 22.94 md was obtained. The possibility of gas leakage to the ground surface was a concern.

Figure 3 – Experiment site location.

Figure 4 – Arrangement of geological investigation boring (sectional view).

Figure 5 – Chronological change of flow rate of injected air (left) and pressure of injected air (right).

Acquisition of Mining Rights (Test Drilling)

When handling underground unused coal, the acquisition of mining rights (test drilling) is necessary even for experimentation. First, although it was intended to submit a special zone application to the special zone promotion office, structural reforms, cabinet secretariat before performing the UCG site experiment, it was judged by the government that experiments be allowed under the current law. Acquisition procedures for mining right (test drilling) were promoted. It was then judged that UCG differs from natural gas and others. The same procedures as those used for conventional coal mine development are acceptable. The chancellor of Muroran Institute of Technology acted as the mining right holder, the mining site was determined, and Muroran Institute of Technology Mikasa underground gasification coal mine was opened on October 24, 2016. Figure 6 portrays a map of the approved experimental drilling site.
PROCESS OF UCG SITE EXPERIMENT AND OBTAINED RESULTS

Process of Experiments

After acquiring mining rights (test drilling), management rules (draft) and security regulations were established. Figure 7 shows the experiment process. Finally, preparations for the experiment were started at the beginning of November 2016. Figure 8 (plan view) presents the arrangement of various sensors on the ground and experimental facilities. Table 1 shows the application of nine boring holes and buried sensors. The output of these sensors was transmitted to the university server via the ecoMo system (Nihonkasetu Corp.), which is explained later. The output was recorded by a group of data loggers in the measurement vehicle.

After preparations, the first UCG site experiment was performed during November 5–10, 2016. The bottom of No.4 pit was being moistened. Therefore, ignition was attempted by direct laser after the dried coal was input. The time from ignition until UCG completion was limited to about 24 hr considering the possibility of gas leakage to the land surface. However, the amount of water oozing from the No. 4 pit bottom was excessive. Continuous gasification was not possible. Therefore, the experiment was postponed until the following year.

The second UCG site experiments were performed during July 24 - August 3, 2017. To remove moisture at the bottom of the No. 4 pit, a cartridge heater was placed at the pit bottom. It was dried for about one week. Following this, ignition of the coal seam was attempted several times using direct laser and gas burner. Production gas was then collected. The experiment completed. Figure 9 portrays a clipping from the local newspaper. Subsequently, following the progress schedule shown in Fig. 7, carbon dioxide was injected.
for extinction. The interior of the No. 4 pit was cleaned and photographed using a borehole camera. Then cement with higher adsorptive property was filled into boring holes except for the No. 8 pit and No. 9 pit to restore it to its original state. Groundwater in the No. 8 pit was sampled further for one month to ascertain the water quality. Botanical exploration was also performed.

Table 1 – Use of boring holes and arrangement of sensors.

<table>
<thead>
<tr>
<th>Name of hole</th>
<th>Use</th>
<th>Contents</th>
</tr>
</thead>
<tbody>
<tr>
<td>No. 1 pit</td>
<td>Observation of environments</td>
<td>Gas concentration (CO, CO₂, CH₄)</td>
</tr>
<tr>
<td>No. 2 pit</td>
<td>UCG measurement</td>
<td>AE (Upper side of combustion)</td>
</tr>
<tr>
<td>No. 3 pit</td>
<td>UCG measurement</td>
<td>AE and temperature change (Lower side of combustion)</td>
</tr>
<tr>
<td>No. 4 pit</td>
<td>Ignition pit</td>
<td>Ignition, Gasification, Collection of generated gas, Measurement of temperature change</td>
</tr>
<tr>
<td>No. 5 pit</td>
<td>UCG measurement</td>
<td>AE and temperature change (Left side of combustion)</td>
</tr>
<tr>
<td>No. 6 pit</td>
<td>UCG measurement</td>
<td>AE and temperature change (Right side of combustion)</td>
</tr>
<tr>
<td>No. 7 pit</td>
<td>UCG measurement</td>
<td>AE ((Lower side of combustion)</td>
</tr>
<tr>
<td>No. 8 pit</td>
<td>Observation of environments</td>
<td>Water level, Water quality, Temperature, Gas concentration (CO, CO₂, CH₄)</td>
</tr>
<tr>
<td>No. 9 pit</td>
<td>Observation of environments</td>
<td>Water level, Water quality, Temperature, Gas concentration (CO, CO₂, CH₄)</td>
</tr>
</tbody>
</table>

Production Gas and Amount of Coal Reacted

Table 2 presents results of analysis of the production gas at UCG site experiment. For this experiment, the oxygen concentration of the oxidizing agent was adjusted to about 50% and injected at the flow rate of 50–60 L/min. Although heat generation is low because of the co-axial type UCG, major gas constituents were identical to those of indoor experiments and artificial coal seam experiments conducted to date.
Next, estimation of gasified coal amount around No. 4 pit was attempted. The following methods were used:

1) Estimation of combustion/gasification zone by AE measurement
2) Estimation of amount of coal reacted by production gas constituents (stoichiometry)
3) Estimation by image analysis of borehole camera

Figure 10 presents the chronological change of the AE event count recorded between August 2 and 3. AE detected by accelerometer provided at No. 3, No. 5, and No. 6 boring holes are shown. Someone was working around No. 4. The area enclosed within a frame represents the working noise. If these are excluded, then waveforms were recorded for only 12 events. Although a triaxial geophone was installed to the No. 2 and No. 7 pit, waveforms detected by these instruments were scarce. No seismic center was identified, probably because the AE generation from gasification was minimal. Therefore, estimation of the combustion/gasification zone by AE measurement was not possible.

Next, estimation of the amount of coal reacted by production gas constituents was attempted (Hamanaka et al., 2017). Results show that the amount of reacted coal was estimated as about 10.8 kg. About 6.0 kg of coal was used for ignition, but the actual coal gasified from the coal seam was estimated as 4.8 kg.

By way of comparison of images in the pit before and after UCG experiment, the amount of coal consumed is 2.1 kg, as estimated from changes in the volume. As shown in Fig. 11, the pit interior is deformed. This change corresponds to this portion. Therefore, if gasification from the periphery of the boring hole is taken into consideration, one can infer that coals in the range of 2.1 kg and 4.8 kg were gasified. This result seems appropriate from the fact that the coal consumption amount per unit time in the co-axial type UCG of identical scale is about 0.5 kg/hr. The amount of production gas is estimated as 2.9–6.7 m³.

Figure 9 – Clipping of Hokkaido Newspaper dated Aug. 4, 2017 (Morning edition, for Hokkaido, local news page 34).

<table>
<thead>
<tr>
<th>Table 2 – Production gas constituents.</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
</tr>
<tr>
<td>Heat Value (MJ/Nm³)</td>
</tr>
<tr>
<td>Laser ignition 1</td>
</tr>
<tr>
<td>Laser ignition 2</td>
</tr>
<tr>
<td>Gas burner ignition</td>
</tr>
</tbody>
</table>
Environment Monitoring System

An environment monitoring system was constructed based on the ecoMo system. Regarding observation items in the boring pit, in addition to AE measurement, temperature measured at three locations, one set of gas detection unit (three CO₂ gas detectors, three CO gas detectors, three CH₄ gas detectors), two water quality measurement devices, and two water level measurement devices were provided. Regarding land surface measurements, 10 sets of temperature measurement units, one set of gas detection (one CO₂ gas detector, one CO gas detector, one CH₄ gas detector, one H₂S gas detector), one meteorological measurement unit (one set of wind direction and wind speed measurement devices, one set of rain precipitation measurement devices), one web camera image unit, and one thermal image unit are used. Data from these instruments were collected as shown in Fig. 12, consolidated in PC in the measurement vehicle and stored in the database via a university server. Data in the server were confirmed from anywhere via the internet. Figure 13 portrays the browsing screen of this system.

Groundwater Survey

A groundwater survey was performed using mainly the No. 8 pit shown in Fig. 4. Regarding observation items of the groundwater by the ecoMo system, little change was noticed for temperature, pH, or dissolved oxygen before and after UCG site experiments. Water was sampled from the No. 8 pit on August 1,
August 4, August 6, August 20, and October 2 and was subjected to water quality analysis. As a result, all items such as benzene, chloroethylene, and phenols were less than the reference level. In other words, results show that the current UCG site experiment exerted no influence at all on the groundwater.

**Botanical Exploration**

For botanical exploration around the UCG site experiment, flora, vegetation map, vegetation section and quadrat survey were conducted before the experiment (November 13, 2015, July 21, 2016) and after the experiment (October 6, 2017). Plants of as many as 59 families and 114 species were confirmed before the experiment; 51 families and 100 species were observed after the experiment. This difference is regarded as attributable to differences of observation times; it is considered that the experiment exerted no influence on the flora. Regarding the quadrat survey, although a seasonal change occurred because of the difference of the time of survey, no marked change was noted in the species composition in each quadrat (1 m square) or change in external morphology (discoloration, mortality, etc.). From this, it is considered that the current experiment exerted no influence on the plants. Figure 14 portrays a vegetation map and sectional view at the site of the UCG experiment site.

![Vegetation map and sectional view at the UCG experiment site.](image)

**CONCLUSIONS**

This research project investigating the utilization of unused coal using UCG produced excellent results from three sub-projects, including those not introduced in this paper. Particularly, UCG site experiments were conducted using a co-axial type UCG system. Flammable gas was collected from the coal seam. Results were realized at a UCG coal mine after acquiring a mining concession (exploration right). It is meaningful that the results show a road to future UCG operation.

For future studies, we intend to promote preparations for a verification test aiming at utilization of UCG to be used as a local energy source (Fig. 15). To do so, boring technology is expected to play an important role. Sophisticated boring technology available at low cost is necessary for application of the retreating type co-axial UCG system developing higher gasification efficiency. It is considered that, if Coiled Tubing Drilling (CID) were realized at reasonably low cost, this technology could be realized soon.
This study was supported by grant funding from the JSPS Scientific Research Fund 15H02332 and was also supported with cooperation from Mikasa City, Hokkaido, special expenditure [Fulfillment of diversified academic research functions using features of universities] by Ministry of Education, Culture, Sports, Science and Technology, Center of Environmental Science and Disaster Mitigation for Advanced Research of Muroran Institute of Technology, Sanbi Kogyo, and JUCG. For promotion of UCG site experiments, Hokkaido Economy and Industrial Bureau of Ministry of Economy, Trade and Industry, Hokkiado Sorachi Development and Promotion Bureau and Forest bureau cooperated greatly. We wish to express our particular appreciation to them.

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CONCEPT ON REPEATED USE OF EXTRACTION WORKINGS WITH APPLICATION OF RESOURCE-SAVING BOLTING SYSTEMS

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ABSTRACT

In the current conditions, the task of providing highly efficient and reliable operation of coal mines is becoming increasingly important. It is formulated as an improvement of low-cost technologies for support setting and maintaining reusable extraction workings based on the control of the state of the rock massif enclosing them.

In terms of methodology, the components of the study are as follows: a generalization of the concepts on the displacement mechanism of coal-overlaying strata and the loading of the support setting system of extraction workings; SSS analysis of the rock massif in the vicinity of a mine working; SSS assessment of the load-bearing elements of the basic support setting system; development and substantiation of the improved support setting scheme; SSS calculation and analysis of the geomechanical system “massif – frame – combined bolting support”; establishing the patterns of the main geomechanical factors’ influence on SSS of the load-bearing elements of the combined bolting system at hardening of the extraction working arch and in combination with tent-shaped yielding support with elongated props (TSYS); development of a methodology for parameters calculating of the combined bolting system of the extraction working arch; carrying out and analysis of mine tests.

To assess the degree of the roof bolts’ loading, the corresponding criteria have been substantiated. The revealed patterns are the basis for the technique development of the bolting system rational parameter’s calculation. And this calculation technique is realized by two conditions: on the one hand, the roof bolts, together with the hardened roof rocks, should maximally unload the frame support to effectively restrict the cross-section loss in mine working; on the other hand, the roof bolts in the roof should work with maximal resistivity to rock pressure, that is, they should be set in a sufficient, but minimal quantity.

The area of mining and geological conditions has been determined, when there is a low efficiency of the roof rocks hardening with resin-grouted bolts within the central part of the mine working arch. To determine the boundaries of this area, the geomechanical parameter’s ratios were obtained for the most commonly used types of extraction workings’ cross-sections.

The practical implementation of the research is the calculation technique, which allows realizing resource-saving conditions of reusable mine workings’ maintenance.

KEYWORDS

EXTRACTION WORKINGS, BOLTING SYSTEMS, CALCULATION TECHNIQUE, RESOURCE-SAVING CONDITIONS, MAINTENANCE.

INTRODUCTION

Currently, there is an advanced and ever-growing trend of the gap between the high level of modern technologies in stoping faces and the possibilities of timely preparation for extraction panel mining. The increase in the length of the longwall to 300 m and the length of extraction panels to 3 km is one of the widely implemented ways to reduce this gap. The second very effective direction in reducing the volumes of driving and maintenance of site workings (by 1.7 – 1.8 times) with high-performance work of production units is to reuse them with stability increase by the rock massif hardening with combined bolting systems as part of the resin-grouted rope bolts. This significantly reduces material and labor costs due to border rocks involvement counteracting the rock pressure manifestations. However, the efficiency of using such combined bolting systems for complex mining and geological conditions can be realized to the full extent by establishing the patterns of its elements loading in concordance to main geomechanical factors (the depth of development, structure and properties of the coal-bearing massif). As a consequence, the obtaining of the calculated expressions for determining the constructive, geometric, and force rational parameters of support, is a very urgent task. The problem solution is carried out by the method of computational experiment. The modeling technology comprises the synthesis of a general geomechanical system of two components: a model of coal-bearing massif and a scheme of mine working support settings. The scheme is performed in two variants: the existing support setting system in compliance with the technical documentation for the extraction working development, which is called the “basic” and an advanced scheme of support setting based on combined systems, substantiated by studies of the rock massif state, the basic support setting system, and the experience of extraction workings maintenance.

The combination of this type of support setting with various security structures allowed reusing and stabilizing the amount of such mine workings maintenance at the level of more than 14,000 m in the Western Donbas mines.

Despite the difficulties in extraction workings maintenance, their reuse in mines of DTEK Energo LLC, and especially in mines of the Western Donbas, is the main direction of the mining operations technology on very thin coal seams with unstable heaving soil; it is based on the formation of a stable rock arch and preservation of its balance, when in the zone of coal-face work influence (Bondarenko et al., 2017a, 2017b).

Application of a method for increasing of the rock massif stability by means of its hardening in depths around the extraction working using combined bolting systems, which consist of the resin-grouted rope bolts, significantly reduces the material and labor costs. The border rocks of the mine working are involved into counteracting to the rock pressure manifestations.
EFFICIENCY ASSESSMENT OF THE HARDENING WITH THE ROOF BOLTS

Analysis of trends in the use of roof bolts as part of support setting systems in mine workings has shown an expansion in the sphere and volumes of their application in combination with the resin-grouted rope bolts for hardening of the roof rocks as an independent support and in combination with other types of bearing supports (Kovalevska et al., 2016; Majcherczyk et. al., 2014; Grechishkin et al., 2015). Such technical solutions are called “combined bolting systems”, because, as a matter of fact, a combination of two types of roof bolts is used, which differ in their parameters and tasks performed for roof rocks hardening in the extraction working. These technologies of the roof bolt hardening are most actively used in layered soft rocks, such as in the Western Donbas. Therefore, it is advisable to assess the influence of hardening with the roof bolts on the mechanism of the roof rocks’ structure transformation in the extraction working. The classical theory of beams’ and plates’ bending confirms (“Strength of Materials”, 1979) that the maximal horizontal tension stresses (compressive stresses) $\sigma_Z$ occur in the surface areas of the beam (plate), and their value is inversely proportional to the squared thickness of a beam (Figure 1).

$$\sigma_Z > (\sigma_{S})_1$$
$$\sigma_Z > (\sigma_{S})_2$$
$$\sigma_Z < (\sigma_{com})_1$$
$$\sigma_Z < (\sigma_{com})_2$$

Figure 1 – The curves of horizontal stresses $\sigma_Z$ distribution to thickness of rock layers of the roof (a) and a scheme of stability assessment (b) at their separate (——) and joint (−−−−−−−−−−) deformations.

For example, if to combine two rock layers with roof bolts, then the bending resistivity of the combined layers increases four times, whereas their total resistivity at separate bending of each layer increases only twice. In case of the rock layers separate deformation, the tension stresses have an increased value and, at a significant bending of a layer, they exceed the tensile strength of the rock – the discontinuity in the thickness of a layer begins. In case of the joint deformation of rock layers combined with roof bolts, the section modulus increases, and this causes the reduction of stresses maxima $\sigma_Z$, including the tension stresses. The process of tension cracks occurrence and development is reduced and not every alternating bending of the immediate roof, hardened with roof bolts, leads to the separation into blocks of the combined rock layers. Besides, with increased thickness of the load bearing beam (plate), its rigidity increases, which restricts the value of the layers deflection and reduces the probability of this rock volume destruction, as well as falling of the thrust system into separately descending blocks. Consequently, the quantity of interacting elements in the thrust system of roof rocks decreases. And this, according to known rules of multi-hinged load-bearing systems calculation, increases the roof rocks stability – the reaction of their resistivity to the rock pressure increases and the value of descending into mine working cavity decreases.
SETTING OF THE PROBLEM AND ANALYSIS OF COMPUTATIONAL EXPERIMENTS

The performance of computational experiments on the basis of the finite element method (FEM) was carried out with the use of recommendations of the National Mining University as regard to geomechanical processes modeling of the coal-bearing massif displacement in the vicinity of extraction workings (Bondarenko et. al., 2007; 2016). The modeling technology comprises the synthesis of a general geomechanical system (Figure 2) from two components: a model of coal-bearing massif and a scheme of mine working support. The scheme is performed in two variants: the existing support system in compliance with the technical documentation for the reusable belt working development, which is called the “basic” and an advanced scheme of support setting, substantiated by studies of the rock massif state, the basic support setting system and the experience of extraction workings maintenance (Bondarenko et. al., 2014). In general, based on the research results of the basic support setting system, it is established: the maximal threat to its stability is constituted by the frame prop stays, especially they are plastically deformed along the whole length from the side of mined-out space; the cap of the frame flattens out with a bend into mine working cavity in the central part and with the occurrence of the limit state (σ = 260 – 290 MPa) at the SCP profile top, and in the area of a contact with the central prop stay of the hardening support, the plastic deformations develop even more intensively (σ ≥ 290 MPa); among the most loaded roof bolts is the optional one located in the roof of mine working, under the security band, and, in part, the peripheral roof bolts are the most loaded with the appearing of σT steel, whilst the central roof bolts in mine working arch are underloaded; the central prop stays of the hardening support (wooden) throughout the entire volume are in the limiting state, but on a contact with the frame cap – in the superlimiting state. The peculiarity of σ distribution along the length of roof bolts indicates the process of their displacement into the mine working cavity together with the hardened roof rocks, which are stratified more intensively in the border part of mine working and less intensively in the embedded part of the roof bolt length.

It also follows that the active displacements of the roof rocks extend outwards the roof bolts length of 2,4 m: the state of the roof from side of the longwall (especially above the security structure) requires not only a thicker reinforcement mesh, but also its deeper distribution. One of the technological solutions to this problem is the setting of rope bolts, which can simultaneously serve as a deeper hardening of the roof rocks, limiting its stratification. Thus, if to set two rope bolts (in increments of L = 1,6 – 2,0 m, that is, every two frames along the mine working length) symmetrically in a cross-section to 0,8 to 1.0 m from the vertical axis of mine working with an angle of 70 – 80º to the horizontal, they harden the roof volume to a width of 7,5 – 8,0 m. This is sufficient to increase the stability of rock consoles and dimensions restriction of the limit state zone of rocks. Analysis of stress intensity curves (Figure 3) of the massif surrounding the mine working with the use of a combined bolting system instead of the basic one has shown a variety of changes in the state of adjacent rocks. In the roof of the extraction drift, an unloading zone σ is formed, which differs from that for the basic scheme of support setting by its shape and dimensions. First, unloading almost does not affect the immediate roof rocks: from the side of mined-out space, a local area with a value σ is formed, which corresponds to the virgin massif state; in rocks above the central part of the arch, an area is created with a concentration σ of 1,6 – 2,0 level from the initial state of the virgin massif. Secondly, the distribution pattern σ in the first layer of the main roof (heavy siltstone) indicates the possibility of a partially independent deflection of the main and immediate roof rocks relative to each other, caused by restricted flexibility of the rope due to its tensile elongation. It is this possibility (even if limited) of horizontal displacements relative to each other in the immediate and main layers of the roof, which induces the occurrence of a local unloading area in them with dimensions in the section plane of up to 0,7 × 1,0 m. The level of unloading is 0,4 – 0,8 from the initial state value of the virgin massif and points to preservation of a part of the horizontal thrust within this local area, above which the siltstone state changes to initial state, and then at half of the rock layer thickness of the main roof, the
concentration $\sigma$ of 1.2 – 2.0 level comes into play. This concentration characterizes the effect of thrust forces in the case when the main roof lower layer resists the deflection, and the absolute values of $\sigma$ are by 1.75 – 2.91 times lower than the compressive resistance of siltstone. In adjacent side rocks from the side of virgin massif, another feature is observed in the distribution of $\sigma$: they are lowered in comparison with the basic variant of support setting. Thus, the concentration $\sigma$ is 1.6 – 2.0 to width into massif up to 2.1 m, which is many times less than the compressive resistance of both as the immediate roof rocks, so the coal seam rocks; in soft rocks of the immediate bottom this concentration causes their weakening and provokes the intensity of the swelling process. If comparing the side bearing pressure when using the combined bolting system, it is as much as 2 – 3 times lower, relative to the basic variant of the support setting.

In such a way, the analysis of the coal-bearing massif state around the belt working proves conclusively that the application of combined bolting systems enables the more efficient use of the load-bearing capacity of the roof rocks by maintaining the thrust forces between the rock blocks and rock armored plate formation. And this, in turn, by analogy with the “chain reaction”, reduces the SSS manifestations’ intensity in sides and bottom of the extraction workings. At the same time, the elements of the support setting system are loaded with 70 – 90% of their load-bearing capacity without the formation of the plastic state zones (Figure 4). It is then necessary to determine the quantity of roof bolts, sufficient for the formation of a rock armored plate in the roof, protecting reliably the extraction working from excessive rock pressure manifestations. It is therefore important to outline the range of conditions for the effective application of combined bolting systems based on established patterns in relation to the roof bolts loading degree and geomechanical factors characterizing the conditions of the extraction working maintenance. Then, based on the developed general algorithm for search of rational parameters of the combined bolting systems setting and within the general scheme of extraction working maintenance, a range of geomechanical factors has been identified and substantiated. These factors have the greatest effect on the degree of the roof bolts loading: depth $H$ of mine working location, average calculated compressive resistance $R$ of adjacent coal-bearing rocks, the ratio of calculated compressive resistance of the coal seam immediate roof rocks to its thickness $R_{1r}/m_{1r}$. To estimate the degree of the roof bolts loading, the appropriate criteria are substantiated: the relative length of the plastic state of the resin-grouted bolts load-bearing element $\Delta$ in the central part of the mine working arch and rope bolts $\Delta_{f,m,\text{rope}}$ in the roof from the side of mined-out space and, respectively, of virgin massif. The general pattern (Figure 5) for all the roof bolts within the combined bolting system is in an increase of the relative length of the plastic state segments with the growth of the parameter $H$; all the patterns are close to the linear functions $\Delta(H)$ and $\Delta_{f,m,\text{rope}}(H)$.

![Figure 4](image1.png)  
**Figure 4 – The curve of stresses $\sigma$ intensity in the support when using a combined bolting system**

![Figure 5](image2.png)  
**Figure 5 – The patterns of influence of the mine working location depth $H$ on the relative length $\Delta$ and $\Delta_{f,m,\text{rope}}$**
of the plastic state segments of the roof bolts

There is a different degree of loading of resin-grouted and rope bolts.

The second geomechanical factor significantly affecting the degree of roof bolts loading within the combined bolting system is the parameter $R$ (Figure 6). The general trend for all the roof bolts is in an increase of $\Delta$, $\Delta_{l, m}^{\text{rope}}$ with a decrease in the parameter $R$, but the dependences $\Delta(R)$ and $\Delta_{l, m}^{\text{rope}}(R)$ differ significantly from each other. The patterns of the parameter $R_{1}^i/m_{1}^i$ influence on the relative length $\Delta$ and $\Delta_{l, m}^{\text{rope}}$ are shown in Figure 7.

The revealed patterns are the basis for the development of methods for calculating the rational parameters of the bolting system. And this is realized by two conditions: on the one hand, the roof bolts, together with the hardened roof rocks, should maximally unload the frame support to effectively restrict the cross-section loss in the mine working; on the other hand, the roof bolts in the roof should work with maximum resistivity to rock pressure, that is, they should be set in a sufficient, but minimal quantity.

When determining the relation of the coordinates $x_i$ of the resin-grouted bolts setting with geomechanical factors, the patterns of the roof bolts loading degree variation (see Figure 5 – 7) are used. The resin-grouted bolts are located along the mine working arch contour. And it follows from these patterns that the force potential of the resin-grouted bolts in the roof is realized only with an intensive manifestation of vertical rock pressure: $H \geq 400 – 450$ m, $R \leq 10 – 15$ MPa and $R_{1}^i/m_{1}^i \leq 15 – 20$ MPa/m. Under these conditions, the resin-grouted bolts actively resist the processes of stratification and lowering of the mine working roof rocks that implies an increased density of their setting ($7 – 9$ roof bolts). In more favorable mining and geological conditions, the quantity of roof bolts in the roof is significantly reduced (up to 3 to 5 roof bolts); since they remain underloaded and their minimal required quantity in the roof is determined by the search for the $x_i$ coordinate of each roof bolt location. With a significant underloaded volume of any roof bolt, they are excluded from consideration, while retaining their load results for more loaded roof bolts. At the same time the load, taken up by underloaded roof bolts is considered when increasing the stresses intensity in the loaded roof bolts due to the reserve of resistance, caused by the point and stage of any steel hardening of the roof bolt’s ‘reinforcement’. With the use of the stated approach, the graphs of dependences (Figure 8) were obtained with the coordinates $x_i$ of the resin-grouted bolts setting on the parameters $H/R$ and $R_{1}^i/m_{1}^i$.

By way of preliminary calculations and experiment it was found out that placing more than six roof bolts in the central part of the roof is not rational due to their low loading. Then, when a symmetrical scheme for the roof bolts is setting, the change in the three coordinates of $x_i (i = 1, 2, 3)$ was analyzed depending on the mining and geological conditions. An inverse relation was established using the coordinates of $x_i$ with the value of $H/R$ ratio, which is steadily repeated for all the calculation variants, regardless of the roof bolts quantity in the mine working arch. The roof bolts are excluded, for which the $x_i$ coordinate reaches or exceeds the coordinate of the frame yielding joist location.

In a different way, the coordinates $x_i$ are affected by the ratio $R_{1}^i/m_{1}^i$: with the roof rock hardness increase, the load on the roof bolts is reduced and it is advisable to decrease their quantity, and this corresponds to an increase in the distance between the roof bolts (the displacement of the coordinates of their setting from joist to spring). The revealed patterns make it possible to obtain a set of regression equations from the calculation of the values $x_1$, $x_2$, $x_3$:

$$x_1 = -5.8 \times 10^{-2} R_{1}^i/m_{1}^i \ln(0.015 H/R - 0.52), \text{m};$$

$$x_2 = 10^{-2} R_{1}^i/m_{1}^i [4.8 - 10.9 \ln(0.015 H/R - 0.52)], \text{m};$$

$$x_3 = 10^{-2} R_{1}^i/m_{1}^i [12.6 - 17.2 \ln(0.015 H/R - 0.52)], \text{m}.$$

In the adjusted values, the index “1” for $x$ stands for the coordinate $x_1$ of the roof bolt setting with the minimal distance from the vertical axis of the mine working, and the index “3” – with the maximal distance. If the value $x_3$
exceeds the horizontal coordinate of the yielding joist location in the frame support with corresponding size of the mine working cross-section, then the end roof bolt is excluded from the scheme of the roof bolts setting in the arch. In a similar way, the expediency of the roof bolts setting in the arch with coordinates $x_2$ and $x_1$ is considered. Then, in some area of geomechanical parameters ratio, there is no need to set the resin-grouted bolts along the arch contour in place between the yielding joists of the frame. The main point of this statement is that the roof, with increased stability, is not able to load the resin-grouted bolts up to the level of their active work on resistivity to vertical rock pressure. There is an effect of a relatively restricted length of the resin-grouted bolts: at this length, the difference in the roof displacement in the area of the roof bolt shank and in place of its joist does not create tensile forces in the roof bolt’s ‘reinforcement’, which are comparable to the yield point of steels. At the same time, with an increased length of the roof bolts (for example, rope bolts), the difference in these displacements increases that induces their full loading and effective resistivity to the roof lowering even under conditions of its relative stability. Thus, there is an area of mining and geological conditions, when a low efficiency is observed of the roof rocks hardening with resin-grouted bolts within the central part of the mine working arch. To determine the boundaries of this area, the following ratios have been obtained of geomechanical parameters for the most commonly used types of cross-sections $S$ of extraction workings with TSYS support:

$S = 11,0(11,7) \text{ m}^2$ \quad $H/R \leq 34,7 + 66,7 \exp(-17,2m_1^i/R_1^i)$, m/MPa;

$S = 15,0 \text{ m}^2$ \quad $H/R \leq 34,7 + 66,7 \exp(-19,8m_1^i/R_1^i)$, m/MPa;

$S = 17,7 \text{ m}^2$ \quad $H/R \leq 34,7 + 66,7 \exp(-22,4m_1^i/R_1^i)$, m/MPa.

When a value $H/R$ is less than calculated by the above formulas, it is not recommended to set the resin-grouted bolts in the central part of the arch; otherwise, their quantity is determined by the $x_i$ calculation formulas. To promptly assess the practicability of the resin-grouted bolts arrangement in the central part of the mine working arch, the graphs are constructed (Figure 9).

**Figure 8** – The relation of the coordinates $x_i$ ($i = 1, 2, 3$) and parameter $H/R$: $R_1^i/m_1^i = 4,8$ MPa/m; $R_1^i/m_1^i = 10,2$ MPa/m; $R_1^i/m_1^i = 15,6$ MPa/m

**Figure 9** – The area of appropriate resin-grouted bolts arrangement

RESULTS OF THE BELT ENTRY UNDERGROUND TRIALS AT AN EXPERIMENTAL SITE OF THE COMBINED BOLTING SYSTEM SETTING

The belt entry was reused as ventilation during the coal-face work at an adjacent extraction site. The scheme of the entry support setting is shown in Figure 10a. In the lower part of its length the experimental site is located with the recommended scheme of support setting (Figure 10b).
It has been established that the combination of the resin-grouted rope bolts creates hardening not only of the immediate, but also of the main roof, binding the rock layers into a single rock armored construction with high thickness (not less than 4 – 5 m), capable to take up and resist increased rock pressure and, thus, to protect the frame support against excessive loads. The formed rock armored plate extends the area of its supports in the mine working sides, and its high load-carrying capacity and rigidity allows not only to reduce the load on the frame, but also to reduce the rock pressure manifestations. And this restricts the horizontal displacements of the border rocks in mine working sides and the intensity of the swelling process development in the bottom rocks. The visual observations (Figure 11) and measurements showed that the shape of the TSYS-15,0 support frame and its elements (cap and prop stays) were almost not changed outside the coal-face work influence zone in comparison with the ones originally set, there were no plastic deformations along the whole contour of the frame; the frame yielding joists did not act and the overlap value of the cap and the frame were also in the primary position about of 350 – 420 mm. This indicates what the restricted load of the frame till its pliable mode of operation was. These observations indirectly indicate a high efficiency of the rock armored plate operation in the roof of the entry: its load-bearing capacity makes it possible to unload the frame support from excessive rock pressure by taking up of the greater amount of its part.

Figure 10 – Schemes for the belt entry support setting with the use:
a) roof bolt support (basic) and b) combined bolting (recommended)

Figure 11 – Fragment of the belt entry state at an experimental site
CONCLUSIONS

In general, summarizing the results of mine observations, it is necessary to note the satisfactory state of the belt entry after the first longwall drivage. Its state allows reusing it as ventilation when mining the adjacent extraction site, if to carry out the moderate repair and recovery works in respect of the frame support, side and roof. A positive result has been achieved through the use of a combined bolting system in the mine working roof.

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APPLICATION OF ROOF-BOLTING SUPPORT IN THE UKRAINIAN MINES

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Maintaining of the mine roadways in stable operational state during the whole period of their exploitation is the most critical problems of the coal-mining industry. Safety of miners, provision of working places with air and materials, productivity of the shaft-sinking and mining machines, load on the longwalls, and, in whole, the total operation of the mine – all of these issues depend on the safety roadway supporting.

Action of the rock pressure forces leads to destruction of the edge rock mass. Commonly used frame supports do not provide sufficient resistance, which could be enough for essential reduce of deformation processes in the rocks around the roadways, Figure 1. The deformation processes in the rock mass can be controlled with the help of the bolt supports, which helps to improve strength properties and bearing capacity of the rocks.

From the late ninetieth, Ukrainian miners began to use bolts with polymer fixing. Scientific and technical aspects of these jobs were supervised by researchers of our Institute of Geotechnical Mechanics of National Academy of Science of Ukraine.

At that time, the roof bolts were installed in our mines in accordance with such normative documents as КД 501 and КД 502 which were based on the then world practice. These documents specified conditions for using the roof bolts and technical requirements for the bolts and their elements, equipment and work technique. Basically, the roof support was used in the roadways with rectangular cross-section, Figure 2, and they were installed perpendicularly to the bedding, Figure 3.
However, in the process of accumulation of practical experience on usage of the roof bolting in the Ukrainian mines, where new equipment was implemented and, therefore, loads on the longwall increased and geological conditions were worsened it became clear that bearing capacity of the most of the known schemes and designs of the roof bolting did not give a possibility to build roadways with the working life more than 4 years. As well, the existing techniques of the roof bolting made impossible to reuse the gate roads though it was very important in view of improving the coal produce costs. 

The reason was that geological and technical conditions in the most of Ukrainian mines were significantly more complicated than in the mines of other leading countries which produced coal. Thus, a simple transition of the accumulated foreign experience failed to provide any effective positive results. The situation demanded to study more deeply mechanism of the roof-bolt operation and to specify the space-time laws of geomechanical processes occurred in the system “edge rock – roof bolt”.

In order to study the mechanism of the roof-bolt operation, we designed a mathematic elasto-plastic model for calculating stress state of the rocks around the roadway with the bolts. This model took into account parameters of the roof bolts and roadway, strength and deformation properties of the rocks, as well as time and site when and where the roof bolt start working. In order to solve this problem, we applied a finite element method. The roof bolt was simulated by the rod finite elements, washer and grab – by the triangle (prismatic) finite elements, and polymer fixture – by special contact elements. Each of the elements of the roof bolting featured certain physical and mechanical properties.

Level and character of the rock stress state variations nearby with the roadway was estimated with the help of the parameter \( Q \) – reduced difference between the maximum and minimum components of the key stresses. This parameter characterizes difference between the stress-filed components and possible occurrence of the rock failure.

In view of studying interaction between the roof bolts and the rocks, we made a series of calculations. The first step was simulation of the stress state in rocks around the unsupported roadway. Result of
the simulation is the basis for the further analysis. We see that in the edge zone, the unsupported rocks are deformed and their failure is developed, Figure 4.

![Diagram showing roadway support options](image)

- a) roadway with no support; b) roadway with 1 roof bolt; c) Roadway with 5 roof bolts.

**Figure 4 – Formation of the rock-bolt support and canopy in the roadway roof, distribution of values of parameter Q**

When one bolt is installed, a certain volume of the roof rocks connected with the bolt is kept from displacing. With the course of time, rocks in the zone around the bolt do not unloaded and are still in the compressed state. At a certain distance from the bolt, deformation processes in the rock mass are developed in the same manner as in the roadway with no bolts. It is possible to specify this zone around the bolt as a support due to its function. This zone keeps the rocks in the edge area from displacing into the roadway, Figure 4.

The roof bolting is a space system of the roof bolts fixed in the bore holes. The key target of the bolt system installation is to obtain such bolt location which provides maximally effective interaction between the supports created by the bolts. How to improve efficiency of interaction between the rock-bolt supports? To increase density of the bolt installation? But when parallel bolts are installed very close to each other it can cause formation of cracks between them. Under certain geological conditions, when the bolts are installed vertically, a rock mass can be divided into blocks. And we know cases when such blocks have failed into the roadways, Figure 5.
We have analyzed a great number of mine, lab and computing experiments and, as a result, we found a solution for the problem of how to improve efficiency of interaction between the rock-bolt supports. The technology of the roof bolting was further developed, and range of its application was essentially expanded thanks to the following principles of the support formation:

- firstly, implementation of the system of bolts, which are set inclined along the roadway axis, and which improve interaction between the rows of the bolts and give a possibility to form the rock-bolt structures;
- secondly, division of the structure of rock-bolt canopy into blocks (the bolts are installed in groups). It makes possible to maintain stability of the roadway as a whole system thanks to the safety displacement of the blocks relatively to each other;
- thirdly, arrangement, from time to time, of the areas with enhanced structural stiffness along the whole length of the roadway;
- fourthly, more rigid limitation for the rock displacement into the roadway in order to keep the rocks in their natural monolithic state and to increase stability of the rock outcropping.

Researchers and specialists of our Institute have designed and tested the reinforced schemes for the roof bolt setting (a structure of the rock-bolt canopy), Figure 6, in which the rock-bolt blocks are formed by way of spatial location of the steel-polymer bolts. In each of such blocks, one part of the bolts are set inclined towards the coal-face, and the rest of the bolts are set inclined towards the back, i.e. towards the roadway mouth. Such location helps essentially improving interaction between the bolts if to compare with traditional schemes with one-direction reinforcement of the edge rock mass.

Assignment of the rock-bolt blocks:
1 – is a load-bearing part of the canopy, which counteract to the deformations of the roof rocks and their displacement into the roadway;

2 – is a backing-up part of the canopy, which takes the load from the load-bearing element of the canopy and transmit it to the support. This part is also needed for controlling the structure life period depending on the assignment of the roadway;

3 – are supports of the canopy, which transmit the load from the canopy to the base;

4 – is a base of the structure, which facilitates to increase stability of the roadway wall and decrease the soil heaving;

5 – is an element of the rock-bolt structure which we call a “bridge”. It is marked by yellow on the scheme. It is a sector of the roadway with the higher stiffness thanks to the additionally set bolts. Length of the sector includes several rows of the bolts. The “bridges” prevent the edge rock mass against falling due to the released strain energy, which is accumulated with each elongation of the roadway. Staggering of such sectors makes possible to control the roadway stability, restore the needed safety factor and reduce expenses on the roadway exploitation.

The supports and schemes of the bolt setting are calculated basing on the requirement to exclude the rock falling around the roadway. Bearing capability of the rock-bolt supports is determined depending on the geological conditions of the mining operations, characteristics of the bolt strength and the bolt fixing in the bore hole. An optimal quantity of the bolts and scheme of the bolt setting are calculated in terms of the support resistance required for blocking displacement of the roadway contour.

One of the key requirements of the technology is to set the roof bolts into the hosting rock mass, which is not unloaded from the rock pressure. In this case, the support, just after having been set, immediately start working on resisting the forces of the rock pressure. The edge rock mass maximally saves its natural monolithic state. If not, the bolts will just “sew together” the stratificated roof rocks.

As well, we developed a new method for calculating parameters of the stress-strain state of the rocks and structure of the rock-bolt bridge for technological cycles of setting the rows of the roof bolts. This method takes into account the rock unloading during the cycle and a moment of time when the row of the roof bolts start to operate. The stress fields changing during technological cycle of the roof bolt setting are shown in Figure 7.

![Figure 7](image_url)

a) before the face driving; b) on the next iteration after the face driving; c) in 30 iterations.

Figure 7 – Parameter $Q$ values changing during technological cycle of the roof bolt setting
The roof bolt is set at distance 0.3 meter from the face. As the face is further driven the bolts are loaded and prevent the rock from displacing into the inside of the roadway. When the bolts are set at a longer distance from the face the edge rock mass can easily displace during certain period of time and become unloaded. Thereby, the next bolts will be set into the disturbed rocks and, in case of the weak rocks with low tensile strength, – into the broken rocks.

Special difficulties present maintenance of the roadways for further reusing it while mining an adjacent longwall. Application of bearing-bolt supports showed good results for these complicated conditions as well.

For example, the reinforced structure of the roof bolting with the strong protective bridges was used in the mother entry 585 of the Yubileynaya Mine, Figure 8. The roadway had an arch cross section. It was kept in operation when the first and the second longwalls had been mined out and later. Results of monitoring showed satisfactory stability of the roadway at all of the stages. The roof was not broken, it just sagged as a single block with turning, first, towards one side and then – towards another. Periodical reinforcement of the bridges helped to restore stability of the roadway and not to accumulate negative affect of the rock pressure with each elongation of the roadway, Figure 9.

Figure 8 – Reinforced structure of the roof bolting in the mother entry 585 of the Yubileynaya Mine
One more advantage of the bearing-bolt supporting must be mentioned which is also best for driving roadways through the gas-bearing coal seams and rocks. Results of solving the equations of elastoplastic deformations and gas filtration were confirmed by data from the mines and have showed the following:

In the roadway where roof is fixed by technology of the bearing-bolt supporting, an impermeable rock-bolt canopy is formed. This canopy prevents escaping of methane from the gas sources in the roof into atmosphere of the roadway. When density of the bolt setting is increased up to 1.1 bolt per meter, average speed of methane filtration and consumption decreases by 75 percent, Figure 10.

![Figure 9 – Mother entry 585 of the Yubileynaya Mine](image)
a) 10 meters to face end; b) 60 meters after the face.

![Figure 10 – Change of speed of methane filtration and flow rate depending on quantity of the bolt setting in the roadway roof](chart)

Application of roof bolting for the blocks in the roadway driven through the gas-bearing coal seam keeps the seam and adjoining rocks in stable state, and blocks processes of crack formation and coal pressing-out. Each of the bolts, set into the side of the roadway, decreases permeability by 20-30 percent.
Such effect essentially decreases volume of methane escaping from the coal seam during mining and exploitation of the roadway, Figure 11.

The bolts which are set in the bottom of the roadway sides prevent the sides from falling and decrease zone with inelastic deformation in the roadway floor and, consequently, diminish soil heaving and filtration permeability.

Therefore, the roof bolting can be considered as a technological method for decreasing methane content in the roadways.

Basing on the research findings, we have developed and approved a normative document, which regulates parameters for the technology of the bearing-bolt supporting. We also have developed an album with typical structures, which regulates choice of the schemes for the roof bolt setting in typical mining, geological and technological conditions of the roadway drivage.

The technology of the reinforced and powerful bearing-bolt support setting designed by our Institute of Geotechnical Mechanics was successfully realized in practice in 52 Ukrainian mines while driving more than 700 preparatory roadways and permanent workings. This approach helped to achieve stable state of the roadways under the complicated mining and geological conditions and obtain good economic effect thanks to the cut expenses spent to the roadway maintenance and repair.

However, the most weighty contribution of the roof bolting technology into cutting costs of coal production is, to our mind, higher speed and essentially better safety of the winning operations. Thanks to the stable, almost unstressed state of the face end, necessity in setting additional supports falls down, and time period needed for the end operations during the face mining becomes essentially shorter. Besides, our technology allows to use effectively any powerful mining machines and significantly speed up rate of the face mining.
STOWING OPERATIONS IN KAZAKHSTAN: STATUS AND PROSPECTS

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PURPOSE OF STOWING

Implementation of the program of forced innovative industrial development of the Republic of Kazakhstan requires, first of all, modernization of the main industrial sectors, including the mining and metallurgical complex. The fulfillment of this task in the mining industry requires the improvement of the technology of mining of solid minerals by underground method, which will allow them to increase their production with a sharp decrease in losses. Providing high-performance, safe and economical extraction of minerals is the most difficult task, especially since the depth of field development reaches a significant level.

The wide use of the stowing of the developed space for many of the country's ore and coal deposits can help solve these problems. The use of the stowing allows the use of highly effective development systems with minimal losses of minerals in the Earth with minimal dilution by empty rocks, increases the safety of mining operations while maintaining the surface from subsidence, simplifies the management of rock pressure, etc. [1].

Modern mines are characterized by high intensity of development of deposits, rapid movement of mining operations to depth and more complex conditions. The systems of development with an open developed space will have to be replaced by systems with a stowing or with a collapse because of the large sizes of natural pillars and intensively increasing rock pressure.

As a result of the use of a hardening stowing, the losses and dilution of ore during extraction are reduced by three to four times, which ensures a high economic effect and pays off the cost of filling operations.

Using of a stowing allows to improve the traditional methods of opening, schemes for preparing floors (panels), create the ascending order of development of places. This circumstance makes it possible to place trunks of shafts in the immediate vicinity of the ore-bearing strata, reducing the length of the cuttings to the limit and passing them immediately to the entire explored depth of the deposit, and start working off from the deepest horizon and lead along the same upward, i.e., work over the developed space, previously laid by a hardening stowing.

The new technology opens up ample opportunities for burying many waste products in mines as part of hardening or free-flowing stowing, which, combined with preserving the extra work area, ensures greater eco-economy of land resources, contributes significantly to the protection of the natural environment. The foregoing testifies to the great potential possibilities of technology for the development of deposits with stowing of worked-out space.

On the basis of normative documents [2], the feasibility, necessity and method of the stowing of the worked area is justified by economic calculation, taking into account geological and mining conditions of the deposit, the value of the mineral, the damage (using the development systems with collapse) from the underuse of land, the cost of reclamation and others.

The technology of underground mining of ores with stowing of mined space is one of the main directions of innovative technical development of mining.

A stowing of the worked out space allows to solve a number of different tasks, providing favorable and safe conditions for conducting mining operations. In this case, we can distinguish two main goals that can be achieved by applying a stowing:

- management of mining pressure in the clearing space, which will allow using highly effective development systems, reduce mineral losses in the bowels and dilution, improve the safety of mining operations, improve the ventilation of the faces;
- the elimination of voids, which will enable to develop fire-dangerous deposits, to conduct joint development by open and underground methods, as well as to make selective excavation of minerals.

MATERIALS AND THEIR PREPARATION

Experience of the development of minerals by systems with the stowing of worked-out space in the Republic of Kazakhstan shows that the quality of the erected artificial massif
depends on the properties of the applied materials to a large extent. The choice of materials is
determined by the requirements for the storage array, the way of transportation and placement of
the stowing in the developed space.

The filling materials are subject to technological requirements and specific requirements
related to the method of transportation and laying in the worked-out space. The main
disadvantage of the previously formulated requirements for backing materials is that they
proceed from the characteristics of the material itself, rather than an artificial massif that will be
obtained as a result of filling operations. In fact, the filling material in the process of its
preparation, transportation and erection of an artificial massif undergoes significant changes.
Water-resistant aggregates of the material decompose, some chemical reactions and changing of
the composition of the particles are possible. As a result of friction, particle collisions between
each other and blows against the walls of pipelines, dropping into the chambers from a great
height, there is an additional crushing of the material, changing the shape of individual pieces.
Therefore, the material from which the backfilling array is formed differs from the original one
primarily by the granulometric composition and shape of the particles. This determines the
characteristics of the requirements for the filling material.

There are a number of requirements for astringent materials. The binder should have the
necessary activity, which is understood as the ultimate strength of the samples of the standard
mixture at the age of 28 days. The cementitious material must be resistant to aggressive media,
which allows it to be used in the development of deposits where the mine waters differ sharply in
their characteristics from neutral ones. The cost of the binder should not be the predominant cost
item in the cost of the stowing.

Previous studies have established that placeholders for a stowing must meet a number of
requirements:
- the stuffing material must form an array when filling out the worked out space, which
  makes it possible to apply the development system most rational for these mining and technical
  conditions;
- the material must have the ability to form in the worked out space a solid, dense, stable
  massif with low air permeability;
- under the influence of its own weight and rock pressure, it must give a minimum
  shrinkage;
- the packing material should not have a tendency to spontaneous combustion (the
  allowable content of combustible components - coal, oil shale, etc., not more than 20%, sulfur
  not more than 5 - 8%);
- the filling material must be sufficiently strong, as during transport it is subjected to
  additional crushing;
- the filling material should be of low cost, and its reserves should ensure the need of the
  mining enterprise during the development period of the deposit.

Water is an indispensable and necessary part of the process of preparation and
transportation of the backing mixture for hydraulic and hardening stowing, as well as the process
of hydration and its quality, the properties of the backing mixture and the future artificial mass
depend to a large extent.

Water can be recognized as suitable if the samples of the stowing made at it at the age of
28 and 90 days have a tensile strength not less than those prepared on clean drinking water at the
same time of hardening.

The introduction of additives into the composition of the backing mix is one of the most
technologically, flexible, accessible and universal ways of improving all the properties of filling
mixes and giving them new, uncharacteristic properties, as well as reducing labor costs, cement
consumption, saving heat and electricity, improving technology and so forth.

Improving of the rheological properties of the stowing when using a plasticizer additive as
a component of the filling mixture was confirmed by the authors of the article at the Ridder-
Sokolny mine of Kazzinc LLP. In particular, in the laboratory conditions, the increase in the
Preparation of layer mixtures, their composition and characteristics

Depending on the conditions of development of deposits, different types of filling facilities are used. They can be stationary, designed for high productivity, mobile or temporary for laying small in volume, but far from significant areas of the worked out space.

At the mining enterprises of Kazakhstan, the technology of production of backing mixes by mixing or mill method on the basis of cement, cement-slag or cement-ash binder with use as a filler of a mixture of crushed rock mass and wastes of mining and metallurgical production has found wide application.

As an example, the figure shows the technological scheme of the laying complex of Maleevsky mine.

Crushing of crushed rock and slag-slag to fineness of 50% of the class content minus 74 μm, as well as disintegration of small aggregate slurry aggregates, intensive mixing of all components of the stowing and preparation of a homogeneous high-density mixture is carried out in ball mills. The use of ultrathin grinding in the technological process makes it possible to substantially reduce the binder consumption on the filling complexes while maintaining the strength characteristics of the formed massifs, which will positively affect the economic performance of the mining enterprise.

![Technological scheme of the laying complex of Maleevsky mine](image)

On the projected laying complex of the Novo-Leninogorsk mine, Kazzinc LLP, it is envisaged to crush the pulp of the ground grind slag to a tonne of 80% of the class-minus 20 μm content (with a specific surface of 5000 cm²/g) in an ultra-fine grinding mill [6]. At the grinding fineness of up to 70% of the minus 74 μm class (grinding only in the mill MSHTS) in order to achieve the normative strength of the stowing at the level of $R_{28} \sim 2.0 \text{ MPa}$ and $R_{90} \sim 3.5 \text{ MPa}$, the following composition of the stowing is accepted, kg/m³: Portland cement M400 - 50; ground grinding slag - 250; filler - 1200, including crushed rock - 600; water - 450. At the same time, with ultrathin grinding of the slag to 80% of the class minus 20 μm (with grinding in an ultrathin grinding mill), the following composition of the mixture, kg/m³, can be used when achieving the same strength characteristics of the stowing: Portland cement M400 - 10; finely ground blast furnace slag - 190; filler - 1375, including crushed rock - 600; water - 420. Thus,
the ultra-fine grinding of the blast-furnace slag at the projected Novo-Leninogorsk mine will reduce the specific consumption of Portland cement M400 by 40 kg / m³ and the slag slag at 60 kg / m³ while maintaining the strength characteristics of the formed stocking masses.

To date, the technology of ultrathin grinding of blast-furnace slag has been successfully introduced in the filling complexes of the Tishinsky and Maleevsky mines of Kazzinc LLP.

Compositions of hardening filling mixes to fill the worked out space are selected taking into account the requirements for the standard strength of the artificial massif and depending on the method of transporting the mixtures to the mine. At the same time, much attention is paid to the utilization of waste from mining, processing and metallurgical production in the waste deposit [7].

TRANSPORTATION OF PLANNING MIXTURES

Analysis of the practice of filling systems showed that the most common way to feed the mine into the mine is gravity and gravity-pneumatic transport of filling mixes through transport pipelines. The transportation of packing mixtures by gravity is limited by the static pressure in the vertical stack [8].

For gravity-pneumatic transportation of filling mixes, a mixture of compressed air and filling mixture moves for a longer distance through the pipeline than with gravity transport. However, the transportation of the filling mixture with compressed air causes its increased stratification. This is due to the impact on the mixture of a high-speed compressed air flow and is due to the heterogeneity of the mixture as it moves through the pipeline.

A special feature of the vibrating-transport method for transporting viscoplastic mixtures is the vibrational effect on the pipe becoming of an alternating force, with vibration acting as a means of creating conditions for lowering the resistance to transportation, and moving the mixture by the hydraulic head of its vertical column.

Vibration of the pipeline greatly improves the transportability of the filling mixes. The frequency and amplitude of the pulses and the angle of their transmission exert a significant influence on the effectiveness of the thixotropic dilution of the wall layer of the filling mixtures. The most favorable conditions are a vibration transmission angle of 30°, at a frequency of 50 Hz and an amplitude of 0.2-0.4 mm. In this case, the transportability index reaches a minimum value, which provides an increase in the range of transportation of the packing mixture in the self-winding regime by 2.5 - 3.0 times.

Forced transport of backfilling mixtures is also possible with concrete pumps and pneumatic blowers [9, 10]. Thus, at the Maleevsky mine to fill the cavities of the spent chambers in the floor of 7-11 horizons of the Maleevsky ore zone (above the level of the filling horizon up to 200 m) on a horizontal stab through the crosscut shl. Maleevskaja the pump unit HSP 25.100 HP Duplex of firm PUTZMEISTER AG is installed which carries out a pressure delivery of a backfilling mix on overlying horizons. At Artemyevskoye mine, Vostoksvetmet LLP, it is planned to use pressure concrete pumps for transporting the packing mixture from surface packing complexes to the flanks of ore deposits of the second stage of the deposit.

For the Suzdal mine JSC FIK "Alel" with the participation of the authors developed a technology for preparing a stowing mixers on the surface complex and transportation of the packing mixture to the places of its laying in the developed space with self-propelled mixers. Laying of the mixture into the worked space is provided by concrete pavers with the size of the inert aggregate up to 20 mm, feeding the mixture into the worked space up to a distance of 80 m.

ARTIFICIAL STOWING (STRUCTURES, OBSERVATION)

The main purpose of the stowing when controlling the rock pressure is to maintain the rocks containing the cleaning workings and when controlling the displacement, to limit the movement of the enclosing rocks to values that ensure the displacement of the rocks in the region of the location of the work piece in acceptable limits.

The main characteristic of the stowing when controlling the rock pressure is its strength in the outcrop on the mine workout contour; when controlling the shift of the overlying rock strata, the stiffness of the filling massif.
In the development of ore deposits with a stowing in the Republic of Kazakhstan, the most widespread are mechanized types of laying with pipeline transportation - hydraulic and hardening. In recent times, especially in the mining industry, the concrete application has been widely used. At the same time as fillers tend to use the tailings of concentrating factories and empty rocks from tunneling works, and as components of a complex astringent - ground granulated slags.

To date, schemes have been developed for reclamation of the rock from excavation work to the laying without issuing an empty rock to the surface, ensuring highly efficient and environmentally friendly operation of the mining enterprise, in demand by the real sector of the economy, based on domestic scientific and technical developments. A number of patents of the Republic of Kazakhstan have been developed for the developed schemes for the disposal of waste rock in the laying and the scheme for the extraction of ore reserves on contact with the filling massif. Developed by the authors of the article, the rock utilization schemes were implemented at the Maleevsky mine of Kazzinc LLP [11, 12].

**PROSPECTS OF DEVELOPMENT OF TECHNOLOGY AND TECHNIQUE OF STOWING WORKS**

The excavation of ore reserves by development systems with a hardening stowing is currently carried out in Kazakhstan or planned to be carried out at a number of mining enterprises, including the Maleevsky mine of the Zyryanovsky plant, the Tishinsky Ridder-Sokolny and the Shubinsky mines, and also in the future at the Novo-Leninogorsk mine of the Ridder PLANT Kazzinc LLP; Orlovsky and Artemyevsky mines, with the excavation of security ladders at the Annensky mine of Kazakhmys Corporation LLP, it is planned to use development systems with the laying of worked out space for the extraction of ore reserves in the safety pillars of the Suzdal deposit of the financial and investment corporation Alel, while working out the Pervomaiskoye mine mine, 10th anniversary of the independence of Kazakhstan of the Donskoy plant, TNK Kazkhrom JSC, while processing the Ushkatyn deposit of Zhayremsky plant, when gold and gold-copper-zinc reserves are being seized Maykain ore deposits in the underground mine Maykain-JSC "Maykain-Gold".

It seems very likely the use of development systems with hardening tabs during the development of the Sekisovskoye deposit of the "Sekisovskoye" mining enterprise and the Tellur deposit of "Akmola Gold" LLP of the company "Hambledon mining company Limited", as well as the underground mining of gold-bearing ores at the Bakyrchik deposit Bakyrchik mining enterprise ".

On the underground mines Corporation Kazakhmys LLP and Kazzinc LLP also found application of hydraulic and dry rock laying of the worked out space.

Technical and economic studies show that the use of a stowing of the worked out space in certain mining and geological conditions is not only technically expedient, but also economically justified. A wider range of stowing systems is facilitated by a number of legislative measures for the rational use of the earth's interior, adopted in the Republic of Kazakhstan and a number of other countries. Of particular importance are systems with a stowing in the development of deposits at great depths, where the fight against increased rock pressure and the occurrence of rock blows is impossible without the use of strong backfilling arrays.

The high cost of stowing at a number of enterprises is one of the main factors holding back more of its use. The main factors determining the high cost of the stowing are the costs of binders (for hardening mixtures) and the insufficient level of mechanization of filling operations.

Of all the known methods of stowing, the most promising are mechanical types with pipeline transport. When mining coal, there is a tendency for further use of the hydraulic and pneumatic method of laying. In the mining industry, a hardening stowing is widely used, which ensures a greater completeness of the excavation and reliable protection of the work-in-process.

The application of high-quality cements by many enterprises of our country significantly increases the cost of the stowing, but only the activity of the binder is only partially used. For filling operations, cement of a brand not higher than 250 is required, which is produced by the
domestic industry in very small volumes. The transition to low-quality binders (cement and ground granulated slags), the search for new more economical binders and fillers, which can be used as tailings concentrators, will undoubtedly promote the spreading of hardening laying in mining practice.

To date, the urgent task is to create high-quality new-generation packing materials based on the management of the processes of structure formation at the micro- and nanoscale levels. Applied interest in nanosystems is due to the possibility of creating rational structures of filling composites due to their significant modification in the transition to the nanoscale, which is accompanied both by a fundamental change in the properties of known materials and by the creation of neocomposites.

At the present time, significant progress has been made in the development of technology and the improvement of the technology of filling operations. Mechanization and automation increasingly reduce their labor intensity and cost, therefore, the technical and economic indices of field development by systems with a stowing are higher in comparison with other types of systems. The introduction of advanced development systems with a stowing will significantly expand the raw material base of many mining enterprises and increase their efficiency.

CONCLUSION

A promising direction for improving laying at mining enterprises should be considered as the use as fillers of tailings of concentrating mills and empty rocks from tunneling works, and as components of a complex binder - ground granulated slags. Re-grinding of ground granulated slags in an ultrathin grinding mill allows to significantly reduce the cost of filling operations.

The use of self-flowing or gravity-pneumatic transportation of filling mixes to the flanks of newly developed plasticizer additives deposits and pressure concrete pumps makes it possible to conduct laying operations at mining enterprises without additional costs for the construction of new packing systems.

Designed at the level of inventions, the scheme for reclamation of the rock from excavation work to the laying without issuing an empty rock to the surface provides a highly efficient and environmentally friendly operation of the mining enterprise.

Effective management of structural changes in cement systems will allow to optimize the costs of raw components of filling mixes and will make it possible to significantly reduce costs in the production of stowing operations at mining enterprises, to utilize waste from mining and processing and metallurgical production as a stowing and to ensure safe mining conditions.

BIBLIOGRAPHY


NUMERICAL MODELING OF GOAF EDGE NOTCH EFFECTS ON INDUCED CAVING DURING PILLAR EXTRACTION UNDER MASSIVE ROOF STRATA

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ABSTRACT

Indian coal measures rocks are often found to be strong and massive with varying degree of sandstone contents. The exposed strong and massive roof during depillaring is found difficult to cave-in and also the sudden release of accumulated stress in large area may lead to the risks of air blast, overriding, and premature collapse of coal pillars in working panel. This situation in any working mines poses a serious challenge to the workplace safety. Though several techniques have been tried in different geomining conditions, induced caving using blasting techniques are widely used due to its high flexibility to collapse the massive roof in a controlled way. The blasting technique uses trial and error method mainly to create a weakness line or notch of suitable dimension along the goaf edge that helps to detach the hanging roof in the goaf area. However, the notch effect on induced caving has not been studied yet numerically to quantify these effects. In this direction, this paper developed a numerical model of goaf edge notch to estimate the relief of abutment pressure in the coal pillars of working panel. This has been performed using finite element analysis on the strong and massive roof exposed after the depillaring in bord and pillar method of an Indian coal mine. The different dimensions of the notch parameters for controlled collapse of exposed roof have been studied using finite element analysis. The different dimensions of the notch to relieve the stress on pillars have been simulated and thus the deformation in roof layer to detach the hanging roof has been estimated. The numerical analysis performed in this paper provide an effective approach to estimate the effects of goaf edge notch parameters to trigger induce caving of the overlying strong and massive strata for higher safety and productivity in coal mines.

KEYWORDS
Bord & pillar method, Induced caving, Coal mining, Depillaring, Numerical modeling, Goaf edge notch

INTRODUCTION

A large number of coal seams have been developed using bord and pillar (B&P) method of mining which is a dominant method of coal mining and accounts for more than 90% underground mines in India (Singh, 2005; Raghvan et al., 2014). The extraction of these developed coal pillars with caving, termed depillaring, under strong and massive roof has always been a problem and challenge for mining engineers (Sheorey et al., 1978; Das, 2000). The major problems associated with strong roof are dynamic loading on supports, extension of Goaf in workings, loss of coal in pillars due to high pressure experienced by ribs and splitted pillars, and risk of air blast due to large standing area of Goaf [Mathur, 1992; Ghose & Dutta, 1987; Tan et al., 2005]. Additionally, the exposed area increases influence of abutment zone in the mining district causing spalling and sometimes roof deterioration in geologically disturbed area. The severity of such problems depends directly on the lithology and rock types of the overlying strata of a coal mine (Brady & Brown, 2014; Sarkar, 1998).

There are several approaches to manage the risks of depillaring under strong and massive roof such as induced caving of immediate roof using blasting in goaf either from underground or from surface (Konicek et al., 2013), high-pressure water injection (Holt, 1989), hydraulic fracturing (Mills et al., 200; Van & Jeffery, 2000), and so on. Though every method has some advantages and shortcomings, induced caving by underground blasting is quite popular in Indian B&P mining at greater depth (Banerjee et
al., 2003). The blasting can be used either for breaking the immediate roof with good fragmentation and bulking factor so that it fills the goaf adequately to prevent overriding and air-blast or for weakening the roof by modifying its geotechnical characteristics by creating artificial planes of weakness in it to reduce its physic-mechanical strength. The weakening the roof can be done by pre-fracturing at the highly stressed edge to trigger the fall in strong rock (Choudhary, 2002).

The artificial planes of weakness, termed notch, can be created at goaf edge during depillaring to induce the caving of overhanging roof strata using induced blasting with a rightly designed blasting pattern and charging (Xia & Yan, 2012; Zhang et al., 2015). The detachment should be controlled so that the fracture does not get extended towards the working area and stress over the working pillars can be relieved. The analyses of induced caving problems using beam and plate theory have investigated widely for longwall method of mining (Sofiano, 1996; Gale & Nemcik, 1998; Hosseini et al., 2014, Guo et al., 2017). Some of the works are found on numerical simulation of caving in Longwall method (Jinpingle et al., 2002; Manteghi et al., 2012). However, the problem is associated with the collapse of massive roof strata over coal pillar under extraction in B&P method of coal mining which differs with that in Longwall method (Hutchinson, 2012; Van der Merwe, 2005). Therefore, the different design parameters of blast-induced caving in B&P method requires to be studied through field and numerical studies.

The trial and errors method are generally practiced to determine geometry and pattern of the blast holes parameter to create a suitable sized notch in B&P method of coal mining. For example, in Churcha colliery of South Eastern Coalfield of India, a blast design to create notch along working face length is regularly practiced with an average pull, termed as notch depth, of 3m (CIMFR report, 2000). However, the same design parameters of the notch such as base, depth, and angle of the final pull may not be suitable to all thickness, length, width, loading patterns, and rock properties of the immediate roof strata. It takes a great length of time and is a very costly exercise to estimate the effective notch depth and angle to initiate the collapse in a working mine. The modelling of controlled collapse of the overhang roof strata taking into account of the notch effects at the goaf edge in B&P mining has not been reported in the literature. The quantification of the notch effects in terms of the stress reduction in coal pillars at goaf edge and deformation in the immediate roof layer to trigger the collapse of the roof optimally in a controlled way has not been done yet.

In this direction, this paper presents a 3D finite element modeling of overhanging immediate roof strata and the effects of the notch on it. The models are developed taking into account of the site conditions and depillaring operations. The beam collapse occurs in three phases which get repeated over the depillaring cycles within a panel. The beam is collapsed for the first weighting with all sides supported. In the second phase, the beam is collapsed with three side supported and one side free. In the third phase, the beam collapses with two sides supported and two sides free. All the three cases are required to be analysed with respect to notch dimension. The total 90 models of the overhang beam with and without notch are solved with respect to the notch dimensions, beam thickness, beam length, and beam materials.

MINE SITE AND NOTCH DESCRIPTION

Mine Site

The mine is situated in the western part of Sonhat Coalfields which constitute the central portion of Jhilimili – Sonhat – Sohagpur master basin. The lower Barakar measures are the Coal bearing rocks. Seam-V being worked has gradient of 1 in 18 (var.) due North 40° West. The geological and mineable reserves are 38 and 25 million tonnes with heat values ranging from 2435-7037 Kilocalorie/kilogram. Specific feature of the mine is the presence of numerous faults and dykes. The property is criss-crossed by
a number of normal local faults running approx. along N-S direction (throw 0.3m to 11m). Many times these fault planes have decided the extent of goaf falls. Slip planes are frequent and sand stone dykes having thickness of about 1m are also found frequently.

Originally, the mine property was planned to be exploited by powered support retreating Longwall Method by shearer. Longwall was started in early 1990 but the face collapsed in the middle of the year 1990. Thereafter, it was proposed to exploit the property by mechanised B&P method by deploying side discharge loaders (SDLs), light duty chain conveyor (LDCC), Belts etc. Presently following B&P development and depillaring districts are being worked by SDLs. Currently, the developed pillars of panel 57 of the mine are being extracted under strong and massive roof strata. The Fig.1 shows a typical layout of the panel-57 located at 237m depth.

![Figure 1- The layout of the panel 57 under study](image-url)

A typical lithology of the study area is shown in Fig.2. The specific feature of the coal deposit is the presence of thick Dolerite Sill which is present 48.1m to 200m above the coal seam and has thickness 14.64m to 140.65m. Fig.2 shows the lithology of roof rock in 57 level panel above the coal seam.

![Figure 2- A typical lithology of the mine and lithology of the seam-V of 57 panel](image-url)

**Methods of Stress Relief on Working Coal Pillars by Creating Notch at The Goaf Edge**

In B&P method, arrangement of coal pillars is highly regular in geometry and it repeats itself over a relative distance. These pillars are developed in a quadrilateral panel which is separated from other panel
by leaving barrier pillar as shown in Fig.1. Pillars are extracted sequentially from dip to rise maintaining diagonal line of extraction. In principle, a coal seam below the earth crust is loaded with the weight of overburden. Thus an opening, made by an underground extraction of the coal, disturbs the natural state of stress equilibrium.

The vertical load of rock strata, directly above the opening, shifts onto surrounding pillars leaving a destressed zone in the roof strata just above the opening. This process results in a higher stress, called mining induced stress, over the coal seam around the excavation as compared to that of the pre-mining stress condition. In order to relieve this induced stress on working pillars, in Indian coal mines, it had been the practice in the mine to do roof blasting in all the original gallery junction roof up to a height of about 1.5m but it was found to be ineffective.

With trial and error, in some of the mines, it has been recommended on the basis of local geomining conditions that blasting at least up to 5m height above roof level should be done to induce caving in goaf, stop extension of caving inside workings and to reduce over hangs and chances of air blast during caving in goaf. In fact, this practice created a notch in the rover-hanging roof strata as shown in Fig.3 with a triangular shape.

Accordingly long hole roof blasting to induce caving was introduced in panel and the success met in the very second blasting which resulted in the main fall 3 hours after blasting.
The blasting pattern, as shown in Fig. 4, has been established on the basis of local geominning conditions and several field trials and adjustments. However, the effects of the notch need to be quantified with respect to the depth, thickness, and rock material, overhang length of the immediate roof strata.

Properties of Coal Measures Rock

Table 1- Physico-mechanical properties of rock upto 25 m above and below the coal seam.

<table>
<thead>
<tr>
<th>Bed</th>
<th>Core Run (m)</th>
<th>Lithology</th>
<th>Density t/m³</th>
<th>Young’s Modulus (GPa)</th>
<th>Poisson's Ratio</th>
<th>Compressive Strength (MPa)</th>
<th>Tensile Strength (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>212-237</td>
<td>Fgd-Mgd-Sst</td>
<td>2.23</td>
<td>6.07</td>
<td>0.25</td>
<td>17.63</td>
<td>2.24</td>
</tr>
<tr>
<td>2</td>
<td>237-240</td>
<td>Coal</td>
<td>1.40</td>
<td>2.000</td>
<td>0.25</td>
<td>18</td>
<td>3.19</td>
</tr>
<tr>
<td>3</td>
<td>240-265</td>
<td>Fgd-Mgd-Sst</td>
<td>2.23</td>
<td>6.07</td>
<td>0.25</td>
<td>17.63</td>
<td>2.24</td>
</tr>
</tbody>
</table>

Fgd-Fine Grained, Mgd-Medium grained, Sst-Sandstone

Table 2- Measured relation of in-situ stress with depth at Churcha Mine

<table>
<thead>
<tr>
<th>Depth</th>
<th>In situ stress relation with depth</th>
</tr>
</thead>
<tbody>
<tr>
<td>0-180m</td>
<td>$S_h=1.2750+0.0128H$</td>
</tr>
<tr>
<td></td>
<td>$S_m=2.2824+0.0202H$</td>
</tr>
<tr>
<td>180-350m</td>
<td>$S_h=2.9362-0.0356H$</td>
</tr>
<tr>
<td></td>
<td>$S_m=11.327-0.0848H$</td>
</tr>
</tbody>
</table>

DEVELOPMENT OF 3D FINITE ELEMENT MODELS

Goaf edge notch model

A three-dimensional finite element model of exposed immediate roof resting on surrounding coal pillars is developed with roof and floor rock strata considering the lithology as provided in Figure 1 and material properties as given in Table 2. The length and the width of the numerical model are 195m and 45 m which are also equal to the width of three galleries of 5m plus four square coal pillars with 45 m side of the square coal pillar. The front view of the mesh model is shown in Fig.5 which has been developed with combination of hexahedral and tetrahedral elements. The Mohr–Coulomb failure criterion has been used in this model.

This study takes the full width of the coal pillar as goaf edge length in the model which is 45m. The length of the exposed roof strata is 105m with two pillars extracted out. The thickness of the roof layer has been taken as variable in the simulation. The thickness of floor is 25 m and the rock material is taken as fine-grained sandstone similar to the roof strata. The thickness of the coal seam is 3m situated at depth of 237m from the surface.
In finite element analysis, the X-direction is taken along the width of the goaf, Y and Z axis directions are along the thickness towards the earth surface and along the goaf length, respectively. The boundary conditions are shown in the Fig.5(a). The bottom and sides along the goaf length of the model are constrained for vertical and horizontal displacement. The measured in-situ stresses, as per the relations given in Table 2, have been applied on both Y-Z planes. Additionally, the overburden load of $\gamma H$ Pa (specific weight*depth) is applied uniformly on the area of roof layer just above the coal pillars. However, no vertical load has been applied on the area of exposed length of the roof layer because this study assumes that the lowest layer is separated under the influence of arch formation and self-weight. A total of 81 models are developed by varying roof thickness, notch angle, notch depth, and orientation of the supported sides.

RESULTS

The summary of results of finite element models is presented in the following sections considering the effects of notch parameters (angle and height) and repeated geometries of roof beam (4-sides, 3-sides, 2-sides supported) formed in the goaf area during the diagonal extraction of the pillar in B&P mining. The effects of the thickness of the roof layers with notch parameters have been investigated.

**Average Stress in Pillar along Goaf Edge with 4-Sides, 3-Sides, and 2-Sides Supported**

Average stress is calculated by averaging the stress values over the coal pillar up to 6m along the centre line perpendicular to the goaf edge. The three cases of 4-sides, 3-sides, and 2-sides supported geometries of the models with and without notch have been studied by varying the values of roof thicknesses, notch angles and notch heights. The graphical results of stress vs. notch heights for these cases are illustrated in Fig.6, Fig.7, and Fig.8.
Figure 6- Average stress in pillar across goaf edge with four sides supported
Figure 7- Average stress in pillar across goaf edge with three sides supported

(a) For immediate roof thickness 10m
(b) For immediate roof thickness 20m
(c) For immediate roof thickness 25m
Maximum Deformation in Roof Layer with 4-Sides, 3-Sides, and 2-Sides Supported

Maximum deformation in roof layer is taken practically over the entire length and it usually recorded at the centre of goaf area. In this study, the three cases of 4-sides, 3-sides, and 2-sides supported geometries of the models with and without notch have been studied by varying the values of roof thicknesses, notch angles and notch heights. The graphical results of maximum deformation vs. notch heights for these cases are illustrated in Fig.9, Fig.10, and Fig.11.
Figure 9- Maximum deformation in roof with four sides supported

(a) For immediate roof thickness 10m
(b) For immediate roof thickness 20m
(c) For immediate roof thickness 25m
(c) For immediate roof thickness 25m

Figure 10- Maximum deformation in roof with three sides supported

(a) For immediate roof thickness 10m  (b) For immediate roof thickness 20m
DISCUSSION

The results obtained in this numerical study indicate that the notch creates a weakness plane along the goaf edge which reduces the stress on pillars and increases the deformation in roof layers above the goaf area.

As per the graphical results of average stress in coal pillars, as shown in Fig.6, Fig.7, and Fig.8, the largest relief of average stress is observed with notch height of 7m, notch angle of 70 degree in 25m thick roof layer with four sides support. However, the second best stress reduction on coal pillar has been observed in a 20 thick roof layer with notch height of 7m and notch angle of 55 degree with three sides support. Therefore, it can be inferred that the notch effects on stress relief over the coal pillars in working area vary with geomining conditions.

Similarly, the maximum deformation recorded through simulation run is shown in Fig.9, Fig.10, and Fig.11 with respect to the varying notch parameters and roof layer thickness. The maximum deformation has been observed in 10m thick roof layer with 2 sides supports, 7m notch height, and 70 degree notch angle. Moreover, the deformation declined as we go for greater thickness of the roof layer and number of side supports. Therefore, it can be sum up that notch can weaken the exposed roof and potentially enhances the roof fall depending upon the geomining conditions of the extraction panel.

Though the finite element model presented in this paper quantified the effects of notch on caving phenomena in massive roof, the further study required to be taken up for varying geomining conditions for better design of the notch. The model should also consider the jointed rock mass in the modeling to arrive at practical results.

CONCLUSIONS

This study developed a finite element model of immediate roof layer formed while extracting the developed pillar by the method of bord and pillar mining. The notch effects have been investigated in the model to quantify their effects on the caving behaviour of the roof layer. The major outcomes of this study are notch with more depth and higher angle can reduce stress on working pillar and increase the chance of the failure of exposed roof in the goaf area. Therefore, a numerical approach
demonstrated in this work can eliminate the trial and error method to establish the right dimension of notch to trigger the caving in board and pillar mining of coal deposits under massive roof. The future work may be taken up to derive realistic design of the notch considering the jointed rock mass and varying geominning site conditions.

REFERENCES


OPTIMISATION OF GATEROAD SUPPORT AT THE GREAT DEPTH IN HARD COAL MINE

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ABSTRACT

The European hard coal industries is facing a problem of excavation at great depths, in many cases exceeding 1000 m. Such a situation results from the fact that significant vertical and horizontal convergence values occur in the gateroads and pressure to ensure its functionality (required dimensions) continues to increase.

This paper presents the complete process of gateroad support optimisation targeted for given geological and mining conditions in one of the hard coal mines. The process consists of the underground investigations on performance of a support system designed originally by colliery and on rockmass behaviour. It is described by the gateroad deformation, support load, and dimensions of the fractured zone in rockmass around the gateroad. All of the information is then reproduced in numerical models and once the models are calibrated, the work to optimize the existing support systems is undertaken. The following means are considered during optimisation: implementation of additional reinforcements of steel arch yielding support (flexible bolts, stringers etc.), steel parameters up-grading, implementation of heavier V profiles of steel arch yielding support (arches and bolted stringers), change of location of bolted stringers. The main advantages of new developed support systems constitute: increased load bearing capacity; improved stress distribution in particular support system elements achieved by additional reinforcements implementation, reduced unit load exerted on primary support elements (steel arches), better utilisation of load bearing capacity of support scheme elements. The last stage of the optimisation process is a validation of the results by underground application and tests of new support systems in similar conditions as in the first stage of the optimisation process.

The article was prepared based on research conducted within the Research Project: Advancing Mining Support Systems to Enhance the Control of Highly Stressed Ground (AMSSTED) – co-financed by European Commission - Research Fund for Coal and Steel (Grant Agreement RFCR-CT-2013-00001) and by the Polish Ministry of Science and Higher Education (Contract 3146/FBWiS/2014/2).

INTRODUCTION

When applying a high efficient longwall extraction system in a hard coal mine, the proper design of gateroad support plays a crucial role. Proper means such a design which ensures that for given mining conditions, workings are stable and of proper size during its life.

In the support design process, it is indispensable to assess and consider many parameters which can be grouped into two categories: the technical parameters of the gateroad, and the geomechanical parameters of rocks around the gateroad. The technical parameters (e.g. purpose of the roadway; shape and dimensions; cross-section area; available support in terms of shape, material and load capacity etc.), depends directly on the implemented extraction system, thickness of extracted coal seam, air demands, dimensions of mining equipment, expected value of roadway convergence and natural and technical hazard which determine the roadway performance (Hucke et al. 2006, Hebblewithe, Lu 2004). The geotechnical parameters of rock in the roadway proximity as well as value and mechanism of load exerted on the support (Bigby 2004, Junker et al. 2009, Lawrence 2008, Barczak 2005) and quality of rocks and mining conditions e.g. dynamic load or corrosion) depends on the geological structure of rockmass, depth, and type of previous and ongoing extraction (Cartwright, Bowler 1999, Colwell, Firth, Mark 1999, Mark 1998, Snuparek, Konecny 2010).

Aforementioned parameters constitute an input data to empirical methods of support design and to numerical modeling design codes. Both load exerted on support and bearing capacity are determined by means of empirical codes (Prusek 2010, Lubosik, Prusek 2010) and by numerical modelling, commercial (COSMOS, ANSYS, RS, FACE, FLAC) or authorial (ABC RAMA) codes (Bock, Prusek, Rotkegel 2009, Walentek et al. 2009, Prusek 2008), while geotechnical parameters and rockmass behavior is determined in laboratory or in situ tests by means of e.g. stiff press, penetrometer, borehole camera, dynamometer,

In Czech hard coal mines, the main roadway support is a steel arch yielding support made with a TH shaped profile which usually consists of three or four arches linked with bolted clamps. Connection between support frames (steel sprags), and lagging (armored concrete prefabricated elements and/or steel-mesh elements) and filling the space between the rock mass and support with stones is similar to that used in other deep mines which use yielding steel arches. In the Czech coal deep mines the gateroads are from 14 to 20 m² in cross section. The most–commonly-used distance between support frames is 0.5m but other distances 0.7, 0.8, and 1.0m are also used. Support frames are made from profiles of elementary mass from 24-36kg/m. The most commonly-used profile is 29kg/m but the most-massive profiles, with elementary masses from 34–36kg/m (TH 34 and TH 36), are used in the case of difficult geological and mining conditions. That arches are often reinforced with additional components. Outby the longwall face, the support is reinforced with the most commonly used: mechanical and hydraulic props (friction and hydraulic) and also with: wooden props, steel TH-profile bars (horsehead), wooden on side cut bars, and bolted roof-bar arches by using a steel bar, full-length grouted anchors, or flexibolts. Currently, Czech coal mines start installation of bolted TH-steel bars by using flexibolts to hang roof arches in higher and stronger strata and then roof-bolting between support frames. All of these reinforcement options (also in combination) are used to enhance the stability of longwall–gateroad T-junctions and minimalize the number of props in this area.

This paper describes an example of gateroad support optimization for longwalls in seam 24, where first the mechanism of support deformation was determined and then the new support solution characterized by higher load capacity was developed.

GEOLOGIC AND MINING CONDITIONS IN LONGWALL PANELS X AND Y

The testing of the support system was conducted in gateroad A for determination of support load and deformation mechanism and in gateroad B for application of new support systems. Both gateroads were longwall headgates. The testing workings were located in seam 24, next to extracted longwall panels. Seam 24 is deposited in this area at a depth of circa 1000 - 1050 m, and has a dip of about 5°, directed to the south-west (Figure 1). Within the panels, seam 24 has a thickness of around 1.7 m. In the roof of the seam there are: 7.25-10.0 m of sandy shale, 1.55 m of sandstone and another 2.2 m of sandy shale. The floor of seam 24 is built of 1.24 m sandy shale (Figure 1). The strength parameters of the coal and surrounding strata are listed in the Table 1.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Roof strata UCS</td>
<td>23.67 – 40.61 MPa</td>
</tr>
<tr>
<td>Floor strata UCS</td>
<td>23.33 MPa</td>
</tr>
<tr>
<td>Coal seam UCS</td>
<td>9.89 – 13.00 MPa</td>
</tr>
</tbody>
</table>

The gateroad B, was located below the gobs in extracted seam 23 and upper bench of 24 seam. The distance between gateroad and the gobs was respectively about: 30 m and 20 m. Longwall 11495 was about 205 m in width and its height was 1.7 m.
ASSESSMENT OF ROCKMASS BEHAVIOUR AND GATEROAD SUPPORT PERFORMANCE

The first stage of the measurements was devoted for establishing the gateroad support performance as well as collecting the data for determination of support loads and deformation mechanisms. For that purpose, the measurement station was installed in gateroad A (see MS No. 0 in Figure 1). The support of this gateroad designed by the mine was an arch yielding support type SPN-16 made of steel with parameters: yield stress 350 MPa and ultimate strength 550 MPa. The set distance was 0.5 m. No reinforcements were used (Figure 2).
Methodology of underground measurements

Measurements of rock mass deformations around the gateroad A and load exerted on its support were carried out in measurement station which was set at the distance of 120 m before longwall face (Figure 3).

![Figure 3 - Scheme of the measurement station in the gateroad A](image)

1) – 10.0 m borehole for a penetrometer and endoscopic camera tests; 2) – 3.0 m borehole for a penetrometer and endoscopic camera tests; 3) – 5.0 m borehole for a penetrometer and endoscopic camera tests; 4) – 10.0 m borehole for an endoscopic camera tests; 5) – 2.4 m instrumented rockbolt; 6) – three-position manual-reading telltale; 7) – hydraulic dynamometers; 8) – points of measurements of horizontal and vertical convergence.

The following quantities were measured: uniaxial tensile and compressive strength of roof, floor and sidewall by means of the hydraulic penetrometer, vertical convergence and horizontal convergence, roof movement by means of triple height telltales, fissures propagation – by means of the borehole camera, load exerted on the SPN-type support sets – by means of hydraulic dynamometers, rockbolts axial load – by means of strain gauged rock bolts. The measurements were carried out periodically according to the assumed intervals which were dependent on the longwall face advance.

Measurements results

The measurements in the gateroad A were conducted during the period of 4 months. The results are presented as graphs depicting: change of height and width of the gateroad (Figure 4), values of roof strata displacement and fissure propagation around the gateroads (Figure 5), load on the SPN support set and distribution of axial forces in the instrumented rock bolt (Figure 6).
The above presented results show that the approach of the advancing longwall face caused the value of the vertical convergence to be 1430 mm and the horizontal convergence - 1640 mm (Figure 4). The underground measurement showed that a phenomenon of floor uplift was crucial in the vertical convergence of the gateroad. It constituted approx. 86% of the total vertical convergence value. This fact is confirmed by the registered readings of the telltales where maximum value of displacement in 10 meters of roof rocks reached 200 mm (Figure 5).

Based on the above-specified results of measurements concerning the gateroads deformation, the minimum value of the cross section area that was within the T-junction was calculated. This is of vital importance in the design of ventilation of the entire area of the longwall, in particular in the case of mining operations within the seams under high methane hazard. The reduction amounted 53% (from 18.9 m² to 8.75 m²).

The endoscopic tests results (Figure 5) showed that the influence of longwall extraction at great depths did not cause a significant increase of the fracture-zone extent but only an increase of the number of fissures and their total stratification. No changes in the maximum range of roof rock fractured zones
between the first and the last measurement were registered, and it was 9.3 m. However, an increase of the number of fissures was noted. Such a situation is considerable, especially in case of a rock bolt support.

The analysis of measurement results concerning the SPN support load carried out using hydraulic dynamometers indicates that the effects of the abutment pressure are noticeable at a distance from ca. 120 m before the longwall face. From this distance, a systematic increase in the load occurs. The total maximum load of a single SPN arch (longwall side and sidewall) set with a spacing of 0.50 m was 230 kN (460 kN/m) (Figure 6).

![Dynamometers](image1.png)  ![Instrumented rockbolt](image2.png)

**Figure 6- Results of measurements of arch and rockbolt support load for gateroad A**

The maximum values of axial forces exceeded the permissible load-bearing capacity of the instrumented bolt specified by the manufacturer of 180 kN and reached 315 kN (Figure 6). All the registered load values in anchors constituted tensile forces. According to the studies, the greatest changes in deformation were registered by tensometers located along the bolts that were between 1.5 and 2.0m. Moreover, a significant increase in the value of axial forces in the bolts was recorded when the longwall face reached the bolt.

**NUMERICAL MODELLING: MECHANISMS OF GATEROAD DEFORMATION AND SUPPORT SCHEME OPTIMIZATION**

**Introduction to numerical modelling**

The mechanism of gateroad deformation was determined on the base of both: the results of underground measurements carried out in neighbouring roadway and support performance by means of COSMOS/M program (COSMOS/M 1999). In course of numerical calculations in discreet model of arches, directional support reactions were compared with the results of underground measurements (e.g. indications of dynamometers installed under sidewall arches).

In order to optimize the support scheme in the gateroad, the steel arch support calculations were conducted by applying FEM method in COSMOS/M program. The mechanisms of gateroad deformation (value and distribution of load) and support behaviour model allowed for conducting of the cycle of strength calculations aiming to optimize the schemes of support systems used in the considered gateroad. In the numerical modelling, the following strength criteria were assumed: - maximum stress in support set elements must be smaller than stress permissible for used steel profiles, - maximum stress in the reinforcement of support sets (steel stringers) must be smaller than stress permissible for profiles, - load of bolts must be smaller than their maximum load capacity.

In the first stage of strength calculations, the model for mapping the construction form of support scheme used in the measurement station, had been created with the application of a CAD program. Further,
using the COSMOS/M program, which calculates the reduced stress according to the Huber-Mises-Hencky hypothesis, corresponding cross-section and material parameters were assigned to particular elements.

Material coefficient \( \gamma \) and material plastification coefficient \( n \) were calculated for the applied sections, which were used for the making of particular elements of the support. Their values for steel used for mining supports were assumed in accordance with standard PN-H-93441-1 and PN-H-84042.

Coefficient of plastic reserve of cross-section \( m \) is strictly related to the shape of the cross-section of support element-support set arch. It is equal to the ratio of plastic section modulus to the bending section modulus:

\[
m = \frac{W_{pl}}{W_s}
\]

The coefficient of plastic section modulus is equal to sum of absolute value of static moment of compression and tension cross-cut area in relation to neutral axis in state of full plastification.

\[
W_{pl} = |S_c| + |S_t|
\]

The permissible stress, determined in the a/m way cannot be exceeded by the reduced stress, calculated - for example - with the application of FEM program:

\[
\sigma_{red} \leq \sigma_{dop}
\]

**Numerical modelling of optimised support schemes**

Prior to the optimization phase of the gateroad B support scheme, the deformation mechanism of gateroad has been determined. Based on the geological-mining conditions analysis and previous underground observations asymmetric loading support with higher load on the side opposite to a longwall has been stated (Table 2).

<table>
<thead>
<tr>
<th>Gateroad B</th>
<th>Legend</th>
</tr>
</thead>
<tbody>
<tr>
<td>Load direction - perpendicular to support</td>
<td></td>
</tr>
<tr>
<td>Load direction - perpendicular to seam</td>
<td></td>
</tr>
<tr>
<td>Support movement direction</td>
<td></td>
</tr>
<tr>
<td>Values recorded on dynamometers &quot;+&quot; higher; &quot;-&quot; lower</td>
<td></td>
</tr>
</tbody>
</table>

According to the above presented methodology and results of measurements and analysis the optimization of the support scheme of gateroad B had been made. Steel arches were reinforced with one or two rows of bolted stringers (long tendon (flexible) bolts). Due to limited mine experience with arches set with the distance bigger than 0.5 m, three support systems were established for testing. In two of the
proposed support schemes, the arches were set with a distance 0.67 m, and in one, the distance was 0.80 m. Table 3 shows the developed improved support schemes of the Gateroad B.

Table 3 - Gateroad B optimised support schemes

<table>
<thead>
<tr>
<th>Numerical calculation</th>
<th>Comments</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>MS No 1</strong></td>
<td>Additional two rows of flexible bolts with additional stringer. Increase distance between arches from 0.5 to 0.67 m. Bearing capacity: increase by 40%.</td>
</tr>
<tr>
<td><strong>MS No 2</strong></td>
<td>Additional one row of flexible bolts with additional stringer. Increase distance between arches from 0.5 to 0.67 m. Bearing capacity: increase by 0%.</td>
</tr>
<tr>
<td><strong>MS No 3</strong></td>
<td>Additional two rows flexible bolts with additional stringer (both side). Increase distance between arches from 0.5 to 0.8 m. Bearing capacity: increase by 32%.</td>
</tr>
</tbody>
</table>

The study showed that the load bearing capacity of innovative support systems can be increased up to about 40% in relation to support system used by colliery what occurred in optimization scheme no 1 where the distance between arches was increased from 0.5 to 0.67 m and the two rows of flexible bolts and stringer were added to reinforce the steel arches. When the distance between arches was increased to 0.8m, the bearing capacity increased only by 32%.

It was decided that all developed optimised support systems would be tested in underground conditions.

**UNDERGROUND TESTING OF OPTIMISED SUPPORT SYSTEM**

The measurements in the gateroad were conducted during the period of three months. The results are presented in Table 4.
The above presented results prove that application of optimised support system improved the maintenance conditions of gateroad B when comparing to gateroad A. In all measurement stations significant reduction of convergence was observed. Even in MS No. 3 where the distance between arches was increased to 0.8 m the convergence was lower by 450 mm (vertical) and 990 mm (horizontal).

Additionally, it appears that the application of bolted stringers not only reduces the convergence, but also improves functioning of all elements of support system and its interaction with surrounding rock mass. It is shown by lower values of roof displacement which was only 15-30 mm in gateroad B and 200 mm in gateroad A. Also the range of fractured zone was reduced to 1.0 - 4.1 m while in gateroad A more than 9.0 m of roof was fractured. Smaller fractured zone had an impact on lower load exerted on standing support elements. The total maximum load of a single SPN arch (longwall side and sidewall) measured in MS No 2 and 3 was 215 kN and 148 kN respectively. Moreover considering higher distance between arches calculated load per one meter of gallery was 320kN/m (MS No. 2) and 187.5 kN/m (MS No. 3). On the other hand, the measurement of showed that increasing distance between arches without additional reinforcement causes heavier convergence. In MS No. 3 – (where distance between arches was 0.8 m) both vertical and horizontal convergence were higher than in MS No. 1 where the distance was 0.67 m and bolting scheme was the same.

**CONCLUSIONS**

In the roadway life when the rockmass movements of various intensity (e.g., roof sag, floor heave, and horizontal convergence) as a result of the extraction pressure impact affected support performance, coal mine management is responsible for applying the appropriate support in the roadways which ensures stability of the workings and work safety for the miners.

The method of roadway support design presented in this paper based on the underground measurements and numerical modelling allows to design economically-friendly solutions of support schemes characterized by higher load bearing capacity. The paper presents whole course of support scheme optimization starting with the test of support designed by mine in gateroad A and later testing the optimized support systems in gateroad B. As this gateroads were localized in adjacent longwalls (Figure 1), the geological and mining conditions were similar.

Implementation of developed optimized support system, where the distance between arches was increased and additional flexible bolts with stringer were implemented, reduced the convergence of...
gateroad B in relation to gateroad A. The highest reduction was measured in MS No. 1 where vertical convergence was lower by 51% and horizontal by 70%.

Based on the above presented results, it can be stated that numerical modelling codes, supplemented by good quality input data which allow to calibrate the models, are suitable for designing appropriate support schemes for given mining conditions.

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INDUSTRIAL SAFETY NEW TECHNICAL SOLUTIONS FOR UNDERGROUND MINING

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ABSTRACT

One of the initiative projects, which are currently being implemented, is "Smart Mine". The main objective of the project is to ensure the safety of personnel in underground mining. One of the main components of the “Smart Mine” is operational uninterrupted communication and alert, real – time determination of the exact location of personnel and equipment. Location systems, used at the mining enterprises, have some advantages and disadvantages. Most of these systems do not have real-time accurate positioning. There is only zoning, which implies the location of personnel in a zone and does not provide every second tracking with the required accuracy. Automated Industrial Safety System “ASP&A-mine” is developed at the Mining Institute named after D.A. Kunayev in accordance with the modern requirements to the accuracy of location, volume of information transmission and the shortcoming of existing systems. “ASP&A-mine”, based on the complex heterogenous Wi-Fi Mesh-network, is a high-speed multi-channel system of on-going data collection, primary analysis and data transmission, ensuring the operation of the dispatcher control room and geoinformational complex.

KEYWORDS

Mine, RTLS, MESH-network, positioning, anchor, individual tag

INTRODUCTION

The fourth industrial revolution and the transition to a new technological structure imply the creation of a fundamentally new model of industrial development. This trend is typical for all branches of industrial development, however, given the importance of the extractive sector in the economy of Kazakhstan, the transformation of the principles of the organization of the raw material industry is the subject of special attention of the state. The Message of the President of Kazakhstan to the people of Kazakhstan "New development opportunities in the fourth industrial revolution" drew attention to the need for a critical re-thinking of the organization of raw materials industries [1].

Meanwhile, implementation of "smart" technologies will not only increase productivity and ensure further development of the resource potential, but also solve a number of challenges in the field of industrial safety in underground mining.

Despite the fact that according to the official data, "accident-free operation of hazardous production facilities has a positive trend" [2], it should be acknowledged that accidents take place at the production facilities, leading underground mining operations and entail human casualties.

At the moment, the processes of safety provision at the mining enterprises are regulated by the "Rules of industrial safety for hazardous production facilities, leading mining and exploration activities" [3]. In accordance with the Rules, the main purpose of industrial safety is to prevent or minimize the consequences of accidents at hazardous production facilities. In particular, paragraph 54 contains requirements to hazardous production facilities, leading underground mining to be equipped with the systems of monitoring, warning of accidents, positioning and searching of personnel, direct telephone and its alternative communication.

One of the initiative projects, which has currently being implemented, is a Digital Mine (Smart Mine). The ideal of the Smart Mine is to minimize / eliminate human presence at underground mining sites. The entire remote control of all production processes. Full robotization of mining, haul and sorting processes.

So far there has been no industrial solutions of such level and it is impossible to exclude the presence of people in underground mining. Therefore, the main target is to ensure the safety of personnel operating at underground mines.

To ensure industrial safety for underground mining there should be used unified solutions or integrated solutions consisting of a number of systems:

- Safety of personnel at underground mines is provided by the solution of the main tasks:
  - communication, alerting and locating of personnel and equipment;
  - status of rock mass monitoring;
  - ambient air & airation control;
  - fire prevention;
  - basic and accessorial technological equipment control systems.

One of the fundamental aspects of the Smart Mine is operational seamless communication and alerting, real-time identification and tracking the location of personnel and equipment (RTLS – Real Time Location System).

The solution of each of these issues is provided by several subsystems. In particular, the solution of communication, alerting and locating of personnel and equipment is provided by individual wired and wireless
communication subsystems, by general and individual alerting subsystem, as well as locating of personnel and equipment subsystem.

Currently some mining enterprises utilize locating systems of such companies as UralTechIS, PBE Group & MRS, Strata, etc.

Each of these systems has its advantages and disadvantages. In particular, most of these systems do not have accurate function real-time location systems (RTLS). There is only zoning, which implies the location of personnel in the zone and does not provide every second tracking with the required accuracy.

Accurate positioning at the moment can be provided by the methods of ToF (Time of Flight) or ToA (Time of Arrival), which are based on the measurement of time of the radio signal from stationary devices to mobile tags.

Focused on these methods devices have their own communication channels that provide interaction with each other, but their bandwidth does not allow to transfer huge volume of information, photo, video and voice data.

In reality because of large number of devices in the network the use of low-speed information transmission channels which leads to increased period between the location definitions to tens of seconds while personnel moving which leads to an error in determining the coordinates from ten to hundred meters.

The only nation product in this field, Automated Industrial Safety System “ASP&A-mine,” is developed in the Mining Institute named after D.A. Kunayev.

This System was developed in accordance with modern requirements to the accuracy of location and the volume of information transmitted and taking into account the shortcoming of existing systems. “ASP&A-mine” is based on the complex heterogeneous Wi-Fi Mesh-network, and it is a high-speed multi-channel system of on-going data collection, primary analysis and data transmission, ensuring the operation of the control room and geoinformation complex.

"MESH is a network topology where devices are combined by numerous redundant connections introduced for strategic reasons".

First of all, the concept of MESH defines the principle of network design with the specific feature to self-organizing architecture that implements the following abilities:

- zones creation of complete informational coverage over a large area;
- network scalability (expanding coverage area and density of information management) in self-organization mode;
- wireless transport channels (backhaul) to connect access points in "each to each" mode
- network stability to the loss of individual elements [4].

MESH is a network topology with the devices, combined with numerous redundant connections via Wi-Fi. MESH-network provides reliable data transmission in case of network segmentation or damage to its individual sections.

The use of high-speed Wi-Fi of MESH - network allows the transmission of photos, video, voice messages and data from various sensors and subsystems, such as:

- Sensors of air and gas control.
- Sensors of rock mass status.
- Data collection from vehicle.
- Sensors of ore quality status.

Meanwhile, the location subsystem for stationary and mobile devices is designed on additional equipment of the CSS standard (Chirp Spread Spectrum). The method of determining the distance between devices is based on the ToF method and provides the accuracy of positioning up to 1 meter in real time (RTLS) with the survey of each of the mobile tags for several times per second [5].

In addition to the distance detection module and high-speed broadband Wi-Fi router, each stationary device includes a system of autonomous reserve power supple and non-volatile memory. Autonomous reserve power supple due to SMART-algorithm allows to flexibly change the operating modes of the equipment to increase the life in the absence of the main power.

Non-volatile memory in the device is designed to store a large volume of operational data concerning the location of mobile tags. The volume of memory can be changed while installing of equipment in underground mines. The volume of operational data depends on the size of the memory and can reach a few weeks or even months. The entire stored information is automatically transferred to the server when the communication channels are restored.
The main processing and long-term storage of data is carried out on the server. The processed data are used by various services and departments for operational management of mining processes.

Friendly interface makes it possible for easy perception of incoming information, control and operational management of current mining processes with remote access possibility (Pictures 1).

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VERIFICATION OF COAL SUSCEPTIBILITY TO GAS-DYNAMIC EVENTS USING MULTIFRACTAL ANALYSIS OF SCANNING ELECTRON MICROSCOPY IMAGES OF COAL SPECIMENS

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ABSTRACT

The relationship between the fractal dimension spectrum characterizing the coal surface structure and coal susceptibility to gas-dynamic events is investigated by means of multifractal analysis of scanning electron microscopy (SEM) images of test coal specimens from outburst-hazardous and outburst-nonhazardous beds. It is demonstrated that the asymmetry of the fractal dimension spectrum for a coal specimen surface can be used to determine quantitative difference in microstructures of coal specimens from outburst-hazardous and outburst-nonhazardous beds: the surface structure of outburst-hazardous coal specimens can be characterized by a fractal spectrum with positive skewness whereas the surface structure of outburst-nonhazardous coal specimens is characterized by a symmetrical fractal spectrum (with negative skewness close to zero). The obtained results are verified for nearly 80% of all analyzed coal specimens. In this study, it is also found ranges of most likely values for the asymmetry coefficient of fractal spectra for outburst-hazardous and outburst-nonhazardous coals.

KEYWORDS
Coal susceptibility to gas-dynamic events; Coal surface structure; Scanning electron microscope; Multifractal analysis; Asymmetry of fractal dimension spectra;

INTRODUCTION

Sudden coal and gas outbursts occurring in coal mines are sudden and rapid destruction of the coal bed (rocks) in mining operations. They are accompanied, as a rule, by a violation of a technological process for coal mining and they are often the cause of underground methane explosions and other accidents with severe consequences.

According to the most widely accepted theoretical view, the risk of gas-dynamic events in coal mines is determined by three main factors: mountain pressure (stress-strain state of the rock massif); gas contained in the coal; the structure and physicochemical properties of coals. Therefore, it is advisable to improve methods for predicting sudden coal and gas outbursts, including those that take into account the coal structure.

At the present time, to analyze the coal structure, it is extensively used the concept of hierarchical structure of geomaterial damages (Astakhov, Bely & Shirochin, 2000; Malyshev, Trubetskoy & Airuni, 2000). This concept was founded by academician M.A. Sadovsky (Sadovsky, Bolkhovitinov, & Pisarenko, 1987), who proposed to consider the rock mass as a block geomedia with a hierarchy of structural levels. In accordance with the concept of hierarchical structure of geomaterial damages, the coal structure can be represented as a fractured porous media with a hierarchy of different size blocks. In this model, small blocks are nested in larger blocks that are enclosed in even larger blocks etc. up to the visible blocks. Coals are divided into blocks by macropores and macrofractures. The main systems of fractures are formed by bedding fractures oriented in parallel to a bed and by two other families which are directed perpendicular relative to each other and to the first system of fractures; such fractures form the largest blocks. Micropores and microfractures vary in wide ranges of size, from 0.3 nm to 10 cm. The inlet diameter of the smallest molecular pores is 0.5–0.7 nm, which is comparable with the diameters of gas molecules. The outlet diameters of macropores and macrofractures can reach 100 nm and more. Outburst-hazardous coal beds have looser micropore structure: they contain few pores with a diameter from 10 to 100 nm and many pores with a diameter less than 10 nm (Malyshev et al., 2000).

Since coals have a hierarchical block structure, the multifractal approach can be used to characterize coal surface structure. Multifractal approach offers to a unique description of the coal surface structure by means of a fractal dimension spectrum (Malinnikova, Uchaev & Uchaev, 2009; Trubetskoy, Ruban, Viktorov, Malinnikova, Odintsev, Kochanov & Uchaev, 2010).

The block structure of coals is identified in the relief of fracture surfaces of coal specimens. Therefore, to characterize multifractal properties of outburst-hazardous and outburst-nonhazardous coals, it can be used coal
specimen images obtained by SEM microscopes. SEM microscopes have some advantages (Zhou & Wang, 2007; Chen, Xu & Chen, 2015):

- great depth of focus, which allows analyzing both polished sections and relief surfaces;
- high analysis speed: SEM microscopes generally only needs several seconds to several minutes to complete a full-spectrum analysis;
- high image magnification: SEM microscopes offer magnification to 1,000,000 times;
- small beam spot: SEM microscopes can perform a micro-area analysis of specimens ranging from several micrometers to hundreds of nanometers.
- SEM microscopes can perform an integrated analysis of the specimen’s composition, morphology and structure.

Paper (Malinnikova, Malinnikov, Uchaev & Uchaev, 2014) demonstrates that the asymmetry of fractal dimension spectra obtained by multifractal analysis of SEM images of coal specimens can be used to determine a quantitative difference in microstructures of coal specimens from outburst-hazardous and outburst-nonhazardous beds. In this study, a relationship between the asymmetry of fractal dimension spectrum characterizing the coal surface structure and coal susceptibility to gas-dynamic events is investigated by means of multifractal analysis of SEM images for test coal specimens from outburst-hazardous and outburst-nonhazardous beds. The main aim of this investigation is to demonstrate that the surface structure of outburst-hazardous coal specimens can be characterized by a fractal dimension spectrum with positive skewness whereas the surface structure of outburst-nonhazardous coal specimens is characterized by a symmetrical fractal spectrum (with negative skewness close to zero).

RESEARCH PROCEDURE

In our research procedure, we investigated a relationship between the fractal dimension spectrum characterizing the coal surface structure and coal susceptibility to gas-dynamic events. For this purpose, we performed a multifractal analysis of SEM images for about 400 test coal specimens from outburst-hazardous and outburst-nonhazardous beds in different coal fields of Kuzbass (S.M. Kirov's Mine, Uskovskaya Mine, Osinnikovskaya Mine), Donbass (Chaikino Mine, Skochinsky Mine, Zapadnaya Mine, Pervomaiskaya Mine, Kommunist Mine, Mine of 50th Anniversary of October Revolution) and Komi Republic (Zapolyarnaya Mine). These SEM images were taken from IPKON bank of data, which contains hundreds of images of coal specimen surfaces. Figure 1 demonstrates SEM images of four analyzed coal specimens which belong to outburst-hazardous and outburst-nonhazardous beds.

In order to perform a multifractal analysis, we used SEM images of coal specimens increased by 1,000 times because such scale provides information about coal grains having size from 0.5 to a few microns. The SEM images obtained with a magnification more than 1,000 times allow selecting finer irregularities but they are useless for analysis of gas release from coals.

At the preliminary stage of the multifractal analysis, we suppressed a noise of the analyzed SEM images which are characterized by a high-level noise from a number of sources (Nesterenko, 2011). To suppress a noise of the SEM images we used the Chebyshev filter that is proposed in (Uchaev, Uchaev & Malinnikov, 2015) and can be applied for denoising of coal SEM images (Malinnikova, Malinnikov, Uchaev & Uchaev, 2016). In this study, for suppression of a SEM image noise, it was assigned the following values for Chebyshev filter parameters: the sharpness factor was equal to 0.6 in order to ensure an acceptable level of the noise, the brightness factor was assumed as 1.0, and the contrast ratio was equal to 1.0 for the initially contrast images of specimens and 1.4 for the weakly contrast images of specimens.
Multifractal analysis of scanned coal surface images was performed using program GEO-PC developed at MIIGAIK. In order to obtain fractal dimension spectra for all analyzed SEM images, we used the technique of generalized local-global multifractal analysis, which was introduced in (Malinnikov, Uchaev & Uchaev, 2010). The main advantage of this technique is that the resulting multifractal spectra do not depend on an image subdivision into cells. Spectra $f(\alpha)$ obtained by multifractal analysis of SEM images from Figure 1 are shown in Figure 2.

As a result of the multifractal analysis, for all analyzed coal specimen SEM images, it was computed values of the asymmetry coefficient $R$ for $f(\alpha)$-spectra characterizing a surface structure of the coal specimens. The values of $R$ were calculated by the following formula (Xie & Bao, 2004):

$$R = \frac{\Delta_l \alpha - \Delta_r \alpha}{\Delta_l \alpha + \Delta_r \alpha},$$

(1)

where $\Delta_l \alpha$ and $\Delta_r \alpha$ are widths of left and right branches of a $f(\alpha)$-spectrum. The coefficient $R$ (Equation 1) takes values from the range $[-1, 1]$ and characterizes a direction and degree of skewness of $f(\alpha)$. If $R > 0$ then the spectrum $f(\alpha)$ is skewed to the left, and when $R < 0$, the spectrum $f(\alpha)$ is skewed to the right (the spectrum $f(\alpha)$ is a symmetric if $R$ is close to zero).
At the final stage of our research procedure, we checked the conjecture from (Malinnikova et al., 2014), that the asymmetry coefficient $R$ can be used to distinguish outburst-hazardous and outburst-nonhazardous coal specimens: if $R < 0$ then we can assume with a high degree of probability that the coal specimen belongs to an outburst-hazardous bed and, conversely, if $R > 0$ then we can assume that the analyzed specimen belongs to an outburst-nonhazardous bed.

**RESULTS**

In our research procedure, we obtained the following results. After calculation of asymmetry coefficient values for all analyzed test coal specimens, we found that the surface structure of outburst-hazardous coal specimens can be characterized by a fractal dimension spectrum with positive skewness whereas the surface structure of outburst-nonhazardous coal specimens is characterized by a symmetrical fractal spectrum (with negative skewness close to zero). This result is verified for nearly 80% of all test coal specimens. Figure 2 also demonstrates that fractal dimension spectra for outburst-hazardous and outburst-nonhazardous coal specimens from Figure 1 have different degrees of skewness. Thus, we can assume that the asymmetry of fractal dimension spectra characterizing coal surface structure can be used as a criterion for verification of coal susceptibility to gas-dynamic events.

In this study, we also have determined two ranges of the most likely values of $R$ for outburst-hazardous and outburst-nonhazardous coals (Figure 3). To find each of these ranges, the following algorithm was used. At the first step of the algorithm, the range $[-1, 1]$ of all possible values for $R$ is divided into intersecting intervals of width $w = 0.001m$, taken with step $h = 0.001n$. After that, the number of test values for $R$ that fall in each of these intervals is counted and it is chosen that interval, which contains the most test estimates for $R$. If any of the intervals contains 60% of all test values for $R$, then this interval is chosen as the desired range of the most likely values for $R$. Otherwise, if we haven't intervals contained 60% of all test values for $R$, then the value of $m$ (or $n$) is increased by one and we go to the next step of the algorithm. At each step of the algorithm, the above-described sequence of actions is repeated.

The centers of found ranges of the most likely values for $R$ are following (Figure 3): $-0.15$ is the center of the range displayed by red dotted line and $0.05$ is the center of the range displayed by blue dotted line. As can be seen from Figure 3, the width of each range does not exceed 0.3. We can also note that the obtained ranges of the most likely values for $R$ have a small intersection area including 0. This means that asymmetry coefficient values that close to zero are not provide reliable estimates of coal susceptibility to gas-dynamic events.
CONCLUSIONS

The statistical analysis of a large volume of test coal specimens from outburst-hazardous and outburst-nonhazardous beds detected the relationship between the coefficient of asymmetry of the fractal spectrum characterizing the surface structure of coal specimens and coal susceptibility to gas-dynamic events. According to the results of the statistical analysis, it was found that, in 60% of cases, the asymmetry coefficient for coal specimens from outburst-hazardous and outburst-nonhazardous beds takes values from the ranges demonstrated in Figure 3. The obtained results for test coal specimens can be a confirmation of our conjecture that the asymmetry coefficient can be used to distinguish outburst-hazardous and outburst-nonhazardous coal specimens. In future, the found ranges of the most likely values for the asymmetry coefficient can be used to develop a technique for estimating the coal susceptibility to gas-dynamic events by means of a structural factor.

In conclusion of this study, we would also like to note the following: estimates of coal susceptibility to gas-dynamic events obtained by a multifractal analysis of SEM images of coal specimens are preliminary. These estimates provide information on coal susceptibility to sudden coal and gas outbursts, depending on the corresponding rock and gas pressures to be taken into account.

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INVESTIGATION AND MAINTENANCE OF UNDERGROUND MINING DEVELOPMENTS IN SEISMIC-TECTONIC ACTIVE AREAS OF THE CENTRAL ASIA

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ABSTRACT

In this article we consider the results of studying the conditions for the construction and maintenance of underground mine workings in regions characterized by seismic and tectonic activity. Output of mineral deposits in the Central Asia is carried out in the difficult conditions characterized by high seismic and tectonic activity of region, considerable geological infringement deposits, danger of rockbursts. Long-term practice of mining works shows, what difficulties create tectonic fields of tension at working out of a mineral deposits. This is due to ensuring the safety of mining works, maintenance and stability of mining workings, installation and reconfiguration of complex mining equipment in the presence of disturbance zones and some other problems.

Certain interest represents studying of maintenance conditions of underground mining developments on mines of Uzbekistan. Feature of region is not only display of dynamic seismic processes. As it is known, the period between earthquakes is characterized by accumulation of elastic energy of rock-mass deformations, that is growth of operating tensions.

Supervision over a condition of the underground mining developments which were under the influence of tectonic tensions were carried out for many years. The study of underground workings has shown that their condition sometimes continues to remain unsatisfactory, despite frequent repairs. Attention is drawn to the amount of costs for maintenance of underground workings, when some sites were repaired on average for 1-1,5 years, although there are areas where the repair of the support was made 2-3 times during the year.

Of interest are conditions where the magnitude of the acting tensions in the rock massif is comparable to the strength of the rocks. In addition, when performing blasting operations, technological disruptions arise, which reduce the strength of the array.

In cases where the acting tensions around the underground mining working slightly exceeds the long-term strength of the rock massif, a zone of inelastic deformations is formed. Round underground development the zone intensive fracture is formed, rocks in which at a relaxation of pressure increase in volume that causes additional displacement of a contour. A zone of intense cracking is formed around the underground development, and rocks that increase in volume during stress relaxation lead to an additional displacement of the outline. A zone of intense cracking is formed around the underground development, and rocks that increase in volume during stress relaxation lead to an additional displacement of the contour.

Analogously to the problems of estimating the stress state of a rocks around the gallery, we solved the task of calculating the stress-strain state of rocks during the production of blasting operations at the construction of a vertical shaft. Quality of blasting operations will also prevent negative impact on the surrounding massif without causing unnecessary cracking and deformation, including the avoidance of dynamic (induced) manifestations of rock pressure in the tectonically tense zone of the rock massif.

Modern mathematical methods and technical computing means allow obtaining qualitatively new and multivariate solutions of problems with subsequent evaluation of the degree of their identity to real processes. The authors developed computer application programs for the quantitative evaluation of the strength of underground workings, taking into account the features of their design, piecewise inhomogeneous physical and mechanical characteristics of the surrounding rock mass for the accepted calculation schemes and numerical models.

KEY WORDS

underground mining, seismotectonic activity, rock massif, earthquake, mine support, rockbursts

INTRODUCTION

In the world practice of mining there is a steady tendency of a natural increase in the depth of mining of mineral deposits. Along with the desire to increase the volume of mining, strict requirements are put forward to improve the working conditions of miners, to ensure the safety of work, the use of new more efficient technologies (Gattinoni, Pizzarotti, & Scesi, 2014; Melikulov, 1993).

The countries of Central Asia are rich in minerals, they traditionally develop the extraction of gold, uranium, oil, gas, copper, coal and a number of other types of mineral raw materials. More than 3,000 deposits of about 100 kinds of minerals have been identified in Uzbekistan, more than 1,400 deposits containing more than 70 types of mineral raw materials, including more than 50 noble deposits, more than 40 deposits of non-ferrous, rare and radioactive metals, etc., have been discovered (Abdykaparov, Imaraliyev, & Mambetov, 2007; Aytmatov, & Kozhogulov, 1988; Golovanov, 2001; Mambetov, Sh., Abdiyev, & Mambetov, A., 2013; Sanakulov, 2010; Ulomov, 1975).

With the development of deposits with favorable geological conditions, as well as lying at shallow depths, the depth of field development increases, mining and geological conditions become more complicated, rock pressure and water cut of the massif increase, which requires additional costs for maintaining underground workings during

A special feature of the Central Asian region is the complex geodynamic situation and seismotectonic activity, which is accompanied by the action of additional stress fields in the surrounding massif, which in magnitude exceed the traditionally taken calculated stresses from the gravitational field. (Uломov, V. I. (1975). Many deposits of minerals in this region are located in the territory of sites, the seismicity of which is estimated up to 7 - 9 points. In some earthquakes, the fields fell into a zone with the maximum intensity of seismic vibrations, which led to disturbances in the stable operation of mines, deformations and destruction of the support of mine workings, and the disabling of stationary equipment (lifting and drainage installations, main ventilation fans). To eliminate the consequences, additional funds are spent, which undoubtedly affects the cost of production, and a significant part of the damage is accounted for the restoration and repair of underground workings (Melikulov, 1993; Pletnev, Rakhimov, Tadzhibayev, Melikulov, 1983).

As is known, the period between earthquakes is characterized by the accumulation of the elastic energy of deformation of the rock mass, that is, the growth of the acting stresses. Obviously, the field of tectonic stresses, the magnitude of which varies relatively slowly (as a quasi-static process), also has a fairly tangible effect on the state of mine workings.

Particular attention from the point of view of underground mining technologies attracts stress fields that are formed under the influence of tectonic forces as a result of uneven movements and deformations of the earth's crust. Such stresses in an array of rocks caused by tectonic processes, according to experts in various mining regions of the world, can exceed 10 to 20 times the tension from the total weight of the rock column. Under the action of these forces acting in the subhorizontal plane, qualitatively new conditions arise both in studying the natural stress field and in calculating the steady state of the rocks around the workings (Kearey, Klepeis, & Vine, 2009; Khloptsov, & Baklashov, 2004; Markov, 1977).

RELATIONSHIP OF SEISMOTECTONIC PROCESSES AND FIELDS OF STRESSES

One of the characteristic features of mineral deposits in the countries of Central Asia is their location in the highlands or even in high mountains. For example, the polymetallic deposit Khandiza (Uzbekistan) is located in the spurs of the Tien Shan at the level of 1300 - 1800 m above sea level; coal deposit Shargun - at a mark over 1550 m; mercurial deposit of Khaidarkan (Kyrgyzstan) - at the level of 1500 - 2350 m. The well-known railway tunnel Angren-Pap, which is the longest in Central Asia, was built in the mountains at 1320-1420 meters, the elevation points on the surface of this mountain pass reach a mark of 2,845-3,476 meters above sea level (Aytmatov, & Kozhogulov, 1988; Golovanov, 2001; Kucherskiy, 2007; Mambetov, Sh., Abdiyev, & Mambetov, A., 2013; Markov, 1977; Pletnev, Rakhimov, Tadzhibayev, Melikulov, 1983; Zhou, Aripov, Guo, Zang, & Melikulov, 2016).

It is easy to interpret the state of the array, when in the coving zone of anticlines the effective stresses are less than in synclines. However, in some cases, on the contrary, fold locks are compression points. The same is also observed in fault zones: in some cases, the zones of decomposition do not exert a noticeable effect on the differentiation of stress fields, while in others the stress state in the zones of decomposition increases sharply. It is especially noted that the tectonic forces on the scales of long geological time vary in both magnitude and direction. (Gzovskiy, 1975; Hunt, 2007; Kurlenya, Seryakov, & Eremenko, 2005; Markov, 1977).

In areas covered by active new tectonic movements, the change in tectonic forces over time is of interest for the problems of rock mechanics. For example, in the Carpathians, the Caucasus and the Kurile-Kamchatka region (Russia), according to the records of strong earthquakes in 6 to 12 years, a qualitative change in the directions of the axes of stresses (Turchaninov, Iofis, & Kasparyan, 1989).

Estimation of the constant "feeding" of tectonic stresses due to modern movements and deformations of the earth's crust is based on the analysis of the following experimental data. According to the results of geodetic survey and deformometers, the rate of accumulation of tectonic deformations of $10^{-7}$ year is typical for aseismic zones, and $10^{-5}$/year for high seismic zones. The magnitude of the deformation change in the case of a catastrophic earthquake was obtained on the basis of a survey near the epicenter. Deformations - forerunners of the shock in catastrophic earthquakes have a magnitude of the order of $10^{-2}$. Deformations from tidal effects in the earth's crust are estimated at $10^{-9}$ (Markov, 1977; Ohnaka, 2013).

MONITORING OF THE CONDITIONS OF UNDERGROUND MINE AND MINE WORKS

The results of long-term observations of the state of mine workings located in the zone of influence of elevated tectonic stresses are known. Survey of preparatory workings of a number of coal mines in Central Asia showed that the conditions for maintaining them are complex, despite frequent repairs. Attention is drawn to the amount of work to maintain the workings, when individual sites on average were repaired after 1-1.5 years, although there are areas on which repair mine supports were made even 2-3 times within one year. The state of the excavations fixed by a metal arch support was studied. This support for structural features should work reliably enough in a compliant mode, i.e. to provide the specified design strains without destruction. In fact, in the zone of tectonic stresses, the mounting frame is compressed by lateral loads, and the development is narrowed as a result of this action, the nodes of compliance are no longer functioning in the supporting construction. Often, the destruction
or loss of load-bearing capacity of the support is preceded by the deviation of the fixing frame from the given design position, i.e. loss of stability. Moreover, the reason for the loss of stability of the fastening frame can be different: the unevenness of the load along the contour due to the anisotropy of the properties of the rock massif, the uneven distribution of the load from the rock pressure to the neighboring fixing frames, the spread of the rigidity of the fastening frames, the loads acting on the frame outside the plane of its location, dynamic loads, etc. It is only natural that the loss of stability of the mounting frame leads to a reduction in its load-carrying capacity, as a result of which such workings are deformed over time and need to be repaired (Melikulov, 1993; Melikulov, & Dzhangiyev, 2004; Pletnev, Rakhimov, Tadhribayev, Melikulov, 1983).

Observations have shown that the magnitude of tectonic stresses can change noticeably when passing from the mine to the mine. Even within the same mine and on the same horizon (for example, in the 600 m of the Rasvumchorsky mine, Russia) tectonic stresses change more than twice in magnitude. Thus, against the background of the general pattern of excess of horizontal tectonic stresses over gravitational verticals, a statistically significant change in the tectonic stresses at different sites is observed (Malovichko, A., & Malovichko, D., 2013; Markov, 1977; Petuchov, Ilin, Trubetskoy, 1997; Sashurin, 1999).

Such factors are the reason that some of the coal mines (Shurab in Tajikistan, Sulukta and Kizilkiya in Kyrgyzstan, et al.), signs of rockbursts have a depth of about 200 m development. The case was observed in the construction of an inclined mine for the transportation of coal by a conveyor on Angrensksy coal mine (Uzbekistan), when the deformation of the array curved the axis of production and made it impossible to equip it with a conveyor.

The unfavorable manifestation of tectonic processes together with the stress concentration in the massif (increased rock pressure), which resulted in the formation of a wide zone of disturbed rocks in the Kayragach gold mine (Uzbekistan), led to difficulties in the construction stage of the gallery, and subsequent frequent repairs of the site forced to abandon its further exploitation and build a new gallery around this tectonically unfavorable zone.

In connection with the technological peculiarities of the repair conditions for the damaged support or the working site in underground cramped and dangerous conditions of the threat of continuing collapses, the cost of repairing each meter of the emergency site of underground mining may even exceed the cost of constructing a meter of new gallery.

ANALYSIS OF THE INFORMATION BASIS ON THE SEISMIC ACTIVITY OF THE REGION

In order to quantify the impact of seismotectonic processes, extensive studies are carried out at geodynamic polygons. Experts based on long-term observations compiled a map of young and modern geological movements of the territory of Uzbekistan. At the Central Kyzylkum geodynamic range, the vertical displacement rate is from (-4) to (+2) mm per year, in the zone of tectonic faults the displacement module reaches up to 18-22 mm per year. In the narrower part of this region in the zone of Marjanbulak earthquake on May 26, 2013 (Uzbekistan, there are several gold mines here), the displacement speed in the flat zone is up to 10 mm per year, in the foothills - up to 20-30 mm per year, in the mountains - up to 50 mm per year; as a result of this earthquake, the magnitude of stress relief is estimated at 12 MPa (Khamidov, & Shukurov, 2015).

In modern geomechanics methods for assessing the influence of acting factors through the probabilities of events (D’Amico, 2012; Glaser, 2016). Especially relevant is the probabilistic approach to the evaluation of random events scattered in space and time, such as earthquakes. In addition, earthquakes vary greatly in intensity, depth of manifestation, distance from the epicenter to the object, engineering-geological conditions of the environment, etc. (Gasanova, Salyamova, & Melikulov, 2018; Hencher, 2015; Ismail-Zadeh, & Tackley, 2010; Kuzuyayev, & Pugach, 2013).

For the probabilistic estimation of the effect of seismic phenomena, statistical studies of the field of events were carried out, which took the region of Central Asia and Kazakhstan, located within the parallels of 35-45 degrees North latitude and meridians 63-82 degrees East longitude. For ease of use, information on earthquakes registered in this region with a magnitude $M \geq 2.8$ based on the annual earthquake bulletins is registered as an electronic database.

The collected database covers on average more than 2800 events (earthquakes) occurring annually within this region, each of which is represented by 7 main features: date, time, geographic coordinates, depth of focus, accuracy class of the epicenter coordinates, energy class (Gasanova, 2017).

TECHNOGENIC-TECTONIC OR INDUCED SEISMICITY IN MINING OPERATIONS

The induced seismicity in the mines takes place both in the areas of active modern mountainengineering movements, and in the areas of old folded systems, and on platforms and shields. The average depth of the work at which seismic phenomena occur is about 600 m in areas of modern orogenesis in Central Asia (Aytmatov, & Kozhogulov, 1988).

Dangerous are mining and mining-tectonic rockbursts, which in the mining industry lead to major accidents, significant material damage and human casualties. The damage from one mining rockburst reaches 100 million rubles, restoration work in a number of cases lasts for months (Petuchov, Ilin, Trubetskoy, 1997).

More than half of the Russian mines are experiencing seismic phenomena, the practical manifestation of which is associated with mining operations. Severouralskiy bauxite mines, Tashtagolsky mine, Kuznetsk coal basin
mines, Khibiny, and Talnakh mines in Russia, the system of Witswatersrand mines, South Africa, Palabora, Kolar gold deposit, India, Solvay mine, USA (Adushkin, & Turuntayev, 2015; Glaser, 2016).

As the strongest (in Russia) seismic events induced by mining operations were noted: Bachatsky earthquake on June 18, 2013 ($M = 6.1$) in Kuznetsk coal basin (epicenter on the territory of the Bachatsky open pit); mining -tectonic rockburst on the Kirov mine of OJSC "Apatite", which occurred on April 16, 1989, with an energy of $-10^{12} \text{J} (M = 4.8-5.0)$; a rockbursts at the mine Kurbazakskaya of the Yuzhnouralsk bauxite mine, which occurred on May 28, 1990, with an energy of $10^{10} - 10^{11} \text{J} (M = 3.5-4.0)$; a powerful mining and tectonic impact at the Umbozero mine of the Lovozero rare earth metals deposit, which took place on August 17, 1999, with an energy of $10^{11} - 10^{12} \text{J}$, equivalent to the initiated earthquake with a magnitude of 4.4-4.4 and destroyed more than $6.0 \times 10^5 \text{m}^2$ of mine workings.

On the basis of the analysis of manifestations of technogenic seismicity, specialists prioritized the maximum magnitude $M_{\text{max}}$ of the technogenic earthquakes that arose in the world. According to this criterion, the marked catastrophic seismic events in different industries are distributed as follows: A) development of oil and gas fields (Gazli, Uzbekistan, $M_{\text{max}} = 7.3$); B) filling of reservoirs (Koina River, India, $M_{\text{max}} = 6.5$); C) underground mining operations (Kuznetsk coal basin, Russia, $M_{\text{max}} = 6.1$); D) underground nuclear explosions (polygon in Nevada, USA, $M_{\text{max}} = 4.5$). (Adushkin, & Turuntayev, 2015).

**COMPUTATIONAL EXPERIMENT: MODELING OF PROCESSES OF SUSTAINABILITY OF UNDERGROUND MINE WORKINGS**

Modern mathematical methods and technical computing means allow obtaining qualitatively new and multivariate solutions of problems with subsequent evaluation of the degree of their identity to real processes. The authors developed computer application programs for the quantitative evaluation of the strength of underground workings, taking into account the features of their design, piecewise inhomogeneous physical and mechanical characteristics of the surrounding rock mass for the accepted calculation schemes and numerical models (Gasanova, Salyamova, & Melikulov, 2015; Fadeyev, 1987; Jaeger, 2009; Salyamova, & Melikulov, 2007).

A flat task in the study of the stress-strain state of a rock mass around underground mine workings.

For the numerical solution of the problem, the infinite region surrounding the underground mine is replaced by a finite region, at the boundaries of which the corresponding boundary conditions are put or loads are applied (Salyamova, & Melikulov, 2010, 2011; Sultanov, Salyamova, Khusanov, & Melikulov, 2003).

Figure 1 shows stress plots in an rock mass around the mine, constructed on the basis of calculations using this technique for the case when the lateral (tectonic) pressure coefficient is 1.38.

Figure 1 - Stress diagrams in an rock mass around the underground gallery: a) isolines of horizontal stresses; b) isolines of vertical stresses; c) isolines of tangential stresses.

**Solution of the problem using the COSMOS/Works software complex with the example of the given mining and geological conditions.**

COSMOS/Works is a finite element analysis system integrated into 3D SolidWorks. The combination of design systems and finite element analysis has made it possible to obtain a tool for calculating and optimizing designs consisting of a large number of elements (Alyamovskiyy, 2004). In the rock massif with the accepted average physical-mechanical properties (modulus of elasticity, Poisson’s ratio, volume weight, angle of internal friction,
coefficient of adhesion), a mining with design parameters is constructed at a given depth. Based on the results of calculations of the state of the array, the acting stresses are set (Melikulov, Salyamova, & Kaygarodov, 2009).

In assessing the stability of rocks, the limiting state is described by the well-known Coulomb law. In the practice of designing the construction of underground structures to assess the permissible level of stress or strength, the safety factor is widely used, defined as the ratio of the tangential stress acting at the considered point to the magnitude of the limiting tangential stress. As an example, we determine the stresses $\sigma_x$, $\sigma_y$, $\tau_{xy}$ at various arbitrarily taken points of the array around the underground gallery, as shown in Fig. 2. Here, for example, 3 rows of 12 points in each row are taken. The first row (points 1 - 12) passes along the left contour of the axis of symmetry of the development, the second row of points (13-24) is displaced from the first into the depth of the massif by a distance of 0.6 m. Similarly, the third row of points (25 - 36) is located at a distance of 0.6 m from the second row into the depth of the array.

Figure 3 shows a graph of the change in the values of the coefficient of stability in each of the 36 points.

In the graph of the change in the coefficient of stability (Fig. 3), the points having coefficient values less than 1.0 are potentially unstable points, i.e. the rocks in this part of the massif is prone to destruction. The graph (Fig. 5) of the change in the stability coefficient of the same points of an artificially fortified rocks shows that there are no points with a coefficient of stability below 1.0.

Finite-element computational modeling of blasting operations in a vertical shaft

Analogously to the problems of estimating the stress state of a rocks around the gallery, we solved the task of calculating the stress-strain state of rocks during the production of blasting operations at the construction of a vertical shaft. The condition of this problem is to study the stress state of the array with successive short-delayed blasting of several groups of charges in order to ensure the quality destruction of rocks by an explosion within the contour with minimal damage to the rocks behind the design contour.

In solving this problem, the following basic parameters of objects and processes are taken into account in the developed model: the physical and mechanical properties of rocks, the depth of the holes and the scheme of their location in the face, the design of charges, the length of the charge in each hole, the energy of blasting, the rate of detonation in the charge, the speed of propagation of elastic waves and the process of crack formation in the array, the parameters of intrahole retarders, the wave interaction of simultaneously exploded charges in each group of the explosion series.
Figures 6 and 7 show horizontal and vertical displacements of points as a result of successive explosions of the central cut hole (single charge), then in a group of cut holes that are disposed around a 1.5 m diameter circle.

The results of solving this problem make it possible, on the basis of the wave interaction of simultaneously exploded charges in each group of the exploded series, to select the optimal range of the following parameters for blasting operations: the mass and number of charges in each series of deceleration, the sequence and the delay of the
explosion of charge explosions both within each series, and between series of explosive charges (Davison, 2008). The control of the action of explosive charge during the construction of the mine makes it possible to achieve a qualitative and uniform fragmentation of the blasted rock mass in the shaft. At the same time, the maximum integrity of the rock massif behind the design outline of mine workings is ensured, which is one of the measures to preserve the stability of the rock massif and the constructing underground gallery (Polukhin, Kaloyerov, Gryadushchyi, & Goryanskaya, 2002). This quality of blasting operations will also prevent negative impact on the surrounding massif without causing unnecessary cracking and deformation, including the avoidance of dynamic (induced) manifestations of rock pressure in the tectonically tense zone of the rock massif. It should be noted that such broad opportunities to manage the process and the result of blasting when constructing underground mine workings arise using modern means of non-electric initiation of charges of explosives.

**CONCLUSIONS**

The permanently movement of our planet and the natural processes taking place in the Earth's crust are the cause of the emergence of vast areas of seismic activity, where thousands of earthquakes of various energy occur every year, which complicate the conditions for the development of mineral deposits. In addition, with the depletion of reserves of deposits with favorable geological conditions, mining works continue to develop for more complex natural conditions.

The Central Asian region and many areas of modern orogenic processes are characterized by the manifestation of seismotectonic processes, the results of which, on the one hand, complicate engineering conditions of work, and on the other become a powerful engine for the further development of mining technologies. As a result of anthropogenic activity, in the solution of global regional problems, there are cases of creating additional sources of danger leading to the stimulation of technogenic-tectonic or induced seismicity.

The human need to solve the problems that have arisen, has led on the basis of study of Earth's physics and engineering mathematics to the intensive development of modern methods of geomechanics, tectonics, geophysics, and geodynamics.

Numerical methods of modeling mining-geological objects and processes of any complexity with the use of modern technical and software tools offer new opportunities in solving technological problems, especially in areas where physical experiments in the traditional view are difficult - in underground mining facilities, processes of destruction and mining of minerals using explosive technologies.

Modern investigations in the discussed field of science and mining production gives an instrument for preliminary substantiated evaluation of technical solutions, the forecast of possible adverse consequences, the threat of catastrophic events, the safety of objects and people, and material and economic losses. The original development of the methodology for calculations with the use of models allows us to solve the modern tasks in a new way using computer programs developed, as well as to offer technological measures for managing the state of the array in the production of blasting operations in underground workings, taking into account the safety and prevention of the dynamic manifestations of the mountain pressure.

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DEVELOPMENT OF GEOMECHANICAL RESEARCH FOR SOLVING PROBLEMS OF WATER RESOURCES PROTECTION IN THE DEVELOPMENT OF SUBSOIL RESOURCES

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ABSTRACT

A very important issue in the development of the Earth's interior is the solution of the problem of the safe state of water resources that fall into the zone of technogenic impact. To this end, it is proposed to conduct a comprehensive investigation in three main areas: the forecast of deformation processes based on the basic provisions of practical geomechanics; mathematical (computer) modeling; provision of geomonitoring. In the complex, this will provide the most reliable information about the dynamics of the change in the massif in which the mineral is mined. This is an effective way that will ensure the safety of water bodies falling into the mining zone.

KEYWORDS

Geomechanics, conservation of Earth's interior, interdisciplinary studies, deformation processes, downhole geotechnology, mining safety, environmental load.

The problem of subsurface conservation in the solution of geomechanical problems forms new scientific directions - identification of environmentally dangerous technogenic geo-processes in the upper part of the lithosphere; development of methods for monitoring and managing these processes; creation of geomechanical bases of resource-saving geotechnologies and nature protection measures; substantiation of the relationship between geomechanics and other fundamental and applied scientific disciplines in preserving the environment and the earth's interior.

Studies of the ecological orientation are developed in the IPKON RAS in three directions. The first is to develop engineering methods that allow predicting deformation processes with dangerous environmental consequences and, managing them [1]. These methods are based on the basic provisions of practical geomechanics, analysis of relevant statistics and common sense. With the help of these methods in many cases, it is possible to quickly obtain a preliminary result and justify the geomechanical principles of resource-saving and ecologically balanced geotechnology for the development of mineral deposits. The second direction provides for computer (mathematical) modeling of natural and technogenic processes, which allows studying non-linear interrelationships in natural and technical geosystems within the framework of interdisciplinary research and assessing possible scenarios for reducing harmful environmental consequences of anthropogenic impact on the geo-environment [2]. The third direction includes the system of geomonitoring of technogenic mining impact on the geo-environment [3]. This system is based on the instrumental control of deformation processes taking into account the apparatus data of geophysical, geochemical, hydrogeological and other studies, and also on the analysis of the dynamics of the change in the current state of the processes under study.

We have examined some of the results of research related to downhole hydrodevelopment.

The introduction of borehole drilling is promising in the development of high ash and watered coal seams. The IPKON RAS has scientifically substantiated and developed a method of downhole mining of minerals, which entitles to reduce the environmental load on the environment by means of underground preparation of the slurry and the abandonment of gangue in the bowels. Another area of possible use of resource-saving borehole geotechnology is the development of coal seams prone to gas-dynamic phenomena. The use of natural gas and geodynamic energy of the rock mass, which is a negative factor in underground geotechnology, for the destruction of minerals, increases the efficiency of developing reserves of mineral raw materials [4].

One of the peculiarities of well drilling with regard to environmental consequences is the development of undesirable deformation and hydrogeological processes in the overlying rock strata, which can be of great importance in mining KMA rich iron ores. Here, according to the technology of borehole hydraulics, it is planned to test limited ore reserves and incomplete removal of pressure in an array of loose ores in order to maximize the floating properties of the watered ore mass for its shift to production wells. As it is supposed, maintaining a certain pressure of water in the ore deposit should not contribute to the development of dangerous technogenic deformation processes in the overlying stratum. However, in the development of highly water-rich rich iron ore, the hydrogeological system of groundwater may be disrupted, which may lead to the loss of sources of drinking water supply to the population of the region [5].

One of the consequences of deformation processes is the formation of main channels for the flow of groundwater in the overburden strata of rocks and mixing of drinking water in the upper horizons and highly mineralized ore water. To determine how the scenarios of geo-processes in the overlying strata will develop, a computer model was developed and modeling of the interaction of geomechanical, geodynamic and hydrogeological processes during the extraction of rich iron ores for the conditions of the Shemraevsky section of the KMA was carried out.

In this model, the concept of natural-technogenic hydronrupture of the rock mass [6,8,9] was used to analyze the conditions of the contact between rocks at the boundary of the aquifer. It is shown that under certain conditions on the boundary of the aquifer, the contact can be stratified and filled with water under hydrostatic pressure. The resulting fissure of natural-technogenic frac can grow with the interaction of two force factors: the pressure of natural water penetrating the crack and the acting stress near the crack.
In the study the stressed-deformed state of the ore deposit at a depth of 500 m and the overlying strata of rocks represented by sedimentary rocks, clayey water bodies, and aquifers were considered. Immediately in the roof of the deposit of loose watered iron ores there is a limestone with a thickness of about 75 m. When some part of the ore is mined (up to 20%), the remaining part of it is loosened, the partially developed space is filled with ore mineralized water that exerts hydrostatic pressure on limestone equal to 5 MPa. The strained state of clay waterproofing and limestone is formed under the influence of the weight of overlapping rocks, horizontal lateral compression, hydrostatic pressure of ore water. The procedure for modeling the stress-strain state and the process of crack formation is described in [6]. Figure 1 shows some results of calculating the stress field in the overburden of rocks during the passage of limestone limestone 500 m.

![Figure 1 - Distribution of the least compressive stresses in the overlying stratum:](image)

The modeling also showed that the active action of water in growing cracks is an important factor: the extent, orientation and scale of the effect of these cracks prove to be substantially different than in the case of the model of "dry" cracks growing only under the influence of rock pressure.

Based on the results of the simulation, a number of conclusions can be drawn regarding the conservation of drinking water horizons in borehole drilling. In particular, if the span of limestone underworking does not exceed 300 m, the main cracks on the boundary of the Callovian-Bathian aquifer and the water retainer are not formed.

In addition, the modeling showed that the picture of the interaction of technogenic geoprocesses strongly depends on geodynamic conditions, namely, on the values of natural stresses in the overlying rock strata. Unfortunately, with respect to the magnitudes of these stresses, conflicting data are given in the literature for some of the regions under consideration. This complicates the work and reduces the accuracy of the results obtained.

Thus, the development of geomechanical studies, which relate to ecological orientation, shows that innovative resource-saving well geotechnologies can be grounded and developed on their basis. They will allow the development of reserves of mineral deposits in difficult mining and geological conditions. The undoubted advantage of downhole geotechnologies is the provision of safe mining through the withdrawal of people from clearing areas and reducing the environmental burden on the environment.

It should also be taken into account that with well drilling in the rock strata, situations with dangerous ecological consequences can be formed. Therefore, it is necessary to use the advanced capabilities of computer modeling of natural-technogenic geo-processes. The results of the simulation will be possible to develop requirements for the organization of the geomonitoring system and to establish the permissible limits of the occurring changes.

References:

GEOTECHNICAL RISKS AND RELATED DANGERS IN THE DEEP MINES OF KAZAKHSTAN

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ABSTRACT

During the development of the deposits using underground methods, there arise geotechnical dangers of regional and local scale (of natural and technogenic character), which are accompanied by the risks involved. As the result there arise questions of the safe operations in complex mining conditions. The authors conducted the assessment of the dangerous level of the arising geotechnical risks, developed the classification by the reasons of occurrence and by scale, developed the system of risk management at mining plant and the necessary measures for its prevention.

KEY WORDS

Underground mining, geotechnical risks, mining and geological conditions, safety of mining operations.

TERMS AND DEFINITIONS

Rock pressure is the forces that arise in the rock massif surrounding the mining.

Mining support is an artificial structure built in the mining to prevent the caving of surrounding rocks and to maintain the necessary cross sectional areas of the mining.

The fractured zone on the earth’s surface is part of the shift trough, where the earth’s surface shifted with the formation of visible cracks.

The zone of dangerous influence of underground mining is part of the trough of earth's surface shift, where deformations that are dangerous for buildings, structures and natural objects appear.

Control is one of the main functions of a control system, which is performed on the basis of monitoring the behavior of a managed system in order to ensure optimal functioning.

Protection measures are the mining or building (constructive) activities aimed to reduce the harmful impact of underground mining on buildings, structures and natural objects in order to avoid violations of normal operation of objects that can lead to accidents.

Monitoring is a continuous process of supervision and notification of object parameters, in comparison with the specified criteria.

Shift trough is a part of the earth's surface, which has undergone a shift owing to the underground mining.

Danger is the situation in nature or technosphere, wherein adverse processes or effects may arise.

Caving of rocks is the random movement of rocks with the loss of natural structure owing to the underground mining.

Risk is a combination of the probability of occurrence and the consequences of adverse events.

Rock-bump hazard is the presence of rocks and stress levels prone to rock bumps, which may cause brittle fracture of these rocks during underground mining.

Rock pressure control is a set of activities aimed to regulate the manifestations of rock pressure.

INTRODUCTION

Most of the deposits of polymetallic ores of Eastern Kazakhstan are developing using filling system of the worked-out area and self-propelled equipment. During the development of the deposits using underground methods, there arise geotechnical dangers of regional and local scale (of natural and technogenic character), which are accompanied by the risks involved. As the result there arise questions of the safe operations in complex mining conditions. The analysis of the possible consequences of the rock massif shift and high rock pressure made it possible to classify the dangers and risks by the scale of its manifestation.

The regional dangers that can lead to a shutdown of the mine and human losses include:

- large-scale rock caving;
- high-power rock bumps;
- formation of cave-in near the mine site;
- large deformations and destruction of buildings and structures within the limits of the mining allotment;
- endogenous underground fires;
- breakout of mine or surface waters.

Local geotechnical dangers include events that may lead to lay-off of the operation of a particular section of the mine, breakdown in the mining technological process, and possible injury of miners:

- dangerous deformations of buildings and structures on the surface and underground;
- destruction of mining and their support;
- dynamic manifestations of rock pressure;
• destruction of pillars of various purpose;
• caving of the roof or the chamber sides.

Underground mines developing polymetallic deposits of East Kazakhstan can be divided into two groups by existing geotechnical risks and related problems of safe mining. The issues of preserving of permanent and opening mining in a stable state for the entire period of operation are acute at the mines of the first group, with increasing depth of development. Thus, at one of the mines, the development of the ore body and driving of the opening mining are performed at various depths (horizons) - from 7 to 18 horizons (400-1100 meters from the surface). Different horizons have different mining and geological conditions (pressed-deformed state of the rock massif, fracturing, strength of rocks, etc.). Security issues related to the risks of possible rock caving are on a priority. At the mines of the second group, the issues of increasing productivity and operation time, rational use of reserves and reduction of losses in the subsoil are more closely related to the problem of mineral extraction under the built-up areas.

Regional Risks, Their Causes and Prevention

Regional risks include the shift of the rock massif and the earth's surface over large areas, which may lead to the shutdown of the mine and possible human losses. The consequences of such risks are:
• formation and development of cave-in on the surface;
• deformation and destruction of buildings and structures;
• shift of the earth's surface in the built-up area of the residential area;
• flooding of mining with surface water.

Formation and Development of Cave-In on the Surface

To determine the dangers and risks of the formation of cave-in on the surface, certain actions are applied at all stages of mining, from design to execution of the works. The design stage includes the estimation of the possibility of crop out of the cave-in or rock fall. If the estimations confirm the potential crop out of the cave-in, the project is reviewed with the adjustment of the chamber parameters, the order of their development and filling.

During the stage of control of the works execution an operational schedule for scanning and filling of voids is developed by the following priority:
• long-standing voids;
• large outcrop area;
• areas of intensive mining.

The stage of corrective action develops the measures to prohibit the large-scale blasts above the level of the 11 horizon, in order to avoid the influence of dangerous earthquake load on the rock massif, wherein is a risk of overlying rocks caving. To control the development (increase in the area) of cave-in, instrumental surveying control using laser scanning is performed. The frequency of monitoring is at least 1 time per year.

If significant changes in the cave-in contours are found, the following measures should be taken to ensure safe execution of works:
• fencing of a potentially dangerous area;
• increase the frequency of laser scanning;
• reclamation of cave-in.

Deformation of the Earth’s Surface, Destruction of Buildings and Structures

The cause of deformation of the earth's surface, the destruction of buildings and structures is the shift of the rock massif and high rock pressure. In this case, mine shafts, complexes of pithead buildings, buildings of concrete-stowing complex, pumping stations and concrete-delivery pipelines are subjected to deformation and destruction. Cracks and subsidence of the earth’s surface appear. To control the deformations of the earth's surface, lines of control and monitoring are laid in areas with an increased risk of formation of critical deformations. A regular instrumental underground mine survey of objects is performed.

The following measures should be taken in case of revealing deformations exceeding admissible values, the formation of new cracks, the increase of existing cracks on the surface, the breaking of load-bearing structures, cracks in the buildings and structures:
• interruption of all types of work until the cause is identified;
• the withdrawal of personnel to a safe place;
• fencing of dangerous area;
• development of measures to rectify the incident;
control for the situation change (monitoring).

Figures 1-2 show the formation of cracks on the earth's surface and the control line.

Figure 1 - Formation of cracks on the surface

Shift of the Earth's Surface in the Built-Up Residential Area

Due to the fact that sometimes mining cover the territory of a residential area, there is a potential risk of shift and deformation of the earth's surface in this area. To determine the specific values of deformations and subsidence of the area, lines, which form an observational network, are laid. The frequency of monitoring by such lines is at least 1 time per month.

Critical values of subsidence and deformation for the given area are:
- subsidence - more than 30 mm / month;
- vertical deformations - 4 mm / m;
- visible cracks on residential buildings, earth’s surface.

The following measures should be taken in case of revealing deformations and subsidence exceeding admissible values, the formation of new cracks:

Figure 2 - Laying of the control line
· withdraw people from the dangerous area;
· restrict the access of people and vehicles to the dangerous area;
· perform the integrated analysis;
· develop measures to rectify all risks;
· control for the development of the situation (monitoring).

Flooding of Mining with Surface Waters

One of the possible risks during underworking of the safety pillars of river valleys, passing through the mine territory, is the flooding of mining with surface waters. At the stage of the mining operations design, measures are taken to protect the safety pillars of the rivers:
· prohibition of mining above the level of a certain horizon;
· stepwise development with division into turns;
· application of a filling system with a complete filling of the worked-out area.

Another potential risk associated with the sudden release of surface water into underground mining is the accumulation of melt water and rainwater, groundwater in the cave-in. One example of possible risks is the flooding of mining with melt water and rainwater through a cave-in located in the pit. Figure 3 shows the release of the water-mud flow through the cave-in located in the pit, into the mine.
Local Risks and Dangers

Local risks include the shift of the rock massif over the small areas at mining and on the surface, which can lead to the shutdown of the mine section, or to a breakdown in the technological process and possible human losses. Local risks include:

- caving of mining;
- formation of dangerous cracks and flaws, delaminations during driving and exploitation of mining;
- destruction of the pillars;
- deformation and destruction of the support;
- sudden release of water cut rock from the chute;
- caving of soil during excavation.

Caving of Mining

The main reasons for this risk include:

- presence of tectonic disturbances, crushed zones, flow rocks;
- weakening of the rock massif, as a result of water cut (saturation of rocks with water);
- increase of the limiting span;
- discrepancy of the support type with mining and technical conditions;
- failure of mining support;
- discrepancy of the filling material with the technical conditions;
- incomplete filling of the worked-out area;
- high pressure in the rock massif;
- large shifts of the rock massif.
The following actions should be taken to prevent this risk:

- qualitative description of the rock massif by the geological and hydrogeological services of the mine;
- accounting for mining and geological conditions during design;
- compliance with design decisions during mining operations;
- quality control of the filling material and completeness of the stope filling;
- quality control of the production of materials for mining support and its installation;
- monitoring of the state of the rock massif by personnel and feedback in case of change in mining and geological conditions;
- integrated geomechanics monitoring.

Figures 6-7 show the examples of caving of mining at one of the mines:

![Figure 6 - Caving of the roof of transport passage by tectonic disturbance](image)

![Figure 7 - Caving of the coupling roof owing to the water saturation of the roof – 18 horizon](image)

**Formation of Dangerous Cracks and Flaws, Delaminations during Driving and Exploitation of Mining**

During driving and exploitation of mining there is a risk of formation of dangerous cracks and delaminations of rock mass on the mining contours. There are three main causes of formation of dangerous cracks and delaminations:

- high pressure on the mining contour;
- structural disturbance of the rock massif (natural fracturing of the massif);
- formation of cracks as a result of earthquake load from explosions.
The following actions should be taken to prevent this risk:
- thorough trimming of flaws;
- selection of support types in accordance with geological conditions;
- application of special drilling and blasting methods, to reduce earthquake load on the contour massif;
- monitoring, identification of causes and prompt reporting on dangers.

Figure 8 shows an example of dome formation as a result of structural disturbance.

![Figure 8 - Dome formation at the coupling of 2 mining – 18 horizon](image)

**Destruction of the Pillars**

During the underground mining, one of the risks is the destruction of pillars of various types: interchamber pillar, crown pillar, etc. The main reasons for the destruction of the pillars are:
- underworking of the pillars;
- long standing period at high loads;
- wrong design decisions during calculation of the pillar parameters;
- tectonic disturbance, water cut of the rock massif.

The following actions should be taken to prevent this risk:
- compliance with design decisions during mining operations;
- checking the pillars stability by calculation methods with change of the technology and design decisions;
- qualitative description of the rock massif by the geological and hydrogeological services of the mine;
- development of measures to strengthen the pillars;
- integrated geomechanics monitoring.

**Deformation and Destruction of the Mining Support**

During driving and exploitation of the mining, there is a risk of destruction and deformation of the support: denudation of the anchors, breakage of the anchor heads, cutting of the anchors, pulling of the anchors, deformation of the base plates, deformation and breaking of the metal mesh, deformation of the steel arch, cracking and delamination of the concrete support.

The main causes of support deformation are:
- discrepancy of the support type with mining and technical conditions;
- corrosion of the support;
- "aging" of the metal of mining support;
- discrepancy of the geometric parameters of the mining with the selected support;
- tectonic and structural disturbance of the rock massif;
- high pressure at the mining contour;
• shift of the rock massif;
• violation of the support installation technology;
• discrepancy of materials with necessary characteristics.

The following actions should be taken to prevent this risk:
• qualitative description of the rock massif by the geological and hydrogeological services of the mine;
• quality control of the production of materials for mine support and its installation;
• revision of the support standard during driving of mining in case of significant changes in mining and geological conditions;
• monitoring of the state of the rock massif by personnel and feedback in case of violation of the support;
• compliance with design decisions during mining operations;
• integrated geomechanics monitoring.

Sudden Release of Water Cut Rock from the Chute

The risk of a sudden release of water cut rock from the chute can occur when the following factors are combined:
• direct water inflow at the mouth of the chute;
• water saturation of rock mass;
• hanging of rock mass (choking).

The following actions should be taken to prevent this risk:
• removal of mine waters from the mouth of the chute;
• actions preventing the hanging rock mass.

Caving of Soil during Excavation

During the execution of works on the earth's surface (the creation of holes, trenches, dug pits, ditches, and stank), exists a risk of the soil caving, if the following conditions are met:
• surface of soil outcrop exceeds the maximum admissible parameters;
• change of the temperature conditions;
• discrepancy of the support sides with the extraction conditions;
• non-compliance with design decisions;
• weakening of soil due to water saturation.

The following actions should be taken to prevent this risk:
• compliance with design decisions;
• accounting for hydrogeological conditions;
• rapid response in case of change in temperature conditions;
• monitoring of the installed support;
• monitoring of the soil state by personnel and feedback in case of change in technical conditions.

CONCLUSION

Geotechnical dangers and associated risks during development of the polymetallic deposits using underground methods are determined in this paper. There are given the evaluation (significance) of risks and the response procedures. All the measures to prevent both global and local risks are aimed at ensuring the safe mining operation.

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PETROGRAPHICALLY OBSERVABLE MINERAL MATTER AND CARBON CONTENT INFLUENCE ON SPONTANEOUS COMBUSTION LIABILITY OF COAL-SHALES

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ABSTRACT

Spontaneous combustion of carbonaceous materials is known to be a major problem in coal mining sectors. The frequent occurrence of self-heating of coal-shales has been reported as a cause of spontaneous combustion in the South African coal mines. The petrographically observable mineral matter and the carbon content of selected coal-shales have been investigated and their relationship with liability towards spontaneous combustion assessed. The spontaneous combustion liability index tests for coal-shales were conducted using a newly developed liability index (Wits-CT index). It was observed that coal-shales with a high carbon content and low petrographically observable mineral matter are more liable to spontaneous combustion. A definite positive correlation coefficient exists between carbon and the Wits-CT index, whilst a negative correlation exists between the petrographically observable total mineral matter and the Wits-CT index. It was found that the petrographically observable total mineral matter and carbon content in coal-shales have significant correlation on spontaneous combustion liability based on the linear regression analysis. The petrographically observable mineral matter along with the XRF data indicated that clay, quartz and pyrite were predominant, while other minerals occur as minor constituents and account for a small percentage of the bulk composition.

KEYWORDS
Spontaneous combustion; carbonaceous material; coal-shales; mineral matter; liability index; and Wits-CT index.

INTRODUCTION

Self-heating of coal is a major challenge during coal mining operations, which generally results in significant economic and environmental impacts (Liu & Zhou, 2010; Wessling et al., 2008). The occurrence involves a range of complex physical and chemical processes, caused by the chemical reaction between coal and oxygen and the associated heat liberated to the surrounding (Carras et al., 2009; Wessling et al., 2008). If adequate oxygen is available but the energy generated is not by the reaction sufficiently dissipated either by conduction, convection, or radiation, a further increase in temperature arises to further promote coal oxidation and self-heating (Yuan & Smith, 2011).

The self-heating of coal-shales has been reported in South African coal mines to be the likely cause of spontaneous combustion. The event has been observed in areas such as overburden shales, selected bands of a coal seam, spoil heaps, and highwalls. It was observed that coal, roof shales, and spoil heaps become self-heated and liberate heat naturally when subjected to atmospheric conditions (Figures 1 and 2). Spontaneous combustion of coal has been extensively studied in underground and opencast mines of South African mines using small-scale tests (Genc & Cook, 2015; Gouws et al., 1987; Wade, 1987). The influence of carbon content has been as one of the intrinsic factors affecting spontaneous combustion (Kaymacki & Didari, 2002; Nimaje & Tripathy, 2016; Sahu et al., 2005). However, no information has been reported on the influence of carbon content on coal-shale spontaneous combustion. Previous studies have been conducted on the self-heating of coal, both experimentally and computationally (Carras & Young, 1994; Gouws & Wade 1989a, 1989b; Kucuk et al., 2003; Stracher & Taylor, 2004), but limited studies exist on reactive material such as coal-shales. The occurrence of various intrinsic affecting self-heating is the reason for the difficulties in understanding the mechanism of spontaneous combustion.

Coal-shale is a sedimentary rock which is comprised of substances such as organic matter (carbon/macerals), inorganic matter (mainly crystalline) and fluids (air or water). The fluid exists in pores within and between the organic and inorganic constituents. The mineral matter includes clays, pyrite, carbonates, sulphides, and minerals. Carbonaceous shales with certain amounts of mineral matter may be reactive to self-heating. The presence of organic and inorganic constituents may be a fundamental method in evaluating the origin of carbonaceous materials (O’Keefe et al., 2013; Taylor et al., 1998) and in establishing their values for industrial purposes. Previous studies on the petrographic or chemical study of the organic constituents has been reported by (Beamish & Arisoy 2008; Beamish & Sainsbury, 2008). There is limited information on the influence of mineral matter on spontaneous combustion liability of coal-shales. However, these coal-shales may exasperate spontaneous combustion in coal mines.
This paper provides the consideration of two important factors (petrographically observable mineral matter and carbon content) that may affect the spontaneous combustion liability of coal-shale. Maceral analysis including mineral matter was conducted to evaluate the organic and inorganic content, while XRF was used to analyse the incombustible matter in order to determine the inferred mineral composition of the coal-shales. An understanding of these two factors may be important in understanding the process of coal-shale self-heating in mines. The samples used in this study form part of a series of samples assessed during a Ph.D. study.

Figure 1 - (a) Self-heating of highwall and waste dump and (b) Self-heating of inseam shale at Goedgevonden Colliery, eMalahleni, South Africa.

Figure 2 - (a) Burning spoil heaps at Tweefontein Mine, eMalahleni, South Africa.

MATERIALS AND METHODS

Sample Collection

The coal-shales used for this research were collected from two different coal mines in the eMalahleni area of South Africa and kept in airtight bags to avoid oxidation. Six representative in situ coal-shales samples were obtained from the affected areas (highwalls and overburden shales).

Sample Preparation and Characterisation

The sample lumps were reduced using a crusher and ball mill to suitable specify sizes as required for each test. The determinations of the moisture, ash and volatile matter contents were carried out according to the American Society for Testing and Material (ASTM) standards (ASTM, D-3173, ASTM, D-3174, ASTM, D-3175). The total carbon content was determined using a LECO TruSpec CHNS analyser after calibration with sulfamethazine based on the International Standard Organisation Standards (ISO 12902: 2001). The results were given in weight percent of air-dried (wt.%, ad). Maceral analysis including mineral matter was conducted to evaluate the organic and inorganic content of the coal-shales. A Zeiss Axio Imager M2m reflected light petrographic microscope fitted with oil immersion lens at a total magnification of x 500 was used. The petrographic pellets/blocks used for the petrographic analyses were produced by mixing 2 to 3 g of the grain size specify with epoxy resin in 3.2 cm diameter phenolic ring form moulds. The moulds were allowed to cure, and ground with the use of 320, 400 and 1000 grit papers. Once ground, they were polished using 1.0 and 0.3 μm alumina slurries on various cloth. Final polishes were achieved with the use of colloidal silica, which provides relief free and scratch-free surface for analysis by reflectance microscopy. The results of the macerals including
mineral matter were given in volume percent (vol.%). The ash oxide and the inferred mineral matter were also determined using the X-ray fluorescence technique (XRF). In this analysis, coal-shale ashes were prepared by placing a weighed amount (5g) of 250µm in a cold muffle furnace according to (ASTM, D-3682). The temperature was gradually raised to 500°C in 1hr and to 815°C in 4hrs. At the end of the heating period, the sample was removed from the furnace, covered, allowed to cool and then weighed. The ashes were fused with lithium tetraborate (Li₂B₄O₇) cast into a glass disk and programmed using a software called SuperQ to determine the major elements. The instrument used for the various elemental concentrations is called AxiosmAX in the School of Geoscience, University of Witwatersrand Johannesburg. The results of the characterisation and spontaneous combustion tests are presented in Tables 1 and 2.

**Spontaneous Combustion Testing Procedure (Wits-CT test)**

A device for predicting the spontaneous combustion liability of coal, coal-shale and other carbonaceous materials under the influence of airflow without any heating system was recently developed in the School of Mining Engineering, University of Witwatersrand, referred to as the Wits-CT test. The liability of different samples to spontaneous combustion was investigated and evaluated for 24 hours under conditions that resembled the atmospheric conditions. This experiment was designed to evaluate the temperature differences within the shortest period of time in a coal mass under the influence of oxygen. It has a capacity to accept a 15kg sample, depending on the packing density. It oxidizes materials at controlled air pressure and constant flow rate. Three uniformly separated temperature sensors placed along the length of the autoclave are used to check and record the temperatures measurement. The sensor arrangement typically comprises a predetermined number of temperature sensors located at different points within the autoclave (locations A, B, and C within the interior of the container). Each of the three thermocouples has two temperature sensors which measure temperatures at different levels at the locations A, B, C in order to keep a recording of the temperature distribution during the reaction of a carbonaceous material and oxygen. At location A, sensors are located at levels 13cm and 23cm within the container; location B, 53cm and 63cm, and location C, 33cm and 43cm. The experimental process simulates a well-insulated system relative to actual conditions in situ. The results obtained from this apparatus matches with work done by (Ozdeniz et al., 2015; Sensogut & Ozdeniz, 2005) in coal stockpiles.

In order to determine spontaneous combustion liability via self-heating, a representative sample of particle size (<6.39mm) was weighed and loaded into an autoclave. The process was repeated till the autoclave was filled to the marked point. The lid was fixed and fastened as soon as the autoclave was filled with the sample. This technique of preparation gives a closely sized sample that would be preferable to measure the rate of the oxidation process. Temperature probes were inserted into the column as the correct sample level was reached. Oxygen was supplied and controlled at a fixed flow rate (20ml/min) by means of a flowmeter before being fed into a manifold attached below the lid of the autoclave. The logging began and the changes in temperatures distribution were recorded by the data logger every minute. The test period lasts for 24 hours. The Wits-CT index uses the total carbon content and the temperature differences due to the reaction with oxygen to predict the spontaneous combustion liability. Carbonaceous materials with a spontaneous combustion liability index below 2.5 are considered to be less reactive, 2.5-5 moderately reactive, 5-7.5 reactive, and carbonaceous materials with values greater than 7.5 are considered to be highly liable to spontaneous combustion. An illustration of the experimental setup is indicated in Figure 3. The formulation of the Wits-CT index based is shown in Equation 1,

\[
\text{Wits-CT index} = \left(\frac{T_M}{24} + T_R\right) \times \%C_{ad}
\]

where,

\(T_M\) is the difference between the sum of maximum temperatures of each thermocouple in the autoclave and room temperature (22°C).

\(T_R\) is the difference between the peak temperature and initial temperature during oxidation reaction in degree Celsius.

\(\%C_{ad}\) is the air-dried percentage of carbon content of the sample.

24 is the test duration and is constant.
Figure 3 - Wits-CT apparatus
RESULTS

Microscopic examination shows that clay, pyrite and quartz were the dominant petrographically observable mineral matter in all the samples (Table 1).

Table 1 – Proximate analysis and total carbon (wt.%, ad), macerals and total petrographically observable mineral matter (vol.%), and spontaneous combustion test results.

<table>
<thead>
<tr>
<th>Samples</th>
<th>Moisture</th>
<th>Volatile matter</th>
<th>Ash (%)</th>
<th>Fixed carbon (%)</th>
<th>Carbon (%)</th>
<th>Total mineral matter (%)</th>
<th>Total vitrinite (%)</th>
<th>Total inertinite (%)</th>
<th>Total liptinite (%)</th>
<th>Clay (%)</th>
<th>Pyrite (%)</th>
<th>Quartz (%)</th>
<th>Wits-CT index</th>
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<tbody>
<tr>
<td>SE</td>
<td>1.7</td>
<td>15.9</td>
<td>68.4</td>
<td>14</td>
<td>15.8</td>
<td>65.2</td>
<td>3.2</td>
<td>26.1</td>
<td>5.5</td>
<td>38.6</td>
<td>16.2</td>
<td>6</td>
<td>1.6</td>
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<tr>
<td>SF</td>
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<td>13.5</td>
<td>76.9</td>
<td>8.7</td>
<td>11.8</td>
<td>76.2</td>
<td>0.6</td>
<td>20.8</td>
<td>2.4</td>
<td>53.7</td>
<td>4</td>
<td>4.8</td>
<td>1.36</td>
</tr>
<tr>
<td>SK</td>
<td>1</td>
<td>11.7</td>
<td>79.1</td>
<td>8.2</td>
<td>9.75</td>
<td>84.5</td>
<td>0.8</td>
<td>13.7</td>
<td>1</td>
<td>77.7</td>
<td>1.2</td>
<td>5.4</td>
<td>1.18</td>
</tr>
<tr>
<td>SL</td>
<td>1</td>
<td>16</td>
<td>74</td>
<td>9</td>
<td>10.5</td>
<td>78.6</td>
<td>8.4</td>
<td>9.6</td>
<td>3.4</td>
<td>74.4</td>
<td>0.8</td>
<td>3.2</td>
<td>1.34</td>
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<tr>
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<td>11.7</td>
<td>76.9</td>
<td>10.6</td>
<td>12.5</td>
<td>68.5</td>
<td>0.8</td>
<td>28.9</td>
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<td>43.4</td>
<td>1.4</td>
<td>13.3</td>
<td>1.44</td>
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<tr>
<td>SN</td>
<td>1.5</td>
<td>16.6</td>
<td>51.5</td>
<td>30.4</td>
<td>33.7</td>
<td>46.9</td>
<td>5</td>
<td>46.1</td>
<td>2</td>
<td>42.3</td>
<td>0.6</td>
<td>3.8</td>
<td>3.99</td>
</tr>
</tbody>
</table>

Table 2 – XRF analyses of ash samples (wt.%).

<table>
<thead>
<tr>
<th>Samples</th>
<th>SiO₂ (%)</th>
<th>Al₂O₃ (%)</th>
<th>Fe₂O₃ (%)</th>
<th>MnO (%)</th>
<th>MgO (%)</th>
<th>CaO (%)</th>
<th>Na₂O (%)</th>
<th>K₂O (%)</th>
<th>TiO₂ (%)</th>
<th>P₂O₅ (%)</th>
<th>Cr₂O₃ (%)</th>
<th>NiO (%)</th>
<th>LOI (%)</th>
<th>Total (%)</th>
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<tbody>
<tr>
<td>SE</td>
<td>60.05</td>
<td>23.51</td>
<td>9.33</td>
<td>0.11</td>
<td>1.05</td>
<td>1.27</td>
<td>0.14</td>
<td>2.73</td>
<td>1.02</td>
<td>0.16</td>
<td>0.04</td>
<td>0.01</td>
<td>0.31</td>
<td>100.72</td>
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<tr>
<td>SF</td>
<td>59.9</td>
<td>27.7</td>
<td>5.35</td>
<td>0.08</td>
<td>1.14</td>
<td>0.6</td>
<td>0.25</td>
<td>2.68</td>
<td>1.2</td>
<td>0.27</td>
<td>0.04</td>
<td>0.01</td>
<td>0.4</td>
<td>99.6</td>
</tr>
<tr>
<td>SK</td>
<td>67.93</td>
<td>24.92</td>
<td>1.11</td>
<td>0.01</td>
<td>0.25</td>
<td>0.13</td>
<td>0.07</td>
<td>2.76</td>
<td>1.56</td>
<td>0.07</td>
<td>0.04</td>
<td>0.01</td>
<td>0.48</td>
<td>99.35</td>
</tr>
<tr>
<td>SL</td>
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<td>30.92</td>
<td>3.39</td>
<td>0.03</td>
<td>1.07</td>
<td>0.45</td>
<td>0.09</td>
<td>1.94</td>
<td>1.24</td>
<td>0.23</td>
<td>0.04</td>
<td>0.01</td>
<td>0.64</td>
<td>99.78</td>
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<tr>
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<td>56.51</td>
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<td>0.15</td>
<td>1.3</td>
<td>1.42</td>
<td>0.1</td>
<td>2.37</td>
<td>1.13</td>
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<td>0.01</td>
<td>0.24</td>
<td>99.74</td>
</tr>
<tr>
<td>SN</td>
<td>59.6</td>
<td>24.9</td>
<td>9.02</td>
<td>0.1</td>
<td>1.18</td>
<td>0.56</td>
<td>0.25</td>
<td>2.85</td>
<td>1</td>
<td>0.14</td>
<td>0.04</td>
<td>0.01</td>
<td>0.22</td>
<td>99.85</td>
</tr>
</tbody>
</table>
DISCUSSION

Coal-shale SN has the lowest petrographically observable mineral matter content and coal-shale SK has the highest petrographically observable mineral matter content. Microscopic studies show that the studied coal-shales are enriched in mineral matter. The high petrographically observable mineral content could be associated with the high contribution of clastic material and conditions unfavourable for the preservation of botanical material. The high petrographically observable mineral matter content might act as a heat absorbing capacity within the coal-shale and hence, hinder the spontaneous combustion. This is in-line with the studies reported on coals by (Humphreys et al., 1981 & Smith et al., 1988). It was observed that the liability of the coal-shales to spontaneous combustion decreases with increasing petrographically observable mineral matter and vice versa. This is similar to the self-heating characteristics of coal in relation to the mineral matter content by (Beamish & Arisoy, 2008; Beamish & Sainsbury, 2008). Figure 4 shows the proportions of various petrographically observable mineral matter present in the coal-shales, while Figures 5 and 6 shows other organic and inorganic constituents in the analysed samples.

Figure 4 – Mineral matter observed from petrographic analysis in coal-shales (vol.%).

Figure 5 - Coal-shale SN showing the occurrence of quartz, vitrinite and clay (Reflected light x500, oil immersion, scale=100µm).
Figure 6 - Coal-shale SE showing the occurrence of pyrite, quartz and organic matter.

The results obtained from the XRF analysis are shown in Table 2. The mineralogical analysis determines which inferred mineral species are present in the ash sample obtained from the original coal-shales. All values reported in Table 2 are based on weight percentages (wt.%) on an ash basis. The coal-shale samples were found to have the relative elemental abundance in the order of SiO$_2$ > Al$_2$O$_3$ > Fe$_2$O$_3$ > K$_2$O > TiO$_2$ > MgO > CaO > Mn > P$_2$O$_5$ > Na$_2$O > Cr$_2$O$_3$ > NiO. The coal-shales were enriched mostly with SiO$_2$, Al$_2$O$_3$ and Fe$_2$O$_3$. Other inferred minerals occur as minor constituents and account for a small percentage of the bulk composition as shown in Figure 7.

Figure 7 - Proportion of minerals in coal-shales from XRF analysis (wt.%).

The air-dried carbon content varies between 9.75 wt.% to 33.7 wt.%. Sample SK has the lowest carbon contents and the lowest liability index, while sample SN with the highest carbon content has the highest spontaneous combustion liability index. It was found that samples with low carbon indicates low liability indices, while those with average high carbon content show a high liability indices. Therefore, as carbon content increases, the liability index increases. Most of the samples displayed low liability indices which may be due to their high petrographically observable mineral matter and low carbon contents. When the carbon content for the coal-shales was compared with each other, samples SN and SE are the most reactive under spontaneous combustion tests. It was found that the liability of coal-shales to spontaneous combustion may be directly related to the amount of carbon and petrographically observable mineral matter content in the material. Therefore, there is a relationship between the spontaneous combustion liability and carbon contents of the analysed coal-shales.
It was found that that coal-shales with high carbon and low carbon contents are likely to show similar characteristics to coal containing both high and low carbon content with respect to spontaneous combustion.

Figure 8 - Relationship between total petrographically observable mineral matter and spontaneous combustion liability.

Figure 9 - Relationship between total carbon content and spontaneous combustion liability.

Figures 8 and 9 show that as the petrographically observable mineral matter increases, the spontaneous combustion liability decreases and as the carbon content increases, the spontaneous liability increases. The sharp fall in the linear graph in Figure 8 is simply that there was no sample with total petrographically observable mineral matter content between 46 – 65 vol.%. Figure 9 shows that as carbon content increases, the Wits-CT index increases. Therefore, coal-shales with a higher petrographically observable mineral matter and high carbon content show a higher spontaneous combustion liability and vice versa.

Linear regression analysis

Linear regression analysis was carried out by correlating carbon content and petrographically observable mineral matter and its constituents as independent variables with the values of the Wits-CT as dependent variables. The overall database involved the spontaneous combustion test results and carbon content and petrographically observable mineral matter of 6 samples in order to determine the influence of these factors on spontaneous combustion. Table 3 presents the results of the linear regression analysis.
The increasing carbon content influences the oxidation potential of the coal-shales by causing a relatively high spontaneous combustion liability as shown in Figures 13. Thus, it was found that as the carbon content increases, the spontaneous combustion liability of the samples is more likely to increase. The values obtained produced a definite linear relationship between carbon content and the Wits-CT index. The results showed a positive effect on the oxidation potential.

The total petrographically observable mineral matter shows a strong linear relationship with spontaneous combustion liability index. Although, a negative effect between the oxidation potentials and various mineral matter exists (Figures 11, 12, 13 and 14). This relationship is partly due to the heat absorbing capacity of the mineral within the coal-shale. It was found that as the total petrographically observable mineral matter increases, the spontaneous combustion liability decreases. The presence of the petrographically observable mineral matter constituents within a coal-shale is more likely to retard the start of self-heating. This is in-line with the studies reported on coal by (Beamish & Arisoy, 2008; Beamish et al., 2005; Humphreys et al., 1981). Clay minerals among the petrographically observable mineral matter constituents have a higher correlation coefficient than the quartz and pyrite as shown in Table 3. As the correlation of coefficient is stronger than the other two mineral matter constituents, clay mineral is more likely to have an influence on coal-shale spontaneous combustion than quartz and pyrite.

Figure 10 - Influence of carbon content on coal-shale spontaneous combustion.
CONCLUSION

This paper used samples from two coal mines which form part of a series of samples assessed during a Ph.D. Six representative in situ coal-shales samples were obtained from areas affected by spontaneous...
combustion (highwalls and overburden shales). Total petrographically observable mineral matter and carbon content have been identified as two main factors influencing the spontaneous combustion of coal-shales. This study shows that coal-shales with a low total petrographically observable mineral matter and high carbon content are more liable to spontaneous combustion. A definite positive correlation coefficient exists between carbon and the Wits-CT index while a negative correlation exists between the total petrographically observable mineral matter and the Wits-CT index. The results acquired from the XRF analysis determined that the ash obtained from the coal-shale samples was rich in SiO₂, Al₂O₃ and Fe₂O₃, which is in-line with the petrographic results. Other minerals occur as minor constituents and account for a small percentage of the bulk composition. It was found that the total petrographically observable mineral matter and carbon content in coal-shales have significant correlation on the spontaneous combustion liability based on the linear regression analysis.

ACKNOWLEDGMENTS

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ROLE OF DIURNAL AND SEASONAL VARIATIONS IN PSYCHROMETRIC PROPERTIES OF INLET AIR ON THE PREDICTION OF CLIMATE IN SHALLOW MINE ENTRIES - A CASE STUDY

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ABSTRACT

Climate prediction in shallow mine entries is a challenging task because of the large-scale variations in the inlet air psychrometric properties between day and night and also among seasons of the year. A characteristic thermal flywheel phenomenon sets in-terms of the heat content in the neighboring rock. Virgin rock temperature (VRT) is higher than the dry bulb temperature (TD) of air during cooler night times and VRT is lower than the TD of the air during warmer daytime. During the warm daytime the surrounding rock tends to accumulate the heat from the incoming air, and during the cooler night times, the stored heat from the rock is released into the flowing air.

A year-long study is conducted in an intake incline of a coal mine in Jharia basin in the Jharkhand State of India, to gain an appreciation of the diurnal and seasonal effects on rock to air heat transfer. The incline has a gradient of 1 in 3 and has a length of 180 m. Air flow through the incline is 36 m$^3$/s, the surrounding formations are mostly sandstone. The geothermal gradient in these parts of the country is taken to vary at 38m/$^\circ$C and the temperature 20 m below ground is taken to be 24$^\circ$C. On a given day in each month inlet and outlet psychrometric properties are measured across the intake incline at 2 hours interval. Based on the findings the cyclic behavior the heat load is documented.

In the present paper climate prediction at the incline outlet is attempted through the radial heat conduction based analytical procedure. The prediction is however limited by the uncertainty as well as variability in the strata thermal properties and quantification of airway wetness in the sense that the precise quantification of the wetness of the airway is a difficult task and may produce errors.

KEYWORDS

Climate prediction, Transient, Thermal flywheel, Psychrometric properties, Shallow mine, Diurnal and Seasonal

INTRODUCTION

In entries of shallow depth mines, surface intake air itself can be a major source of heat and moisture. The psychrometric properties of intake air, such as the dry-bulb temperature (Td) and the wet-bulb temperature (Tw), have higher variations which get dampened out as it travels through the mine. The walls and rock surrounding the intake shaft would absorb the heat from the incoming air during warm daytime conditions and the heat may be released back during the night time. This phenomenon is described by McPherson (1986) as “Thermal Flywheel”. It is also possible that heat is retained by surrounding rock during hot seasons of the year and released during the colder seasons.

Different methods for the prediction of underground (U/G) climate have been developed under two basic categories, the empirical and the analytical methods. The works of Lambrechts (1959), Wiles (1959) and Whillier (1967) are of empirical nature, while the methods given by Starfield (1966), Jaeger, J. C., & Chamalaun (1966), Amano et al (1982), Banerjee & Chernyak (1985), and Mishra (1987) belong to theoretical category. Most of these methods assume that the air temperatures in the airway are either constant or display small changes. Vost (1976) pointed out the large variations in wet and dry-bulb temperature in an airway due to variations in atmospheric conditions on the surface. He conducted measurements for fourteen days in an airway to access the changes in rock temperature and airway psychrometric properties due to variations in atmospheric condition at the surface. He found out that rapid changes in air temperature throughout the ventilation circuit give rise to large ranges in air temperature gradient, and therefore it is not sufficient to predict the average gradients and the average temperatures at output; and concluded that “In a theoretical solution, the full air-temperature history at input must be used with a method incorporating Duhamel's theorem”. Cheung (1988) has given an algorithm based on Duhamel’s theorem to predict heat transfer under variable air temperatures in the dry roadway.

Peavy (1961) presented the results of a mathematical analysis of the heat transfer between the air and the rock, assuming that the temperature of the entering air shows periodical and sinusoidal variation. To show the application of the method in practical problems, the maximum and minimum temperatures at a distance in a tunnel was calculated. Although Peavy’s model is only applicable for dry airways and no wet bulb calculations are given, his model is among the first transient climate prediction models.
Although there are many transient underground prediction models to predict transient heat flow between strata and ventilating air and hence psychrometric properties of the airway, most of these models are given for section of an underground roadway where the inlet $T_d$ and $T_w$ are taken as steady state while the VRT boundary is assumed to be function of the age of the roadway. These models cannot be relevant for the shallow mine entries where the inlet air properties are highly variable and the rock to air heat transfer is significantly affected by the thermal flywheel effect.

This paper aims to highlight the comparison between the field observations of 24 hours psychrometric study of shallow intake incline on different days for a year at one of the Indian coal mines; and the corresponding climate prediction at the incline outlet with the help of the radial heat conduction based analytical procedure. The impact of the temporal variations in the psychrometric properties of intake air on the climatic condition at a depth of 62 m in an intake incline of 1 in 3 gradient will be the main focus of the discussion.

### SITE AND APPARATUS

On different days spread over a year, twenty-four-hour studies are conducted at time intervals of two hours, comprising a psychrometric survey at the surface (top of incline) and at an underground measurement station (U/G) in intake incline. The incline has been serving as an intake for the last 15 years. The U/G is situated at a distance of 180 m from incline shaft collar. The test section of the airway (surface to U/G) selected for the measurements have 3.05 m by 6 m cross-section and there are no splits in the airflow or sharp bends. The 24-hour studies are carried out on thirteen occasions from 2016 to 2017. An open drain varying in width between 0.3 m and 0.5 m ran down one side, but there is a very little flow of water. A description of the wetness of the floor and walls between the top of the incline and underground measurement station is given in Table 1.

<table>
<thead>
<tr>
<th>Distance from Shaft collar (m)</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>0 to 50</td>
<td>Dry Floor, roof, and side walls.</td>
</tr>
<tr>
<td>50 to 60</td>
<td>Seepage of water from roof, wet floor, water in drain.</td>
</tr>
<tr>
<td>60 to 100</td>
<td>Drain contained a little water, rest of floor damp.</td>
</tr>
<tr>
<td>100 to 180</td>
<td>Drain contained a little water, rest of floor dry.</td>
</tr>
</tbody>
</table>

Aneroid Barometer with least-count of 1mmHg is used to measure barometric pressure at measuring points. A Lambrecht Assmann psychrometer having with a least-count of 0.2 °C, is used to record the temperatures. At least 5 minutes was allowed for the psychrometer to attain equilibrium before the measurements are taken on each occasion. Since temperatures can vary in a given cross-section at a given time, psychrometric measurements are taken at the midpoint of the cross-section.

The procedure of air quantity survey is carried out with the help of the one-minute vane anemometer and the metric tape. The one-minute vane anemometer used for velocity measurement in the incline using continuous traversing procedure is a Wilh Lambrecht GmbH instrument having a least-count of 1.0 m/min. The anemometer is calibrated in a low-speed wind tunnel before readings are noted. The air quantity at the measurement station is measured once every 24-hour and assumed to be constant for the study period.
RESULTS

Field Study

The dry-bulb and wet-bulb temperatures obtained at the surface and at underground measurement station for each survey occasions. Figure 1 and 2 shows the diurnal variation of $T_d$ and $T_w$, at both the locations, for a survey done on 29th April’17. The temperatures are marked on the vertical axis in $^\circ$C and survey times are marked on the horizontal axis in hours. Along with $T_d$ and $T_w$, barometric pressures (P) are also measured.

![Figure 1 - Dry-bulb temperatures at surface and underground measurement station (U/G) for 29th April’17](image)

![Figure 2 - Wet-bulb temperatures at surface and underground measurement station (U/G) for 29th April’17](image)

During the study period, it is noted that the variation in $T_d$ is found to be more than the variation in $T_w$ for both the locations (surface & U/G). Typically, the $T_d$ at the surface are at higher values than that of U/G in daytime i.e. from 8.30 AM to 6.30 PM. The values of $T_w$ for the surface air, at a given time, are found to be less than that for the U/G. This is understandable, as the intake air receives additional moisture from the mine seepage water.

In a day, barometric pressure typically shows the semi-circadian behavior (12-hour cycles). Figure 3 shows the variation in barometric pressure on 29th April 2017.
All other psychrometric properties of air are calculated using corresponding Td, Tw, and P values. Figure 4 shows the average values of moisture content (mr) at U/G and relative humidity (RH) at the surface, for each survey periods. The monthly precipitations at the surface are also presented with these values.

Figure 4 shows the relation of U/G moisture content with the precipitation rate and relative humidity at the surface. The months of higher precipitation rate have higher values of moisture content at both the survey locations i.e. at the surface and U/G. The higher precipitation rate at the surface increases the moisture content of intake air from the surface and also enhances the rate of seepage of water in underground and hence increases the moisture content of air at U/G.

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The seasonal variation of sensible heat (SH) and latent heat (LH) at U/G are shown in Figure 5. The average values for SH and LH for U/G are plotted for different survey occasions. It can be seen from this figure that
the difference of LH and SH are less for the cooler and dry months (Nov’16 to Mar’17), but as the month progresses
the contribution of latent heat increase.

Simulation of the Climatic Conditions in the Subsurface Under Study

The common feature of mine climate simulation models is that they are based on solutions of the radial
heat conduction equation, and on the utilization of the dimensionless Fourier and Biot numbers. However, the
programs may vary in the manner in which they determine rock surfaces temperatures and heat transfer coefficients,
and in the characterization and treatment of wet surfaces (Mousset-Jones & McPherson 1986). In this section, the
underground climate simulation package CLIMSIM (McPherson 1986a) is discussed. Although the equations for
heat conduction, psychrometry, and algorithm given by Gibson for computation of the dimensionless temperature
gradient (G) are exactly same, the simulation model is modified in such a way that it can take a number of discrete
input values to produce a number of discrete output values. MATLAB 2016a is used for this purpose. Along with
measured psychrometric properties (P, Td, Tw) at the surface, different airway and rock thermal properties are also
given in this modified simulation model as input which are listed in Table 2.

Apart from the different psychrometric properties of intake air and the parameters listed in Table 1, one of
the most critical inputs to the simulation model is wetness factor (wf). The wetness factor is defined as the fraction
of airway perimeter that is wet (McPherson 1986b). The usual procedure is to assign a number between 0 (for fully
dry) and 1 (for fully wet) by visual inspection of the airway condition. Most of the times the value assigned by the
user could not present the real picture of wetness of the airways as in the majority of mines evaporation occurs even
for apparently dry rock. The dry bulb temperature of the air is particularly sensitive to the wetness of the airway.
Fortunately, wet bulb temperature, which is much more important in governing the cooling power of the air, is not
affected directly by wetness factor. It is, however, influenced slightly and indirectly by airway wetness in that the
dry bulb temperature affects the heat flow into the airway and this, in turn, governs the enthalpy (total heat content)
of the air and, hence, the wet bulb temperature.
Table 2 - Input parameters

<table>
<thead>
<tr>
<th>Parameters</th>
<th>Values</th>
</tr>
</thead>
<tbody>
<tr>
<td>Length of the airway (m)</td>
<td>180</td>
</tr>
<tr>
<td>Height of the airway (m)</td>
<td>3.05</td>
</tr>
<tr>
<td>Width of the airway (m)</td>
<td>6</td>
</tr>
<tr>
<td>Vertical depth from surface (m)</td>
<td>60</td>
</tr>
<tr>
<td>Friction factor (Ns²/m⁴)</td>
<td>0.014</td>
</tr>
<tr>
<td>Quantity of airflow (m³/s)</td>
<td>37</td>
</tr>
<tr>
<td>Virgin rock temperature at inlet (°C)</td>
<td>24</td>
</tr>
<tr>
<td>Geothermal gradient (°C/m)</td>
<td>0.033</td>
</tr>
<tr>
<td>Rock Thermal conductivity (W/m°C)</td>
<td>2.5</td>
</tr>
<tr>
<td>Rock heat capacity (J/kg/°C)</td>
<td>1000</td>
</tr>
<tr>
<td>Rock density (kg/m³)</td>
<td>2300</td>
</tr>
<tr>
<td>Number of segments</td>
<td>10</td>
</tr>
<tr>
<td>Age of the airway (s)</td>
<td>3.46E+08</td>
</tr>
</tbody>
</table>

Figure 6 - Measured and predicted dry-bulb temperature using different wetness factor values at U/G for 29th July’17

During the study period, it is observed that the rate of water seepage changes according to the time of the day and also to a day of the year. This makes even more difficult to correctly predict the psychrometric properties at the outlet by using a single value of wetness factor for a mine airway.
Figure 6 and 7 shows a comparison between actual measurements at U/G and the predicted Td and Tw at different wf for one of the survey occasion (29th April’17), respectively. The effect of wetness factor of the airway on the Td and Tw at U/G can be observed clearly from these figures. The climate simulation model appears to over predicting the Td values at U/G at lower wf values. The satisfactorily predicted Td values at U/G is found by using wf in the range of 0.3 to 0.4. The plots for predicted wet-bulb temperatures at U/G, for different wf values, are overlapped. Also, the predicted Tw values are not in agreement with the measured values at the U/G, although the trend matches.

![Graph showing measured and predicted wet-bulb temperature using different wetness factor values at U/G for 29th April’17](image)

**CONCLUSIONS**

Variations in atmospheric conditions on the surface produce large changes to the underground environment. Figure 1 to 3 shows how these climatic variations in psychrometric properties of surface air influence the underground environment of an intake incline. The non-uniform variation in surface temperature dampens out as it travels through the 180 m incline, and gives rise to less difference in daily maximum and minimum temperatures at the underground measurement station. It has been also observed that the psychrometric properties at U/G changes according to the time of the day and also the day of the year. The large variations in the psychrometric properties of air on surface and underground suggest that it is not sufficient to use average values of the psychrometric properties of intake air for predicting underground air properties. The need of providing the input air temperature history to calculate the average gradients and the average temperatures at the outlet is also mentioned by Vost (1976).

An underground climate simulation model is used to understand the reliability and reproducibility of the actual underground situation and to compare the predicted results with the actual field measurements. This simulation model uses the radial heat conduction based analytical approach and the algorithm provided by Gibson (McPherson 1986a). During the study period it has been observed that, although the airway under study appears to be relatively dry, the wetness factor for the airway section under study has the value in the range of 0.3 to 0.4. This suggests that the wetness factor needed to address carefully for predicting underground air properties.
Apart from the fact that this underground climate prediction model was originally developed for the sections of the underground roadways and working faces, the major reason behind the difference in measured and model output values might be due to the fact that this model is originally developed to capture the steady state behaviour rather than the transient behaviour, as in the present case. Therefore, a more refined transient model is desirable to capture the diurnal and seasonal effect on heat flow between strata and ventilating air for mine entries.

REFERENCES


INFLUENCE OF DISTRIBUTION OF DIAMETERS OF WATER DROPLETS CONDENSED IN COOLED WET AIR STREAM ON PRESSURE LOSS IN VERTICAL DUCT

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INFLUENCE OF DISTRIBUTION OF DIAMETERS OF WATER DROPLETS CONDENSED IN COOLED WET AIR STREAM ON PRESSURE LOSS IN VERTICAL DUCT

ABSTRACT

This research paper presents the methodology for determining the water mass in ducts with a fixed two-phase flow. The dispersed liquid phase (water) is present in the air stream in the vertical duct due to the occurrence of the water mass source distributed along the pathway. This source results from the phenomenon of water vapor condensation in the cooled stream of wet air. These research studies have predetermined that water droplets have different diameters and their spectrum can be described by the Rosin-Rammler distribution. Assuming the exemplary values of two experimental constants present in this distribution and using specific data for the studied duct, the size of the mass flow rate of water droplets falling counter-currently in the vertical duct, were quantified. For such a case of a fixed flow, static pressure loss resulting from the gravity force of the mass of water droplets present in the air stream of the studied duct were specified. The determined amount of pressure loss was compared with the value corresponding to the co-current no-slip flow of water droplets and with the case of their monodisperse flow with the air in this duct.

KEYWORDS

two-phase flow, co-current and counter-current flows of the dispersed phase, pressure loss, flow characteristics of the duct, Rosin-Rammler distribution

INTRODUCTION AND STUDY OBJECTIVE

Effective ventilation of underground workings in mines is carried out by continuous feeding of adequately large flows of wet air. This air flows through workings referred to as ducts (Waclawik, 2010). These ducts are variously shaped and the air, along its pathway, is subjected to thermodynamic changes: heating, humidifying, cooling, mixing. These phenomena are described in the research paper (Ptaszyński, Łuczak, Zyczkowski, & Kuczer, 2018), which provides examples of such transformations of wet air. In some of these ducts, water droplet condensation may occur in the workings. Since the volume of the wet air stream flowing through the main mine workings is usually large, it is important to determine the value of the mass flow rate of water droplets flowing through the ducts on the air discharge pathway from the mine. This procedure is described in (Ptaszyński, 2016; Ptaszyński, 2018a; Ptaszyński, 2018b). To maintain air flows in the mine, main ventilation fans are used, which usually work in a suction mode. They generate static underpressure along the air flow pathway. This underpressure increases its value on the pathway traveled by the flowing air. The phenomenon of the changing pressure of the flowing air is described in (Ptaszyński, 2016), where the method of using the Mollier diagram for determining thermodynamic processes in the wet air stream occurring at the significantly changing static pressure is presented. As a result, for known parameters of thermodynamic air flow rates (defined by standard ventilation measurements in underground workings), it is possible to determine whether there are sections of ducts on the air discharge pathway where the phenomenon of water vapor condensation occurs, and what the water mass flow rate is, relative to the unit of length in a given duct. From the considerations presented in the paper (Ptaszyński et al., 2018), it can be concluded that positive water mass sources from water vapor condensation, which are distributed mass sources at the finite sections of the ducts, as well as locally located water mass sources or their outlets, resulting from the water condensation or evaporation of water droplets from the mixing of wet air flow rates and haze air flow rates containing water mist, may occur on the pathways of air discharge through the ducts. As it is demonstrated in these studies, some of the values of water mass flow rate depend on the time of the year (summer/winter). In the course of a year, the character of the local source may change, which in one period may be positive (source) and in another negative (outlet). The Author also identifies those ducts on the analyzed air pathway, whose only function is to carry a fluid, i.e. those with no water mass source or outlet from the condensation of water vapor. The research paper (Ptaszyński, 2015) introduces the concept of flow characteristics of a duct through which a two-phase medium flows. It has been pointed out that for inclined ducts with such a flow, the value of the so-called pressure loss is important. This pressure loss results from the component of the force of gravity of the mass of water droplets present in the air flow rate in the duct, parallel to the velocity vector of a two-phase fluid. The papers (Ptaszyński, 2018a; Ptaszyński, 2018b) specifies the value of this pressure loss for a two-phase flow in variously shaped ducts, both with and without this internal water mass source. In the first case, these values were calculated taking into account the no-slip co-current flow of the gas phase (which is the saturated wet air) and of the dispersed phase (which are water droplets). In the second case, the values of the mentioned pressure loss were determined for the co-current flow of these phases, yet considering the existence of the so-called slip between them, which is the difference in velocity of the flows of these phases. At the same time, monodispersity of the dispersed phase, the whole of which is carried up with the air to the surface of the mine, has been assumed. The results demonstrated in these papers prove that the aforementioned pressure loss can
vary within such a wide range of values that it may be essential to ensure effective and safe ventilation in the mine. Although the discussed cases of a two-phase flow (no-slip and monodisperse with a slip) do not occur, but as theoretically extreme, they determine the range of possible values of the aforementioned pressure loss. In real situation, we do not deal with droplets of the same diameter, nor with their flow with velocity which is the same as the one of the air in vertical or inclined ducts.

This research paper concentrates on the fact that in two-phase flows of the air and water droplets in mine workings, their diameters are distributed. The influence of taking the spectrum of flowing droplets into consideration, while maintaining the assumption of the non-existence of the phenomena of water droplet merging and disintegration, will allow to specify the mentioned pressure loss more accurately. The knowledge of this value may be important for designing safe conditions for ventilation of underground mine workings.

**SPECTRUM OF WATER DROPLET DIAMETERS IN AIR**

Water droplets contained in haze air streams can be treated as a statistical set, where the diameter of a droplet $d_{w}$ is the random variable. The distribution of droplet diameters creates the spectrum of their diameters (Orzechowski & Prywer, 2008). From mathematical statistics, this concept is understood as the dependence between the number of droplets $\Delta n_i$ in the interval of diameters $<d_{wi} - \Delta n_i/2, d_{wi} + \Delta n_i/2>$, where $\Delta n_i$ is the fixed width of the interval and the diameter $d_{wi}$, which is the median of each i-th interval of its diameters. The total number of droplets N is the sum of the number of water droplets in all intervals of its diameters, i.e. $N = \sum^m_{i=1} \Delta n_i$, where m is the number of intervals of droplet diameters. Such a function presents the probability density distribution of a continuous random variable $d_{wi}$ and it is called a number frequency droplet distribution curve. This function is defined in the interval of values $d_{wmin},d_{wmax}$ with the following Formula (Orzechowski & Prywer, 2008):

$$f_n(d_w) = \frac{\Delta n_i}{\Delta d_{wi}} \frac{1}{\sum^m_{i=1} \Delta n_i}$$

If the diameter $d_{wi}$ is assumed to be the continuous variable in the interval $(0, \infty)$, the probability density function (the number frequency droplet distribution function) is expressed by the dependence:

$$f_n(d_w) = \frac{dn}{\Delta d_{wi}}$$

The use of one or the other interval of diameter values does not result in significant errors (Orzechowski & Prywer, 2008). The cumulative distribution function of droplets, i.e. the cumulative distribution function of the variable $d_{wi}$, for the above intervals of the random variable, takes the following forms:

$$\phi_n(d_w) = \sum^i_{1} \Delta n_i$$

$$\phi_n(d_w) = \frac{\int^d_{d_{w}} \frac{dn}{\Delta d_{wi}} d(d_w)}{\int^\infty_{0} \frac{dn}{\Delta d_{wi}} d(d_w)}$$

For the entire range of the random variable, the following dependencies hold:

$$\sum^m_{i=1} \Delta n_i = 1$$

$$\int^{\infty}_{0} \frac{\Delta n_i}{\Delta d_{wi}} d(d_w) = 1$$

The above distribution functions of the number of water droplets in the air can also be used to describe the distribution of mass, volume and area of these droplets. In the general case, the probability density function and cumulative distribution function for droplets take the following forms (Orzechowski & Prywer, 2008):

$$f_d(d_w) = \frac{dn}{\Delta d_{wi}} \frac{d(d_w)^a}{\int^{\infty}_{0} \frac{dn}{\Delta d_{wi}} d(d_w)^a d(d_w)}$$

$$\phi_d(d_w) = \frac{\int^d_{d_{w}} \frac{dn}{\Delta d_{wi}} d(d_w)^a d(d_w)}{\int^{\infty}_{0} \frac{dn}{\Delta d_{wi}} d(d_w)^a d(d_w)}$$
If \( a = 0 \), then the above Formulas refer to the quantitative distribution of droplets, if \( a = 2 \), then to the surface distribution, and when \( a = 3 \), these Formulas refer to the mass distribution of droplets.

The aforementioned distribution functions can be expressed by different equations which, in a better or worse way, approximate the spectrum of water droplets in the air. The measurement of the spectrum of water droplets in the ventilation air in mines has not been carried out so far, primarily due to the lack of a suitable device for such tests, as well as technical difficulties in the implementation of such measurements. Therefore, it is possible to use only the results of experimental research regarding the spectrum of dispersed water droplets in the air carried out in other fields of technology. These results demonstrate that the Rosin-Rammler distribution is best suited for determining the cumulative mass or volume distribution of water droplets in the air. Therefore, the Rosin-Rammler distribution was used for this study. This equation is a cumulative distribution function of the mass distribution of water droplets in the air and has the following form (Orzechowski & Prywer, 2008):

\[
\Phi_3(d_w) = 1 - \exp \left[ -\left( \frac{d_w}{X} \right)^\delta \right]
\]

(9)

where \( X \) is a parameter of diameters size, \( \delta \) is a parameter of diameters distribution defining their uniformity. This equation determines the mass concentration of droplets with diameters smaller than \( d_w \). For \( X = d_w \) the value of \( \Phi_3(d_w = X) = 0.6321 \). It can be said that the value of \( X \) determines such a diameter \( d_w \) of the droplets for which \( \Phi_3(d_w) = 0.6321 \). Thus, the parameter \( X \) demonstrates that 63.21% of the mass of water droplets present in the air consists of droplets with diameters smaller than the value of \( X \). The influence of the parameter \( \delta \) proves that the smaller its value, the less homogeneous the distribution of water droplets is. This is manifested by a smaller slope of the distribution curve. Under real conditions of water dispersion in the air, the parameter \( \delta \) falls within the value interval of \( 2 \div 4 \) (Orzechowski & Prywer, 2008).

Having differentiated the Rosin-Rammler equation relative to the variable \( d_w \), a probability density function is obtained. Its form is expresses by Formula:

\[
f_3(d_w) = \frac{d}{d(d_w)} \Phi_3(d_w) = \frac{\delta}{X^\delta} (d_w)^{(\delta - 1)} \exp \left[ -\left( \frac{d_w}{X} \right)^\delta \right]
\]

(10)

Here, it may be asked, for what value of \( d_w = d_M \), the value of the function \( \Phi_3(d_w) = 0.5 \). By substituting the value of 0.5 for the left side of the equation (9), after the transformations presented below, it is obtained:

\[
0.5 = 1 - \exp \left[ -\left( \frac{d_M}{X} \right)^\delta \right]
\]

(11)

\[
-\left( \frac{d_M}{X} \right)^\delta = \ln(0.5) = -0.69315
\]

(12)

\[
d_M^\delta = 0.69315X^\delta
\]

(13)

\[
X^\delta = \frac{d_M^\delta}{0.69315}
\]

(14)

By substituting (14) to the equation (9), it is obtained:

\[
\Phi_3(d_w) = 1 - \exp \left[ -0.69315 \left( \frac{d_w}{d_M} \right)^\delta \right]
\]

(15)

The value of \( d_M \) denotes such a value of the droplet diameter which divides their population into half, taking into account their cumulative mass. This means that 50% of the mass of water droplets have diameters \( d_w \leq d_M \), and the remaining 50% of their cumulative mass have diameters \( d_w > d_M \). Having adopted the values: \( d_M = 0.0008 \text{ m}, \delta = 2.5 \), the curve defined by Formula (15) has been illustrated in Figure 1.
Each diameter of a water droplet $d_w$ in the air can be assigned a velocity of its free fall $v_\infty(d_w)$, according to the following Formula (Ptaszyński, 2018a; Ptaszyński, 2018b):

$$v_\infty(d_w) = 1.74 \sqrt{\frac{g \rho_w}{\rho_p}} \sqrt{d_w}, \text{ m/s}$$  \hspace{1cm} (16)$$

where: $\rho_w$, $\rho_p$, $g$, $d_w$ denote, respectively: water density, air density, kg/m$^3$, gravitational acceleration, m/s$^2$, water droplet diameter, m.

By determining the value of $d_w$ from the equation (16), which is equal to:

$$d_w = \frac{[v_\infty(d_w)]^2 \rho_p}{1.74^2 g \rho_w}$$  \hspace{1cm} (17)$$

and substituting it to Formula (15) for $\Phi_3(d_w)$, the following dependence is obtained:

$$\Phi_3(d_w) = 1 - \exp \left(-\frac{0.69315 \rho_p \delta}{1.74^2 g \rho_w} \left(\frac{[v_\infty(d_w)]^2}{d_M}\right)^\delta\right)$$  \hspace{1cm} (18)$$

Having predetermined that $g = 9.81$ m/s$^2$; $\rho_w = 998$ kg/m$^3$, the probability density function takes the following form:

$$f_3(d_w) = \frac{1.3863 \cdot 29641.4^{-\delta} \cdot \rho_p \delta [v_\infty(d_w)]^{\delta \cdot [v_\infty(d_w)]^{2}}}{d_M} \cdot \exp \left[-\frac{0.69315 \cdot 29641.4^{-\delta} \cdot \rho_p \delta \cdot [v_\infty(d_w)]^{2}}{d_M}\right]$$  \hspace{1cm} (19)$$

Having predetermined that $g = 9.81$ m/s$^2$; $\rho_w = 998$ kg/m$^3$, functions $\Phi_3(d_w)$ and $f_3(d_w)$ described by equations (18) and (19), mathematically dependent on variables: $\delta$, $d_M$, $v_\infty(d_w)$, $\rho_p$. The first two of these are constants of the discussed Rosin-Rammler distribution. The third one is the velocity of free fall of water droplets in still air, expressed in m/s, which is a function of the diameter of water droplets, and the fourth one is the air density. Such a presentation of these distribution functions is convenient: by predetermining the first two constants and knowing the vertical component of the mean velocity of the air in the duct and its density, it is easy to determine the mass concentration of the droplets transported with the air via this duct. In the further part of this paper, it was predetermined that in each duct being the analyzed wet air discharge pathway from an underground mine, the distribution of the diameter of water droplets present in the air flowing through a given duct is the same.

Figure 2 illustrates the form of the cumulative distribution function of droplet diameters in the duct (1-2) in which the mean air density is $\rho_p = 1.1902$ kg/m$^3$, and the distribution constants are equal to $\delta = 2.5$; $d_M = 0.8 \cdot 10^{-3}$ m. In addition, the points corresponding to the values of the mean air velocity at individual sections of the wet air discharge pathway, described in (Ptaszyński et al., 2018), were marked on the curve. As it can be seen, all points, except for one, correspond to the values of the cumulative distribution function of droplet diameters equal to one. This one point, corresponding to a mean air velocity equal to $\nu = 4.56$ m/s, refers to the first duct (1-2) on the air discharge pathway described in the above-discussed studies.
Figure 2 - Cumulative distribution function $\Phi_3(d_w)$ of water droplet diameters in duct (1-2) depending velocity of free fall of water droplets $v_{\infty}$. 

Figure 3 illustrates the probability density function and cumulative distribution function of water droplets in the duct (1-2).

The above illustrations show that conditions prevailing in the vertical duct (1-2), through which wet air flows in the upward direction, and in which there is a distributed mass source of water droplets with a constant unit value at the finite section (W-2) of this duct (1-2), allow for the transportation of the mass of water in a part equal to 0.5376 of its total flow rate generated at the section (W-2) in the upward direction with the air. This result was obtained by calculating, or reading for the abscissa of this point equal to $v = 4.56$ m/s, the value of the cumulative distribution function of the diameters equal to 0.5376. The remaining part of the mass flow rate of water droplets in this duct, with a relative concentration equal to (1-0.5376) will be falling counter-currently, downward in this duct, i.e. along the sections (W-2) and (1-W). It should be remembered that the section (1-W) of the vertical duct (1-2) is only 80 meters long (compared to the length of the duct (1-2) equal to 600 meters) and, along its entire length, there is no internal distributed mass source of water droplets from the condensation of water vapor contained in the flow rate of the flowing air. Due to the complexity of the effects of taking into consideration the distribution of diameters of water droplets in the duct (1-2), their influence on determining the static pressure loss in this duct will be discussed in the further part of this research paper. In the other ducts of the discharge pathway, the mass flow rate of water droplets is entirely carried up to the surface, co-currently with the air, but the slip value between the flowing phases depends on the distribution of droplet diameters.

**MASS OF WATER IN ANALYZED DUCT (1-2) AND RESULTANT LOSS OF STATIC PRESSURE**

The duct (1-2) is 600 meters long and is the first vertical duct on the discussed pathway of the air carried from underground mine workings up to the surface. In (Ptaszyński et al., 2018), the parameters of the ducts forming this pathway have been described in detail. In the further part of this paper, the influence of the distribution of water droplet diameters on the procedure and the value of water mass determination, as well as the above-mentioned pressure loss in the vertical duct (1-2) will be determined. The wet air, flowing in the upward direction through the vertical duct (1-2) and getting cooled down in the place denoted as W, reaches the state of water vapor saturation.
Flowing further, towards the terminal node 2, it becomes haze water which occurs in the form of droplets. At the
section 1-W, which is 80 meters long, there is no condensation of water vapor, so there is no internal water mass 
source. At the section W-2 of this duct, which is 520 meters long, there is a continuous water mass source on the
discharge pathway, with a unit constant mass output equal to \( m_{w,j} = 0.001240 \, \text{kg/(s·m)} \), defined in (Ptaszyński et al.,
2018). Throughout the entire length of the duct (1-2), the mean velocity of wet air in the cross-section changes 
slightly. For the section W-2, the arithmetic mean value of the velocity is \( v_{wp} = 4.56 \, \text{m/s} \). The unit flow rate of the 
condensed water \( m_{w,j} \) at the section W-2 consists of the sum of two components. The first component, \( m_{w,j,wn} \), stands 
for the unit mass flow rate of the resulting water droplets, whose diameters allow for their co-current flow with air in 
the upward direction, and the second component \( m_{w,j,op} \), is a unit mass flow rate of water droplets with diameters that 
cause their counter-current flow in this duct. This is reflected in Formula below:

\[
m_{w,j} = m_{w,j,wn} + m_{w,j,op}, \text{kg/(s·m)} \quad (20)
\]
in which: \( m_{w,j,wn} \) is a unit mass flow rate of water droplets in the duct W-2, discharged with the air, \( \text{kg/(s·m)} \), whereas \( m_{w,j,op} \) is a unit mass flow rate of water droplets falling counter-currently in the duct W-2, \( \text{kg/(s·m)} \). Using 
the value of the cumulative function of the predetermined distribution of droplet diameters read out from Figure 4 
(\( \delta = 2.5; \, d_M = 0.8 \cdot 10^{-3} \, \text{m} \)) for the mean air velocity in this duct, i.e. assuming that: 
[\( v_s(d_w) = v_{wp(W-2)} \), and \( \rho_p = 1.1902 \, \text{kg/m}^3 \)], it can be written that:

\[
m_{w,j,wn} = \phi_3(d_w)|_{v_{wp(W-2)} = 4.56 \text{m/s}} \cdot m_{w,j} = 0.5376 \cdot 0.001240 = 0.0006666 \, \text{kg/(s·m)} \quad (21)
\]

\[
m_{w,j,op} = \left[ 1 - \phi_3(d_w)|_{v_{wp(W-2)} = 4.56 \text{m/s}} \right] m_{w,j} = 0.4624 \cdot 0.001240 = 0.0005734 \, \text{kg/(s·m)} \quad (22)
\]

![Figure 4 - Determination of value \( \phi_3(d_w)|_{v_{wp(W-2)} = 4.56 \text{m/s}} \) for duct (1-2), where mean air velocity is \( v = 4.56 \, \text{m/s} \), from curve of cumulative mass concentration of water droplets](image)

The mass flow rate of water droplets discharged with the air in a co-current flow in the duct (1-2) at the section W-2, 
\( M_{W,j,wn(W-2)} \), is a function of distance “\( s \)” and it is determined by the following Formula:

\[
M_{W,j,wn(W-2)}(s) = \int_{s_w}^{s} m_{w,j,wn} \, ds = 0.0006666(s - s_w), \, \text{kg/s} \quad (23)
\]

where \( s_w \leq s \leq s_2 \).

The mass flow rate of water droplets from an internal water mass source distributed along the pathway, and falling in 
a counter-current flow in the duct (1-2) at the section W-2, \( M_{W,j,op(W-2)} \), is a function of distance “\( s \)” and it is 
determined by the following Formula:

\[
M_{W,j,op(W-2)}(s) = \int_{s_w}^{s} m_{w,j,op} \, ds = 0.0005734(s - s_w), \, \text{kg/s} \quad (24)
\]

where \( s_w \leq s \leq s_2 \).

The falling mass flow rate of water droplets at the section W-2 also falls at the section 1-W of the duct (1-2), but at 
this section there is no internal distributed water mass source from the condensation of water vapor contained in the 
air. At the section 1-W, the entire mass flow rate of the falling water from the section W-2 falls down, and this is a 
constant value over the whole length of the section 1-W. The mass flow rate of falling water droplets at the section 1-
W of the duct (1-2) can be expressed as:
where $s_1 \leq s \leq s_w$ and $H(s_w - s)$ is the Heaviside’s function.

In the whole vertical duct (1-2), the water mass carried with the air in the upward direction $m_{wy(1-2)}$, can be determined from the dependence (Ptaszyński, 2018a):

$$m_{wy(1-2)} = \int_{s_1}^{s_w} \left( \frac{v_{\infty}}{v_{p\infty}(s_1-w)} - v_{p\infty} \right) ds = \frac{0.0006666}{2(v_{p\infty}(s_1-w) - v_{p\infty})} (s_2 - s_w)^2, \text{ kg}$$  \hspace{1cm} (26)

where: $v_{\infty}$ is the velocity of free fall of water droplets carried with the air in the duct (1-2) in the upward direction ($v_{p\infty(2)} > v_{\infty}$), m/s.

In the whole vertical duct (1-2), the water mass falling counter-currently in the air flowing in the upward direction $m_{op(1-2)}$, can be determined from the dependence:

$$m_{op(1-2)} = \int_{s_2}^{s_w} \left( \frac{v_{\infty}}{v_{p\infty}(W-w)} - v_{p\infty} \right) ds = \frac{0.0006666}{2(v_{p\infty}(W-w) - v_{p\infty})} (s_2 - s_w)^2, \text{ kg}$$  \hspace{1cm} (27)

where: $v_{\infty}$ is the velocity of free fall of water droplets falling counter-currently in the duct (1-2) in the air flowing in the upward direction, therefore $|v_{p\infty(W-1)}| < |v_{\infty}|$ and $|v_{p\infty(W-2)}| < |v_{\infty}|$, m/s. Both components of Formula (27) are positive values.

The flowing air in the vertical duct (1-2) contains the water mass from the droplets which are present there. The value of the water mass can be calculated from the following relationship:

$$m_{w(1-2)} = m_{wy(1-2)} + m_{op(1-2)} = \frac{0.0006666}{2(v_{p\infty}(W-w) - v_{p\infty})} (s_2 - s_w)^2 + \frac{0.0005734}{2(v_{p\infty}(W-w) - v_{p\infty})} (s_2 - s_w)^2$$  \hspace{1cm} (28)

In the above Formula, $v_{\infty}$ and $v_{\infty}$ are the velocities of free fall of water droplets in the air in m/s, with ($v_{\infty}$) for the droplets carried with the air in the upward direction in the co-current flow, and ($v_{\infty}$) for the droplets falling counter-currently in this duct. If these velocities are to be substituted by the mean values of these velocities from the corresponding intervals of the diameter values of the moving droplets, then the value of the water mass will be close to the value actually present in the duct (1-2). If we assume, in accordance with the example discussed in (Ptaszyński, 2018a), that: $s_1 = 0 \text{ m}$, $s_2 = 600 \text{ m}$, $s_w = 80 \text{ m}$, $v_{p\infty(W-2)} = 4.56 \text{ m/s}$, $v_{p\infty(W-1)} = 4.36 \text{ m/s}$, then the mass value of the water droplets present in this duct, calculated from Formula (28) is equal to:

$$m_{w(1-2)} = \frac{0.0006666}{2(4.56-v_{p\infty})} (600 - 80)^2 + \frac{0.0005734}{2(4.56-v_{p\infty})} (600 - 80)^2 + \frac{0.0005734}{2(4.56-v_{p\infty})} (600 - 80)^2$$  \hspace{1cm} (29)

This Formula clearly demonstrates that the mass value will depend on the assumed values of $v_{\infty}$ and $v_{\infty}$. If these quantities were substituted by the arithmetic means of the values from relevant velocity intervals, i.e.: $v_{\infty} = 0.5(4.56 + 2.38) = 3.47 \text{ m/s}$ and $v_{\infty} = 0.5(4.56 + 12.07) = 8.31 \text{ m/s}$, then the mass of water droplets in the duct (1-2) would be equal to:

$$m_{w(1-2)} = \frac{0.0006666}{2(4.56-2.38)} (600 - 80)^2 + \frac{0.0005734}{2(8.31-4.56)} (600 - 80)^2 + \frac{0.0005734}{2(8.31-4.56)} (600 - 80)^2 = 41.34 + 6.04 + 290.71 = 338.09 \text{ kg}$$  \hspace{1cm} (30)

For that quantity to be determined even more accurately, the knowledge of the function $\Phi_3(d_w)$ should be used. Then, Formula (28) can be written in the following form:

$$m_{w(1-2)} = \frac{(s_2 - s_w)^2}{2} \int_0^{v_{\infty}} \left[ \phi_3(d_w) \left( \frac{v_{p\infty}(W-w)}{v_{p\infty}(W-w) - v_{p\infty})} dV_{w\infty} \right) + \phi_3(d_w) \left( \frac{v_{p\infty}(W-w)}{v_{p\infty}(W-w) - v_{p\infty})} dV_{w\infty} \right) + \phi_3(d_w) \left( \frac{v_{p\infty}(W-w)}{v_{p\infty}(W-w) - v_{p\infty})} dV_{w\infty} \right) + \phi_3(d_w) \left( \frac{v_{p\infty}(W-w)}{v_{p\infty}(W-w) - v_{p\infty})} dV_{w\infty} \right)$$  \hspace{1cm} (31)

It is possible to use the fact that the integral determined from the above functions is the summation of the surface areas determined by the width of the intervals $v_{\infty}$ and the increment of the function $\Delta\Phi_3(d_w)$. Then, the water mass can be determined from Formula (31) in the way presented below. The interval of velocities $v_{\infty}$, corresponding to the
droplets flowing co-currently with the air in the duct (1-2), is replaced by five intervals $v_{\infty i}$ for which the following values of $\Delta \phi_3(d_w)$, have been calculated:

$$\Delta \phi_3(d_w)_1 = \phi_3(v_{\infty_1} = 4.56, n = 25, d_M = 0.0008, \rho = 1.1902) - \phi_3(v_{\infty_1} = 3.56, n = 25, d_M = 0.0008, \rho = 1.1902) = 0.337154$$

$$v_{\infty_1} = 4.06 \text{ m/s}$$

$$\Delta \phi_3(d_w)_2 = \phi_3(v_{\infty_1} = 3.56, n = 25, d_M = 0.0008, \rho = 1.1902) - \phi_3(v_{\infty_1} = 2.56, n = 25, d_M = 0.0008, \rho = 1.1902) = 0.15834$$

$$v_{\infty_1} = 3.06 \text{ m/s}$$

$$\Delta \phi_3(d_w)_3 = \phi_3(v_{\infty_1} = 2.56, n = 25, d_M = 0.0008, \rho = 1.1902) - \phi_3(v_{\infty_1} = 1.56, n = 25, d_M = 0.0008, \rho = 1.1902) = 0.038494$$

$$v_{\infty_1} = 2.06 \text{ m/s}$$

$$\Delta \phi_3(d_w)_4 = \phi_3(v_{\infty_1} = 1.56, n = 25, d_M = 0.0008, \rho = 1.1902) - \phi_3(v_{\infty_1} = 0.56, n = 25, d_M = 0.0008, \rho = 1.1902) = 0.003586$$

$$v_{\infty_1} = 1.06 \text{ m/s}$$

$$\Delta \phi_3(d_w)_5 = \phi_3(v_{\infty_1} = 0.56, n = 25, d_M = 0.0008, \rho = 1.1902) - \phi_3(v_{\infty_1} = 0.00, n = 25, d_M = 0.0008, \rho = 1.1902) = 0.000022$$

$$v_{\infty_1} = 0.28 \text{ m/s}$$

Then, the first component of Formula (31) is equal to:

$$\frac{(s_2-s_1)^2}{2} \int_0^{\phi_3(d_w)} \frac{d \varphi_{\text{vir}}}{(v_{\text{spir}}(W-2) - v_{\infty})} = \sum_{i=1}^{5} \frac{\Delta \phi_3(d_w)_i}{(v_{\text{spir}}(W-2) - v_{\infty})} = \frac{(600-80)^2}{2} \left[ \frac{0.337154}{(4.56-4.06)} + \frac{0.15834}{(4.56-3.06)} + \frac{0.038494}{(4.56-2.06)} + \frac{0.003586}{(4.56-1.06)} + \frac{0.000022}{(4.56-0.28)} \right] = 133.497 \text{ kg}$$

(42)

The interval of velocities $v_{\infty i}$ corresponding to the droplets flowing counter-currently relative to the air flowing in the duct (1-2), is replaced by six intervals $v_{\infty i}$ for which the following values of $\Delta \phi_3(d_w)$, have been calculated:

$$\Delta \phi_3(d_w)_6 = \phi_3(v_{\infty_2} = 5.56, n = 25, d_M = 0.0008, \rho = 1.1902) - \phi_3(v_{\infty_2} = 4.56, n = 25, d_M = 0.0008, \rho = 1.1902) = 0.337306$$

$$v_{\infty_2} = 5.06 \text{ m/s}$$

$$\Delta \phi_3(d_w)_7 = \phi_3(v_{\infty_2} = 6.56, n = 25, d_M = 0.0008, \rho = 1.1902) - \phi_3(v_{\infty_2} = 5.56, n = 25, d_M = 0.0008, \rho = 1.1902) = 0.11647$$

$$v_{\infty_2} = 6.06 \text{ m/s}$$

$$\Delta \phi_3(d_w)_8 = \phi_3(v_{\infty_2} = 7.56, n = 25, d_M = 0.0008, \rho = 1.1902) - \phi_3(v_{\infty_2} = 6.56, n = 25, d_M = 0.0008, \rho = 1.1902) = 0.008566$$

$$v_{\infty_2} = 7.06 \text{ m/s}$$

$$\Delta \phi_3(d_w)_9 = \phi_3(v_{\infty_2} = 8.56, n = 25, d_M = 0.0008, \rho = 1.1902) - \phi_3(v_{\infty_2} = 7.56, n = 25, d_M = 0.0008, \rho = 1.1902) = 0.000064$$

$$v_{\infty_2} = 8.06 \text{ m/s}$$

$$\Delta \phi_3(d_w)_{10} = \phi_3(v_{\infty_2} = 9.56, n = 25, d_M = 0.0008, \rho = 1.1902) - \phi_3(v_{\infty_2} = 8.56, n = 25, d_M = 0.0008, \rho = 1.1902) = 1.5 \cdot 10^{-8}$$

(51)
\[ \nu_{\text{air}2} = 9.06 \text{ m/s} \]  

\[ \Delta \phi_3(d_w)_{11} = \phi_3(\nu_{\text{air}2}) = 12.07, n = 2.5, d_M = 0.0008, \rho = 1.1902 \]  

\[ \phi_3(\nu_{\text{air}2}) = 9.56, n = 2.5, d_M = 0.0008, \rho = 1.1902 \]  

\[ \nu_{\text{air}2} = 10.82 \text{ m/s} \]

The second and third components of Formula (31) are equal to:

\[
\frac{(s_2 - s_w)(s_w - s_1)}{2} \int \frac{\phi_3(d_w)}{\nu_{\text{air}2}^{12.07/m/s}} \frac{\phi_3(d_w) m_{w_j}}{(\nu_{\text{air}2} - v_{p_{ir}(1-2)})^4.56/m/s} d\nu_{\text{air}2} + \sum_{i=1}^{11} \frac{\Delta \phi_3(d_w)_{ij}}{(v_{\text{air}2} - v_{p_{ir}(i-2)})^2} \]

\[
\sum_{i=1}^{11} \frac{\Delta \phi_3(d_w)_{ij}}{(v_{\text{air}2} - v_{p_{ir}(i-2)})^2} \]  

\[
= 0.001240 \left[ \begin{array}{c} (600 - 80)(80 - 0) \cdot \left[ \begin{array}{c} 0.337306 \\ 0.11647 \\ 0.008566 \\ 1.5 \cdot 10^{-8} \\ 9.06 \cdot 4.36 \\ 2.7 \cdot 10^{-8} \\ 10.82 \cdot 4.36 \end{array} \right] \end{array} \right] + \left[ \begin{array}{c} 0.337306 \\ 0.11647 \\ 0.008566 \\ 1.5 \cdot 10^{-8} \\ 9.06 \cdot 4.36 \\ 2.7 \cdot 10^{-8} \\ 10.82 \cdot 4.36 \end{array} \right] = 0.001240 \cdot (23028.4 + 102171.1) = 28.55 + 126.69 = 155.24 \text{ kg} \]

The mass of water droplets present in the duct (1-2) is therefore:

\[ m_{w(1-2)} = m_{w(y-1-2)} + m_{w(p-1-2)} = 133.50 \text{ kg} + 155.24 \text{ kg} = 288.74 \text{ kg} \]

The determined mass of water droplets in the duct (1-2) allows for the specification of the static pressure loss caused by its presence. Using the dependence defined in (Ptaszyński, 2018b) and the values of the water mass defined in this paper, it is possible to determine the said pressure loss more accurately from the dependence:

\[ A_{\text{m}(1-2)} = \frac{m_{w(1-2)} g}{F_{(1-2)}} \]

where: \( F_{(1-2)} \) is the cross-sectional area of the duct (1-2), m\(^2\); \( g \) is the value of gravitational acceleration, m/s\(^2\). If we distinguish between the values of the flow rates of water moving upwards and falling in the duct (1-2), but we fail to take into account the detailed distribution of water droplet diameters, then by substituting the determined value of \( m_{w(1-2)} = 338.09 \text{ kg} \) to Formula (57), we obtain the value of a pressure loss equal to:

\[ A_{\text{m}(1-2)} = \frac{338.09 \cdot 9.81}{38.48} = 86.19 \text{ Pa} \]

If we take into account the determined mass distribution of water droplets moving upwards and falling as a function of its diameter, then the value of a pressure loss equals to:

\[ A_{\text{m}(1-2)} = \frac{288.74 \cdot 9.81}{38.48} = 73.61 \text{ Pa} \]

CONCLUSIONS

During the flow of wet air, along the pathway of its discharge from underground mine workings, this air gradually cools down and, as a consequence, condensation of water vapor usually occurs. The objective of these research studies was to determine the effects of this phenomenon in the form of static pressure losses generated by the main ventilation fans. The Author's earlier studies have proven that using the proposed procedures and calculations it is possible to determine the character of internal water mass sources resulting from the occurrence of this phenomenon. The measurement data and the results of calculations of the unit flow rate of the distributed internal mass source of water droplets in the duct (1-2) on its discharge pathway were used. Using this data, studies were carried out to determine the influence of distribution of water droplet diameters on the water mass in the duct and on the loss of static pressure. The Author's earlier studies have demonstrated that if the whole condensed water in the analyzed vertical duct (1-2) is carried up with the air from the duct, then:

1. In the case of a no-slip flow, i.e. the co-current flow of the gas phase, which is saturated wet air, and the liquid phase of dispersed water droplets at the same velocity, the mass of water droplets in the duct will be \( m_{w(1-2)} = 36.81 \text{ kg} \). This mass of water droplets in the duct (1-2) will cause a static pressure loss equal to \( A_{\text{m}(1-2)} = 10.36 \text{ Pa} \).
2. In the case of a co-current flow of both phases in the upward direction, taking into account the slip between the gas phase and the air droplets of the same diameter (at which droplets may be carried up), the water mass and the resulting pressure loss depends on the diameters of all the droplets, as demonstrated in Table 1.

Table 1 - Water mass and the resulting pressure loss depends on the diameters of all the droplets

<table>
<thead>
<tr>
<th>Droplets diameter $d_w$ (mm)</th>
<th>$m_{w(1-2)}$ (kg)</th>
<th>$A_{m(1-2)}$ (Pa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.1</td>
<td>56.32</td>
<td>15.86</td>
</tr>
<tr>
<td>0.2</td>
<td>72.16</td>
<td>20.32</td>
</tr>
<tr>
<td>0.3</td>
<td>92.03</td>
<td>25.91</td>
</tr>
<tr>
<td>0.4</td>
<td>119.85</td>
<td>33.75</td>
</tr>
<tr>
<td>0.5</td>
<td>163.36</td>
<td>46.00</td>
</tr>
<tr>
<td>0.6</td>
<td>243.17</td>
<td>68.47</td>
</tr>
<tr>
<td>0.7</td>
<td>441.53</td>
<td>124.32</td>
</tr>
<tr>
<td>0.8</td>
<td>1,833.97</td>
<td>516.40</td>
</tr>
</tbody>
</table>

Two constants were predetermined for the analysis of this specific duct in the recommended Rosin-Rammler distribution: $\delta = 2.5; \ d_M = 0.8 \times 10^{-3} \text{m}$. From the calculations carried out as part of this research work, it follows that:

3. In this duct (1-2), part of the mass flow rate of the internal water source will be carried with the air to a higher level. The mass content of this flow rate was determined at 0.5376, which means that 53.76% of the total mass flow rate of the water flowing into this duct as a result of water vapor condensation, is carried up to the surface. The remaining part of the total mass flow rate of the condensing water vapor, due to the larger diameters of the droplets forming this flow rate, will be counter-currently falling in the duct.

4. The occurrence of two water mass flow rates from the condensation of water vapor in the analyzed duct was taken into account, which allowed for:
   a) the performance of approximate calculation of water mass in the duct equal to $m_{w(1-2)} = 338.09 \text{kg}$ and the corresponding value of the pressure loss of $A_{m(1-2)} = 86.19 \text{Pa}$.
   b) the performance of more accurate calculations using the distribution of water droplet diameters in the flow rate moving in the upward and downward directions. As a result, the water mass in the studied duct is equal to $m_{w(1-2)} = 288.74 \text{kg}$, and the corresponding value of the static pressure loss is $A_{m(1-2)} = 73.61 \text{Pa}$.

5. The research has proven that the phenomenon involving the falling of a part of the mass of water droplets in the analyzed duct (1-2) has resulted in the increased value of a static pressure loss in this duct. If the whole condensed water in this duct was carried up by the air, then, taking into account the adopted distribution of water droplets in the interval of the diameters allowing droplets to be carried up with the air, the pressure loss would be $A_{m(1-2)} = 63.30 \text{Pa}$.

6. The falling mass flow rate of water droplets determined in this paper will cause a pressure loss equal to $A_{m(1-2)} = 39.58 \text{Pa}$, of which $A_{m(1-2)} = \frac{126.69 	imes 9.81}{38.48} = 32.30 \text{Pa}$ is the pressure loss at the section W-2 of the duct (1-2). The pressure loss caused by the falling of the determined mass of water droplets at the 80-meters-long section 1-W, will be equal to: $A_{m(1-2)} = \frac{28.55 	imes 9.81}{38.48} = 7.28 \text{Pa}$.

7. As it has already been mentioned earlier, the obtained results of static pressure loss and mass of water droplets in the studied duct (1-2), for a no-slip flow and a flow with a slip with a monodisperse liquid phase, are extreme values defining an interval in which the real values of these quantities may be contained. This interval may be defined by the values $A_{m(1-2)} = 10.36 \text{Pa}$ ($m_{w(1-2)} = 36.81 \text{kg}$) for a no-slip flow, $A_{m(1-2)} = 516.40 \text{Pa}$ ($m_{w(1-2)} = 1833.97 \text{kg}$) for a flow with a slip of the liquid phase in the form of water droplets with the same diameter equal to 0.8 mm >. The results of the research studies carried out in this work allowed for the exact calculation of these values in the analyzed duct. When the mass of the water droplets present in the duct is $m_{w(1-2)} = 288.74 \text{kg}$, then the loss of the static pressure in the duct (1-2) will be equal to $A_{m(1-2)} = 73.61 \text{Pa}$.

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PRESSURE DROP IN THE AIR STREAM WITH THE MONODISPERSE LIQUID PHASE COMING FROM THE CONDENSATION OF WATER VAPOR IN THE COOLED AIR FLOWING OUT OF THE INCLINED CONDUIT

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PRESSURE DROP IN THE AIR STREAM WITH THE MONODISPERSE LIQUID PHASE COMING FROM THE CONDENSATION OF WATER VAPOR IN THE COOLED AIR FLOWING OUT OF THE INCLINED CONDUIT

ABSTRACT

The paper presents methodology for determining water mass in ducts with fixed co-current two-phase slip flow. The dispersed liquid phase (water) is present in the air only as a result of distributed or local water mass sources (or its local outlets), formed by the phenomenon of water condensation, occurring on the pathway of wet air discharge from underground excavations in some sections of these ducts. Co-current air flow and monodisperse water stream were considered. For such cases, static pressure losses were determined, resulting from the force of gravity of the mass of water droplets present in the air stream in the duct, and they were compared to the values which would occur in a co-current no-slip flow.

KEYWORDS

two-phase slip flow, pressure loss in the flow, flow characteristics of the duct, water mass sources from condensation of water vapor

INTRODUCTION AND STUDY OBJECTIVE

Discharge of wet air to the surface from excavations of underground mines is carried out by suction main ventilation through vertical and inclined ducts. In order to force such flows, mechanical energy sources are used, called main ventilation fans. In the research study (Ptaszyński, Łuczak, Życzkowski, & Kuczera, 2018), an exemplary pathway of air discharge from the mine was presented, and the calculation procedure used there allowed to analyze thermodynamic transformations of the air occurring in these ducts, as well as to determine water mass sources and outlets (distributed along the pathway, or local ones) resulting from the condensation of water vapor contained in the air stream (Ptaszyński, 2016; Ptaszyński et al., 2018). Some of these values of water mass flow rate determined for such conditions depend on the time of the year (summer/winter). In the course of a year, the character of the source may change, which in one period may be positive (source) and in another negative (outlet). The only role of some ducts defined in (Ptaszyński et al., 2018) is to discharge a fluid flowing into them, as they have no water mass sources or outlets resulting from condensation of water vapor.

In the research papers (Ptaszyński, 2015; Ptaszyński et al., 2018) it was pointed out that the fixed flow of the two-phase medium in the transportation duct can be described by the so-called flow characteristics in which the value of $A_m$ is significant. For the pathway analyzed in (Ptaszyński et al., 2018), the values of $A_m$ were determined for each of the ducts making up the above-mentioned pathway for the co-current, no-slip flow of humid air and water droplets. Some of the studied ducts performed only the transportation function, but there were also those in which the water mass flow rate, or its volumetric concentration in the air, were a function of distance. These ducts, besides the transportation function, performed an additional function of the water mass source. For such ducts, the values of $A_m$ were determined as a line integral with respect to the length of the quotient of water mass (being a function of distance) and the actual velocity of the transported water droplets.

This paper aims to determine the value of $A_m$ for the ducts making up the air discharge pathway described in the research paper (Ptaszyński, 2018), but taking into account the fact that the dispersed phase, discharged to the surface with the air, formed by water droplets, is monodisperse.

These quantities were calculated taking into account that water droplets move in the duct in the same direction as the air (co-current flow), but at the speed which is different than the speed for the no-slip flow discussed in (Ptaszyński, 2018). This approach will make it possible to define discharge pathways for each duct, the values of $A_m$ closer to those that occur in reality, and compare them with the values of $A_m$ determined for no-slip flow. For this purpose, the results of the calculations obtained in the research papers (Ptaszyński, 2015; Ptaszyński, 2016; Ptaszyński, 2018; Ptaszyński et al., 2018) were used in this study, in particular the calculated water mass flow rate transported with air through individual sections of the air discharge pathway. In order to calculate the values of $A_m$ for individual ducts, it is necessary to take into account at what actual speed the water droplets (of a certain diameter) are carried away with air to the surface. The air discharge pathway, described in (Ptaszyński, 2018), will be defined below.

EXEMPLARY PATHWAY OF AIR DISCHARGE FROM THE MINE

The pathway of air discharge from the mine, discussed in (Ptaszyński, 2015; Ptaszyński, 2018) is illustrated in Figure 1. It consists of a two-level ventilation shaft, a ventilation pipe and a ventilation duct.
The ventilation shaft has a circular cross-section, a diameter of 7 m, cross-sectional area $F = 38.48 \text{ m}^2$ and a depth of 1,030 m. The bottom of the lower level with the air stream inlet is located at 1,023 m, and the bottom of the upper level is at 423 m. The length of the ventilation pipe is $l_{\text{con}} = 28 \text{ m}$, and its angle of inclination is $\alpha = 45^\circ$. The ventilation duct has a length of 45 meters, a rectangular cross-section measuring $4 \times 5 \text{ m}$ and its angle of inclination $\beta = 5^\circ$. For the example of air discharge from the mine, analyzed in (Ptaszyński, 2018), it was determined that:

1. Condensation of water vapor in the wet air stream, flowing in the vertical duct 1-2, occurs at the last 520 meters of its length. The unit flow rate of the distributed water mass source $\dot{m}_{w1-2}(s)$ at this 520-meters-long section is equal to $0.001240 \text{ kg/(s·m)}$ (Ptaszyński et al., 2018). The water mass flow rate, flowing concurrently in the analyzed duct, is defined by Formula (1), (Ptaszyński, 2018):

$$ \dot{M}_{w1-2}(s) = \int_{s_1}^{s} \dot{m}_{w1-2}(s) \, ds = 0.001240 \, (s - s_w), \text{ kg/s} $$

(1)

This quantity is the function of distance, but it is independent of the season since thermodynamic parameters of this wet air stream do not change throughout the year. The mean values of wet air velocity and density at the section W-2 are, respectively: $v_{w(W-2)} = 4.56 \text{ m/s}, \rho_{w(W-2)} = 1.1902 \text{ kg/m}^3$.

2. In the 6-meter-long duct 2-4, at the initial cross-section of this short duct, there is a water mass outlet with a unit mass flow rate $\dot{m}_{w2-4}(s) = -0.09977 \, \delta(s - s_2)$, where $\delta(s - s_2)$ is Dirac’s pseudo-function. In this duct, the mass flow rate $\dot{M}_{w2-4}(s)$ of the water flowing concurrently, is expressed by Formula (2), (Ptaszyński, 2018):

$$ \dot{M}_{w2-4}(s) = 0.6448 - 0.09977 \, \mathcal{H}(s - s_2), \text{ kg/s} $$

(2)

where $\mathcal{H}(s - s_2)$ is a Heaveside’a function. The quantity $\dot{M}_{w2-4}(s)$ is constant on the pathway 2-4, because the outlet is located at the inlet cross-section of the duct, and its value does not depend on the season, since thermodynamic parameters of the wet air stream do not change throughout the year. In any cross-section of this duct, the mean air velocity and its density are the same, that is, e.g., as at the terminal point: $v_4 = 9.055 \text{ m/s}, \rho_4 = 1.1394 \text{ kg/m}^3$.

3. Condensation of water vapor in the wet air stream, flowing in the vertical duct 4-5, occurs at its entire length. The unit flow rate of the distributed water mass source $\dot{m}_{w4-5}(s)$ is a constant value throughout this 400-meter-long section, and it is equal to $0.00185 \text{ kg/(s·m)}$, and the water mass flow rate $\dot{M}_{w4-5}(s)$, flowing concurrently in the considered duct, is expressed by Formula (3), (Ptaszyński, 2018):

$$ \dot{M}_{w4-5}(s) =  \dot{M}_{w2-4}(s) + \dot{m}_{w4-5}(s), \text{ kg/s} $$

(3)
The quantity $M_{w(4-5)}(s)$ is the function of distance but it is independent of the season, since thermodynamic parameters of this wet air stream do not change throughout the year. At this section of the duct, the mean air velocity is $v_{4(4-5)} = 9.238$ m/s, and the mean wet air density is $\rho_{w(4-5)} = 1.1163$ kg/m$^3$.

In the research paper (Ptaszyński, 2018), it was demonstrated that at the further sections of the air discharge pathway, the flow rate parameters of the medium depend on the time of the year. Therefore, two periods were considered: summers and winters. In the summer season, further ducts were described by the numbers of the points representing the thermodynamic states of the discharged stream, denoted as: 5-8, 8-11, 11-K. In the winter period, further ducts were denoted with the numbers of the points representing the thermodynamic states of the discharged stream: 5-10, 10-12, 12-K. Point K denoted the thermodynamic state of the stream at the inlet to the main ventilation fan.

4. In summer, in the 6-meter-long duct 5-8, there is a mixing of the two-phase stream of wet air and water droplets (water mist) with the wet air stream drawn through the shaft outset seal. In the analyzed case, external losses accounted for 10% of the volumetric output of the air discharged from the mine, expressed as dry air stream. As a result, a two-phase mixture was produced, but as demonstrated in the calculations included in (Ptaszyński et al., 2018), at the initial cross-section of this short duct, there was a water mass outlet with a unit mass flow rate equal to $m_{w(j)(5-8)}(s) = -0.04281 \delta(s - s_5)$. In this duct, the mass flow rate $M_{w(5-8)}(s)$ of the water flowing co-currently, is expressed by Formula (4), (Ptaszyński, 2018):

$$M_{w(5-8)}(s) = 1.28503 - 0.04281 \mathcal{H}(s - s_5), \text{ kg/s}$$

The quantity $M_{w(5-8)}(s)$ is constant on the pathway 5-8, because the outlet is located at the inlet cross-section of the duct, and it refers to the summer season. At any cross-section of this section, the mean air velocity is $v_8 = 10.173$ m/s, and the wet air density is $\rho_8 = 1.0936$ kg/m$^3$.

In winter, in the 6-meter-long duct 5-10, there is a mixing of the two-phase stream of wet air and water droplets (water mist) with the wet air stream drawn through the shaft outset seal, with the same proportions of their mixing. As a result, as demonstrated in (Ptaszyński et al., 2018), a two-phase mixture is produced, which indicates that at the initial cross-section of this duct, there is a local water mass source with a unit mass flow rate $m_{w(j)(5-10)}(s) = 0.29968 \delta(s - s_5)$. The mass flow rate $M_{w(5-10)}(s)$ of the water flowing co-currently in the examined duct, is expressed by Formula (5), (Ptaszyński, 2018):

$$M_{w(5-10)}(s) = 1.28503 + 0.29968 \mathcal{H}(s - s_5), \text{ kg/s}$$

The quantity $M_{w(5-10)}(s)$ is constant on the pathway 5-10, because the local water mass source is located at the inlet cross-section of the duct, and it refers to the winter season. At any cross-section of this section, the mean air velocity is $v_{10} = 10.447$ m/s, and the wet air density is $\rho_{10} = 1.0805$ kg/m$^3$.

5. In the ventilation pipe, the flowing two-phase stream gets cooled in summer. Therefore, on the pathway 8-11, there is a fixed distributed unit water mass source equal to $m_{w(j)(8-11)}(s) = 0.0306 \delta(s - s_8)$ kg/(s·m), (Ptaszyński, 2018). The water mass flow rate $M_{w(8-11)}(s)$, flowing co-currently with the air, can be calculated from Formula (6), (Ptaszyński, 2018):

$$M_{w(8-11)}(s) = \int_{s_8}^{s_{11}} [M_{w(8-11)}(s = s_8) \delta(s - s_8) + m_{w(j)(8-11)}(s)]ds = \int_{s_8}^{s_{11}} [1.24222 \delta(s - s_8) + 0.0306]ds = 1.24222 \mathcal{H}(s - s_8) + 0.0306 (s - s_8), \text{ kg/s}$$

The quantity $M_{w(8-11)}(s)$ is a function of distance and relates to the summer period. The mean air velocity at this section of the shaft is equal to $v_{8(8-11)} = 20.416$ m/s, and the mean wet air density at this section is $\rho_{w(8-11)} = 1.0871$ kg/m$^3$.

In winter, in the ventilation pipe, the flowing two-phase stream also gets cooled. Therefore, on the pathway 10-12, there is a fixed distributed unit water mass source, and its unit water mass flow rate is equal to $m_{w(j)(10-12)}(s) = 0.0275$ kg/(s·m) (Ptaszyński et al., 2018). The water mass flow rate $M_{w(10-12)}(s)$, flowing co-currently with the air, can be calculated from Formula (7), (Ptaszyński, 2018):

$$M_{w(10-12)}(s) = \int_{s_{10}}^{s_{12}} [M_{w(10-12)}(s = s_{10}) \delta(s - s_{10}) + m_{w(j)(10-12)}(s)]ds = \int_{s_{10}}^{s_{12}} [1.58471 \delta(s - s_{10}) + 0.0275]ds = 1.58471 \mathcal{H}(s - s_{10}) + 0.0275 (s - s_{10}), \text{ kg/s}$$
The quantity \( M_{w10-12}(s) \) is a function of distance and relates to the winter period. The mean air velocity at this section of the shaft is equal to \( v_{w10-12} = 20.234 \) m/s, and the mean wet air density at this section is \( \rho_{w10-12} = 1.0935 \) kg/m\(^3\).

6. In summer, the two-phase stream 11-K in the ventilation duct is not cooled. Therefore, the unit distributed water mass source from the condensation is \( m_{w11-K}(s) = 0 \). In winter, the situation is the same: \( m_{w12-K}(s) = 0 \). There is no internal water mass source in the duct, but the water from the condensation occurring in the preceding ducts is discharged through it. The water mass flow rate \( M_{w11-K}(s) \), \( M_{w12-K}(s) \), flowing in the air stream in the duct is constant over its entire length and the value depends on the period of the year. For the summer, (Ptaszyński, 2018):

\[
M_{w11-K}(s) = 2.0990 \times 10^4 \times H(s - s_{11}), \text{ kg/s}
\]  

(8)

The air velocity in the shaft duct in summer is equal to \( v_{w11-K} = 20.165 \) m/s, and the wet air density at this section is \( \rho_{w11-K} = 1.0805 \) kg/m\(^3\).

For winter, (Ptaszyński, 2018):

\[
M_{w12-K}(s) = 2.35471 \times 10^4 \times H(s - s_{12}), \text{ kg/s}
\]  

(9)

The air velocity in the shaft duct in winter is equal to \( v_{w12-K} = 19.629 \) m/s, and the wet air density at this section is \( \rho_{w12-K} = 1.0605 \) kg/m\(^3\).

Figures 2 and 3, quoted from the research paper (Ptaszyński, 2018), illustrate the change in the water mass flow rate from condensation of water vapor carried away to the surface along the air discharge pathway in summer and in winter. As it was mentioned above, these quantities do not depend on the velocity of the water droplets.

Figure 2 - Change in the mass flow rate of water droplets from water vapor condensation carried away along the air discharge pathway in summer (Ptaszyński, 2018)
WATER MASS FLOW RATE AT INDIVIDUAL SECTIONS OF AIR DISCHARGE PATHWAY

If the flow of water droplets was treated as a co-current flow upwards, in which the velocity of water droplets and the velocity of the air stream were different (a slip flow), then the water mass \( m_w \), in such a duct, e.g. (a-b), would be determined by Formula:

\[
m_w(a-b) = \int_{a}^{b} \frac{m_w(a-b)(s)}{v_f(a-b) - v_e} \, ds
\]  

(10)

where \( v_e \) is the so-called velocity of the free drop of water droplets in the air, which depends on the diameter of droplets in the duct (a-b).

It follows therefrom that the value of the water mass in the duct also depends on the diameter of the droplets, which in turn affects the value of \( A_m(a-b) \). Therefore, the occurrence of the so-called slip between two phases flowing co-currently was taken into account. The slip between phases is understood as the difference between the velocity of the gas phase and the velocity of water droplets (this paper considers the water from condensation).

Thus, for simplicity, it is predetermined that the droplets are spherical and they are not subject to secondary processes of merging or disintegration. The liquid phase is monodisperse, and the range of the analyzed droplet diameters takes into account physical conditions of their stability and allows their transport to the surface as a whole. Research studies for such conditions allow to estimate their effect on the value of \( A_m(a-b) \) mentioned above.

Water droplets with a spherical shape, depending on their diameter, have the so-called velocity of free fall, which is the maximum dropping speed of droplets in the ambient still air. This velocity in the environment where the presence of other droplets does not affect it with additional forces, can be calculated from the following dependence (Orzechowski, 1990):

\[
v_e = \sqrt{\frac{2g(\rho_w - \rho_p)\Delta d_w}{\rho_p C_x}}, \text{ m/s}
\]

(11)

Drag coefficient \( C_x \) can be calculated for a wide range of numbers \( Re_e \), from Kaskas equation (Orzechowski, 1990):

\[
C_x = \frac{24}{Re_e} + \frac{4}{\sqrt{Re_e}} + 0,4
\]

(12)

The value of Reynolds number is determined by the dependence:
If the above-mentioned flows in the studied ducts are carried out within the turbulent flow III (Orzechowski, 1990), then $C_l$ takes the value of 0.44.

For the range of the turbulent flow III, Formula (13) takes the form of Formula (14), (Orzechowski, 1990):

$$Re_{\infty} = \frac{v_\infty d_w}{\dot{o}^w}$$

(13)

$$v_\infty = 1.74 \sqrt{\frac{g (\rho_w - \rho_p)}{\rho_p}} \sqrt{d_w} \approx 1.74 \sqrt{\frac{g \rho_w}{\rho_p}} \sqrt{d_w}$$

(14)

In vertical sections of the discharge pathway of the air containing water droplets from the condensation of water vapor, the velocity of water droplets transported to the surface can be calculated from the dependence:

$$v_w(s) = v_p - v_\infty = v_p - 1.74 \sqrt{\frac{g \rho_w}{\rho_p}} \sqrt{d_w}, \text{ m/s}$$

(15)

In sections of the misty air discharge pathway, inclined at the $\alpha$ angle to the horizontal plane, the velocity of water droplets transported to the surface can be calculated from the dependence:

$$v_w(s) = v_p - \sin \alpha \cdot v_\infty = v_p - \sin \alpha \cdot 1.74 \sqrt{\frac{g \rho_w}{\rho_p}} \sqrt{d_w}, \text{ m/s}$$

(16)

It is important to determine the range of diameters of water droplets in the air that they can take. This can be determined by defining the maximum value of the droplet diameter that it can take. This value can be calculated from the dependence (17), (Orzechowski & Prywer, 2008):

$$d_{MAX} = 1.74 \sqrt{\frac{\sigma W_{eK}}{g \rho_w}}$$

(17)

The surface tension of water in the air $\sigma$ at 20°C, takes the value of 0.073 N/m. The value of water density is 998 kg/m$^3$. The so-called Critical Weber number $W_{eK}$ most often takes the value of 14 (Orzechowski & Prywer, 2008). By substituting the above-mentioned values into Formula (17), it was obtained: $d_{MAX} = 5.9 \times 10^{-3}$ m.

It can be concluded that during the co-current upward flow of air and water droplets, their maximum diameter may be 5.9 mm. Droplets with diameters larger than this value are unstable in such a flow, and they are subjected to secondary disintegration.

Taking into account that water droplets move to the surface at a relative velocity, equal to the difference between air velocity and free drop speed, Formulas for water mass $m_w$, kg, contained in the ducts transporting air to the surface of the mine on the analyzed discharge pathway, and the values which they determine, for the individual ducts of this pathway are as follows:

$$m_w(1-2) = \int_{s_1}^{s_2} \frac{m_w(s)}{v_{p(1-2)} - 1.74} dS = \frac{0.001240}{2} \frac{1}{v_{p(1-2)} - 1.74} \frac{g \rho_w}{\sqrt{\rho_p \rho_w}} \sqrt{d_w} \frac{(s_2 - s_0)^2}{2}$$

(18)

In the duct (W-2), the mean air velocity is $v_{p(w-2)} = 4.555$ m/s, and at this velocity, droplets of water with diameters satisfying the condition defined by Formula (19) will be carried away to the surface:

$$d_w < 8.33 \times 10^{-4} m$$

(19)

According to Formula (19), for the duct (W-2), water droplets with diameters $d_w < 8.33 \times 10^{-4} m$ will be carried away to the surface. Table 1 presents results of the calculations for the mass of water droplets $m_w$ in the duct (W-2), in kg, according to Formula (18), for droplet diameters $<1 \times 10^{-4} m + 8 \times 10^{-4} m^3$, for which the whole mass of condensed water vapor in this duct will be transported to the surface, and static pressure losses $A_{m(1-2)}$, in Pa, induced by this process, will be according to Formula (20), (Ptaszyński, 2018):

$$A_{m(1-2)} = \frac{m_w(1-2) g}{F_{(1-2)}}$$

(20)
<table>
<thead>
<tr>
<th>Droplets diameter $d_w$ (mm)</th>
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<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0.1</td>
<td>0.2</td>
<td>0.3</td>
<td>0.4</td>
<td>0.5</td>
<td>0.6</td>
<td>0.7</td>
</tr>
<tr>
<td>$m_{w(1-2)}$ (kg)</td>
<td>56.32</td>
<td>72.16</td>
<td>92.03</td>
<td>119.85</td>
<td>163.36</td>
<td>243.17</td>
<td>441.53</td>
</tr>
<tr>
<td>$A_{m(1-2)}$ (Pa)</td>
<td>15.86</td>
<td>20.32</td>
<td>25.91</td>
<td>33.75</td>
<td>46.00</td>
<td>68.47</td>
<td>124.32</td>
</tr>
</tbody>
</table>

In the duct (2-4), water droplets with diameters $d_w < 3.15 \cdot 10^{-3}$ m will be carried away to the surface. Table 2 presents results of the calculations for the mass of water droplets $m_w$ in the duct (2-4), in kg, according to Formula (21):

$$m_{w(2-4)} = \int_{s_2}^{s_4} \frac{M_{w(2-4)}(s)}{\rho_{w(2-4)}} \frac{s}{\sqrt{\rho_{w(2-4)}}} ds = \frac{0.54503(s_4 - s_2)}{\sqrt{\rho_{w(2-4)}}}$$

(21)

for droplet diameters of $<1 \cdot 10^{-4}$ m + $3.1 \cdot 10^{-3}$ m>, for which the whole mass of condensed water vapor in this duct will be transported to the surface, and static pressure losses $A_{m(2-4)}$, in Pa, induced by this process, will be according to Formula (22), (Ptaszyński, 2018):

$$A_{m(2-4)} = \frac{m_{w(2-4)} \rho_{w(2-4)}}{F_{(2-4)}}$$

(22)

Table 2 - Mass of water droplets $m_{w(2-4)}$ in the duct (2-4) for monodisperse liquid phase and pressure loss $A_{m(2-4)}$

<table>
<thead>
<tr>
<th>Droplets diameter $d_w$ (mm)</th>
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<tbody>
<tr>
<td></td>
<td>0.1</td>
<td>0.2</td>
<td>0.3</td>
<td>0.4</td>
<td>0.5</td>
<td>0.6</td>
<td>0.7</td>
<td>0.8</td>
</tr>
<tr>
<td>$m_{w(2-4)}$ (kg)</td>
<td>0.44</td>
<td>0.48</td>
<td>0.52</td>
<td>0.56</td>
<td>0.60</td>
<td>0.64</td>
<td>0.68</td>
<td>0.73</td>
</tr>
<tr>
<td>$A_{m(2-4)}$ (Pa)</td>
<td>0.12</td>
<td>0.14</td>
<td>0.15</td>
<td>0.16</td>
<td>0.17</td>
<td>0.18</td>
<td>0.19</td>
<td>0.20</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Droplets diameter $d_w$ (mm)</th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0.9</td>
<td>1.0</td>
<td>1.1</td>
<td>1.2</td>
<td>1.3</td>
<td>1.4</td>
<td>1.5</td>
<td>1.6</td>
</tr>
<tr>
<td>$m_{w(2-4)}$ (kg)</td>
<td>0.78</td>
<td>0.83</td>
<td>0.88</td>
<td>0.94</td>
<td>1.00</td>
<td>1.08</td>
<td>1.16</td>
<td>1.26</td>
</tr>
<tr>
<td>$A_{m(2-4)}$ (Pa)</td>
<td>0.22</td>
<td>0.23</td>
<td>0.25</td>
<td>0.26</td>
<td>0.28</td>
<td>0.30</td>
<td>0.33</td>
<td>0.35</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Droplets diameter $d_w$ (mm)</th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>1.7</td>
<td>1.8</td>
<td>1.9</td>
<td>2.0</td>
<td>2.1</td>
<td>2.2</td>
<td>2.3</td>
<td>2.4</td>
</tr>
<tr>
<td>$m_{w(2-4)}$ (kg)</td>
<td>1.36</td>
<td>1.48</td>
<td>1.62</td>
<td>1.78</td>
<td>1.97</td>
<td>2.20</td>
<td>2.48</td>
<td>2.84</td>
</tr>
<tr>
<td>$A_{m(2-4)}$ (Pa)</td>
<td>0.38</td>
<td>0.42</td>
<td>0.46</td>
<td>0.50</td>
<td>0.55</td>
<td>0.62</td>
<td>0.70</td>
<td>0.80</td>
</tr>
</tbody>
</table>
In the duct (4-5), water droplets with diameters $d_w < 3.21 \cdot 10^{-3}$ m will be carried away to the surface. Table 3 presents results of the calculations of the mass of water droplets in the duct (4-5), in kg, according to Formula (23):

$$m_{w(4-5)} = \int_{s_k}^{s_5} \frac{\bar{m}_{w(4-5)}(s)}{v_p\tau_{(4-5)}} - 1.74 \frac{\sqrt{g \rho_{w}}}{\rho_p \tau_{(4-5)}} \sqrt{d_w} ds$$

$$+ \frac{0.54503 (s_5 - s_k)}{v_p(4-5)} - 1.74 \frac{\sqrt{g \rho_{w}}}{\rho_p \tau_{(4-5)}} \sqrt{d_w} + \frac{1.74 \frac{\sqrt{g \rho_{w}}}{\rho_p \tau_{(4-5)}}}{2} \sqrt{d_w} (s_5 - s_4)^2$$

(23)

for droplet diameters of $<1 \cdot 10^{-4}$ m $\div 3.21 \cdot 10^{-3}$ m, for which the whole mass of condensed water vapor in this duct will be transported to the surface, and static pressure losses $A_{m(4-5)}$, in Pa, induced by this process, will be according to Formula (24), (Ptaszynski, 2018):

$$A_{m(4-5)} = \frac{m_{w(4-5)} g}{\rho_p (4-5)}$$

(24)

Table 3 - Mass of water droplets $m_{w(4-5)}$ in the duct (4-5) for monodisperse liquid phase and pressure loss $A_{m(4-5)}$

<table>
<thead>
<tr>
<th>Droplets diameter $d_w$ (mm)</th>
<th>0.1</th>
<th>0.2</th>
<th>0.3</th>
<th>0.4</th>
<th>0.5</th>
<th>0.6</th>
<th>0.7</th>
<th>0.8</th>
</tr>
</thead>
<tbody>
<tr>
<td>$m_{w(4-5)}$ (kg)</td>
<td>48.11</td>
<td>52.79</td>
<td>57.5</td>
<td>61.22</td>
<td>65.43</td>
<td>69.76</td>
<td>74.29</td>
<td>79.07</td>
</tr>
<tr>
<td>$A_{m(4-5)}$ (Pa)</td>
<td>13.55</td>
<td>14.86</td>
<td>16.06</td>
<td>17.24</td>
<td>18.42</td>
<td>19.64</td>
<td>20.92</td>
<td>22.26</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Droplets diameter $d_w$ (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.9</td>
</tr>
<tr>
<td>$m_{w(4-5)}$ (kg)</td>
</tr>
<tr>
<td>84.15</td>
</tr>
<tr>
<td>$A_{m(4-5)}$ (Pa)</td>
</tr>
<tr>
<td>23.69</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Droplets diameter $d_w$ (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.7</td>
</tr>
<tr>
<td>$m_{w(4-5)}$ (kg)</td>
</tr>
<tr>
<td>145.28</td>
</tr>
<tr>
<td>$A_{m(4-5)}$ (Pa)</td>
</tr>
<tr>
<td>40.91</td>
</tr>
</tbody>
</table>
In the duct (5-8), water droplets with all possible diameters will be carried away to the surface. Table 4 presents results of the calculations of the mass of water droplets in the duct (5-8) in the summer period, in kg, according to Formula (25):

\[
m_{w(5-8)} = \frac{M_{w(5-8)}(s)}{\sqrt{P_{ir(5-8)}}} \cdot ds = \frac{1.24222 (s_0-s_8)}{\sqrt{P_{ir(5-8)}}} \cdot \frac{\frac{g}{\rho}}{\sqrt{P_{ir(5-8)}}} \cdot \sqrt{d_w}
\]  

(25)

for droplet diameters of \(<1 \cdot 10^{-4} \text{ m} + 3.8 \cdot 10^{-3} \text{ m}>, for which the whole mass of condensed water vapor in this duct will be transported to the surface, and static pressure losses \(A_{m(5-8)}\), in Pa, induced by this process, will be for the summer period according to Formula (26), and for the winter period according to Formula (27):

\[
A_{m(5-8)} = \frac{m_{w(5-8)} \cdot g}{F_{(5-8)}}
\]  

(26)

\[
A_{m(5-10)} = \frac{m_{w(5-10)} \cdot g}{F_{(5-10)}}
\]  

(27)

For the winter period, the mass of water droplets in the duct (5-10) is calculated from Formula (28):

\[
m_{w(5-10)} = \frac{M_{w(5-10)}(s)}{\sqrt{P_{ir(5-10)}}} \cdot ds = \frac{1.58471 (s_{10}-s_8)}{\sqrt{P_{ir(5-10)}}} \cdot \frac{\frac{g}{\rho}}{\sqrt{P_{ir(5-10)}}} \cdot \sqrt{d_w}
\]  

(28)

for droplet diameters of \(<1 \cdot 10^{-4} \text{ m} + 3.97 \cdot 10^{-3} \text{ m}>, for which the whole mass of condensed water vapor in this duct will be transported to the surface.

Table 4 - Mass of water droplets \(m_{w(5-8)}, m_{w(5-10)}\) in the ducts (5-8), (5-10) for monodisperse liquid phase and pressure losses \(A_{m(5-8)}, A_{m(5-10)}\), in summer (winter) periods

<table>
<thead>
<tr>
<th>Droplets diameter (d_w) (mm)</th>
<th>0.1</th>
<th>0.2</th>
<th>0.3</th>
<th>0.4</th>
<th>0.5</th>
<th>0.6</th>
<th>0.7</th>
<th>0.8</th>
</tr>
</thead>
<tbody>
<tr>
<td>(m_{w(5-8)}) (kg)</td>
<td>0.87</td>
<td>0.95</td>
<td>1.02</td>
<td>1.08</td>
<td>1.15</td>
<td>1.21</td>
<td>1.28</td>
<td>1.35</td>
</tr>
<tr>
<td>(m_{w(5-10)}) (kg)</td>
<td>(1.08)</td>
<td>(1.17)</td>
<td>(1.25)</td>
<td>(1.33)</td>
<td>(1.41)</td>
<td>(1.49)</td>
<td>(1.57)</td>
<td>(1.65)</td>
</tr>
<tr>
<td>(A_{m(5-8)}) (Pa)</td>
<td>0.24</td>
<td>0.27</td>
<td>0.29</td>
<td>0.30</td>
<td>0.32</td>
<td>0.34</td>
<td>0.36</td>
<td>0.38</td>
</tr>
<tr>
<td>(A_{m(5-10)}) (Pa)</td>
<td>(0.30)</td>
<td>(0.33)</td>
<td>(0.35)</td>
<td>(0.37)</td>
<td>(0.40)</td>
<td>(0.42)</td>
<td>(0.44)</td>
<td>(0.46)</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Droplets diameter (d_w) (mm)</th>
<th>0.9</th>
<th>1.0</th>
<th>1.1</th>
<th>1.2</th>
<th>1.3</th>
<th>1.4</th>
<th>1.5</th>
<th>1.6</th>
</tr>
</thead>
<tbody>
<tr>
<td>(m_{w(5-8)}) (kg)</td>
<td>1.42</td>
<td>1.50</td>
<td>1.58</td>
<td>1.67</td>
<td>1.76</td>
<td>1.86</td>
<td>1.96</td>
<td>2.08</td>
</tr>
<tr>
<td>(m_{w(5-10)}) (kg)</td>
<td>(1.74)</td>
<td>(1.83)</td>
<td>(1.92)</td>
<td>(2.02)</td>
<td>(2.12)</td>
<td>(2.24)</td>
<td>(2.36)</td>
<td>(2.49)</td>
</tr>
<tr>
<td>(A_{m(5-8)}) (Pa)</td>
<td>0.40</td>
<td>0.42</td>
<td>0.44</td>
<td>0.47</td>
<td>0.50</td>
<td>0.52</td>
<td>0.55</td>
<td>0.59</td>
</tr>
<tr>
<td>(A_{m(5-10)}) (Pa)</td>
<td>(0.49)</td>
<td>(0.52)</td>
<td>(0.54)</td>
<td>(0.57)</td>
<td>(0.60)</td>
<td>(0.63)</td>
<td>(0.66)</td>
<td>(0.70)</td>
</tr>
</tbody>
</table>
In the ventilation pipe (8-11) in summer, and (10-12) in winter, water droplets of all possible diameters will be carried away to the surface. This is due to the fact that taking into account the angle of inclination of the pipe to the horizontal surface equal to $\alpha = 45^\circ$, and multiplying the mean air velocity in the ventilation pipe by sine, a vertical component of the air velocity was obtained, the value of which in summer and in winter is greater than 14 m/s. At this value of the component of air velocity, all water droplets of the diameters $<0.5 \text{ mm}$ in the pipe will be carried away to the ventilation duct. Table 5 presents results of the calculations of the mass of water droplets in the duct (8-11) in the summer period, and in (10-12) in the winter period, in kg, according to Formulas (29) and (30):

$$m_{w(8-11)} = \int_{s_{9}}^{s_{11}} \frac{M_{w(8-11)}(d)}{v_{pr(8-11)} - 1.74 \sin \alpha} \sqrt{\frac{\rho_{w}}{\rho_{pr(8-11)}}} \, ds = \int_{s_{9}}^{s_{11}} \frac{1.24222 \lambda (x_{5} - x_{9}) + 0.0306 (x_{5} - x_{9})^{2}}{v_{pr(8-11)} - 1.74 \sin \alpha} \sqrt{\frac{\rho_{w}}{\rho_{pr(8-11)}}} \, ds =$$

$$= \int_{s_{10}}^{s_{11}} \frac{1.58471 \lambda (x_{5} - x_{9}) + 0.0275 (x_{5} - x_{9})^{2}}{v_{pr(10-12)} - 1.74 \sin \alpha} \sqrt{\frac{\rho_{w}}{\rho_{pr(10-12)}}} \, ds =$$

for droplet diameters of $<1 \cdot 10^{-4}$ m + 5.9 $\cdot 10^{-3}$ m, for which the whole mass of condensed water vapor in this duct will be transported to the surface, and static pressure losses $A_{m(8-11)}$ in summer and $A_{m(10-12)}$ in winter, in Pa, induced by this process, will be according to Formulas (31), (32) for the summer and winter periods, respectively (Ptaszyński, 2018):
Table 5 - Mass of water droplets $m_{w(8-11)}$, $(m_{w(10-12)})$ in the ducts (8-11), (10-12) for monodisperse liquid phase and pressure losses $A_{m(8-11)}$, $(A_{m(10-12)})$, in summer (winter) periods

<table>
<thead>
<tr>
<th>$m_{w(8/10-11/12)}$ (kg)</th>
<th>Droplets diameter $d_w$ (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>$A_{m(8/10-11/12)}$ (Pa)</td>
<td>0.1</td>
</tr>
<tr>
<td>$m_{w(8-11)}$ (m_{w(10-12)})</td>
<td>2.43 (2.89)</td>
</tr>
<tr>
<td>$A_{m(8-11)}$ $(A_{m(10-12)})$</td>
<td>0.86 (1.02)</td>
</tr>
<tr>
<td>$m_{w(8/10-11/12)}$ (kg)</td>
<td>0.9</td>
</tr>
<tr>
<td>$A_{m(8/10-11/12)}$ (Pa)</td>
<td>2.77 (3.29)</td>
</tr>
<tr>
<td>$m_{w(8-11)}$ (m_{w(10-12)})</td>
<td>0.98 (1.16)</td>
</tr>
<tr>
<td>$A_{m(8-11)}$ $(A_{m(10-12)})$</td>
<td>1.7</td>
</tr>
<tr>
<td>$m_{w(8/10-11/12)}$ (kg)</td>
<td>3.00 (3.57)</td>
</tr>
<tr>
<td>$A_{m(8/10-11/12)}$ (Pa)</td>
<td>1.06 (1.26)</td>
</tr>
<tr>
<td>$m_{w(8-11)}$ (m_{w(10-12)})</td>
<td>2.5</td>
</tr>
<tr>
<td>$A_{m(8-11)}$ $(A_{m(10-12)})$</td>
<td>3.21 (3.83)</td>
</tr>
<tr>
<td>$m_{w(8/10-11/12)}$ (kg)</td>
<td>1.13 (1.35)</td>
</tr>
<tr>
<td>$A_{m(8/10-11/12)}$ (Pa)</td>
<td>3.3</td>
</tr>
</tbody>
</table>
In the ventilation duct (11-K) in summer and in (12-K) in winter, water droplets of all possible diameters will be present, but only a part of them will be carried away to the surface. This is due to the fact that taking into account the angle of inclination of the ventilation pipe to the horizontal surface equal to \( \beta = 5^\circ \), and multiplying the mean air velocity in the ventilation pipe by \( \sin \beta \), a vertical component of the air velocity was obtained, the value of which in summer is 1.76 m/s, and in winter it is 1.71 m/s. Therefore, a significant number of the water droplets will drop to the bottom of the ventilation duct. However, all of those with diameter ranging \(< 0.5 \text{ mm} >\) will be present in the air stream in the ventilation duct. Table 6 demonstrates results of the calculations of the mass of water droplets in the duct (11-K) in summer and in (12-K) in winter, in kg, according to Formulas (33) and (34):

\[
m_{w(11-K)} = \int_{s_{11}}^{s_{11}} \frac{m_{w(11-K)}(s)}{p_{t(11-K)}} \cdot 1.74 \sin \beta \sqrt{\frac{\rho_{w}}{p_{t(11-K)}}} ds = \int_{s_{11}}^{s_{11}} \frac{0.20990 \frac{H(s-s_{11})}{p_{t(11-K)}}}{2.09990 (s-k-s_{11})} \sqrt{\frac{\rho_{w}}{p_{t(11-K)}}} ds =
\]

\[
m_{w(12-K)} = \int_{s_{12}}^{s_{12}} \frac{m_{w(12-K)}(s)}{p_{t(12-K)}} \cdot 1.74 \sin \beta \sqrt{\frac{\rho_{w}}{p_{t(12-K)}}} ds = \int_{s_{12}}^{s_{12}} \frac{2.35471 \frac{H(s-s_{12})}{p_{t(12-K)}}}{19.629 (s-k-s_{12})} \sqrt{\frac{\rho_{w}}{p_{t(12-K)}}} ds =
\]

for droplet diameters \(< 1 \cdot 10^{-4} \text{ m} < 5.9 \cdot 10^{-3} \text{ m} >\) and static pressure losses \( A_{m(11-K)} \) in summer and \( A_{m(12-K)} \) in winter, in Pa, induced by this process, will be according to Formulas (35), (36), for the summer and winter periods, respectively (Ptaszyński, 2018):

\[
A_{m(11-K)} = \frac{m_{w(11-K)}}{F(11-K)} \sin \beta
\]

\[
A_{m(12-K)} = \frac{m_{w(12-K)}}{F(12-K)} \sin \beta
\]

Table 6 - Mass of water droplets \( m_{w(11-K)}, (m_{w(12-K)}) \) in the ducts (8-11), (10-12) for monodisperse liquid phase and pressure losses \( A_{m(11-K)}, (A_{m(12-K)}) \), in summer (winter) periods
CONCLUSIONS

1. The procedure and calculation Formulas presented in the article allow to determine the value of $A_{m_{K}}$ for the analyzed ducts with two-phase flow of wet air and water droplets, derived exclusively from condensation of water vapor contained in the air discharged from the mine to the surface. This value is necessary to determine flow characteristics of the duct in which there is a predetermined co-current two-phase flow, especially if there is a difference in the velocities of the gas phase and the liquid phase.

2. The study considered fixed co-current two-phase flow of air and monodisperse water, the whole of which was carried away to the surface.

3. The calculations were carried out in the following ducts:
   a) vertical, in which the water mass flow rate was constant over their entire length (transportation ducts),
   b) vertical, in which there were mass sources of the liquid phase distributed over the length of the ducts along the pathway. These sources were related to condensation of water vapor contained in the air and, as a result, the mass flow rate of the liquid phase changed on the pathway,
   c) vertical, in which there were mass sources of the liquid phase, local and distributed, associated with condensation of water vapor contained in the flowing air,
   d) inclined to the horizontal plane at a certain angle, in which the water mass flow rate was constant over its entire length,
   e) inclined to the horizontal plane at a certain angle, in which there was a mass source of the liquid phase distributed over its length. The source was a result of condensation of water vapor and resulted in changes of the water mass flow along the pathway in the duct,
In each of the above cases, calculation Formulas for the water mass contained in the air in the analyzed duct and for the quantity $A_m$ were different and were given in this research paper.

4. In the studied flows, the values of water mass and pressure loss $A_m$ increase with the increase in diameters of water droplets transported co-currently with air in the monodisperse stream, in the range of possible diameters of water droplets.

5. Pressure loss $A_m$ in the air discharge pathway, expressed in Pa, is the sum of the calculated pressure losses for individual ducts of this pathway, and its value (as demonstrated by the calculations for the occurrence of the liquid phase on the discharge pathway only due to the phenomenon of water vapor condensation) during the transportation of the whole condensed water to the surface may be similar to, or greater than, pressure increase of the main ventilation fans used in underground mines. This indicates that pressure loss defined in such a way has a great influence on the air flow conditions in the whole network.

6. The greatest contribution to the total loss of pressure $A_m$ on the discharge pathway are made by those long sections of vertical ducts in which water mass in the air is the largest and the air velocity is lower, and in which there is a water mass source distributed along the pathway. In the duct (1-2), for water droplets with a diameter of $d_w = 0.8 \text{ mm}$, the loss of static pressure will be about 520 Pa, and in the duct (4-5), for water droplets with a diameter of $d_w = 3.2 \text{ mm}$ (the maximal, at which they would be carried away with the air), the pressure loss will be about 5,130 Pa. The sum of these two values exceeds the values of pressure increase of the main ventilation fans currently used in Polish underground mines.

7. For comparison of the defined values of the total pressure loss $A_m$ on the pathway of air discharge from the mine, it should be remembered that for the no-slip conditions for the same water masses present in the analyzed ducts and transported to the surface, derived exclusively from condensation of water vapor on its flow pathway, the values of the total pressure loss amounted to several tens of Pa. This comparison clearly demonstrates the scope of possible change for this value.

8. The value of $A_m$ determined from the research studies for a specific discharge pathway may be essential for the transportation of wet air from underground mine workings to the surface.

**FUNDING**

This study was supported by the AGH University of Science and Technology statutory research No. 11.11.100.920

**NOMENCLATURE**

- $C_s$ - drag coefficient,
- $d_w$ - droplet diameter, m,
- $W_{e_{KR}}$ - Critical Weber number,
- $\rho_p$ - air density, kg/m$^3$,
- $\rho_w$ - water density, kg/m$^3$,
- $\sigma$ - surface tension of water in the air, N/m,

**REFERENCES**


Ptaszyński, B. (2018). The minimum pressure drop caused by the condensation of water vapor in the cooled air flowing in the inclined conduit. 25$^{th}$ World Mining Congress, June 19-22, 2018, Astana.


THE MINIMUM PRESSURE DROP CAUSED BY THE CONDENSATION OF WATER VAPOR IN THE COOLED AIR FLOWING IN THE INCLINED CONDUIT

B. Ptaszyński

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THE MINIMUM PRESSURE DROP CAUSED BY THE CONDENSATION OF WATER VAPOR IN THE COOLED AIR FLOWING IN THE INCLINED CONDUIT

ABSTRACT

The paper presents methodology for determining water mass and pressure drop in ducts with fixed co-current and no-slip two-phase flow. The dispersed liquid phase (water) is present in the air as a result of wet air from underground excavations being distributed in some sections of a discharge pathway, or local water mass sources (or its local outlets) formed by the natural phenomenon of water vapor condensation. For such cases, static pressure losses were determined, which resulted from the force of gravity of the mass of water droplets occurring in the air stream in the duct.

KEYWORDS

two-phase no-slip flow, pressure loss, flow characteristics of a duct, water mass sources from water vapor condensation

INTRODUCTION AND STUDY OBJECTIVE

Wet air discharged to the surface from deep excavations of underground mines, flows through vertical and inclined ducts. In order to force such flows, mechanical energy sources are used, called main ventilation fans. They usually work in a suction mode, producing static pressure by air flow, which increases its value along the pathway traveled by the flowing air. In the research study (Ptaszyński, 2016), the author presents the methods of using the Mollier diagram to determine thermodynamic processes of the wet air stream occurring at substantially varying static pressure. This allowed to determine thermodynamic changes of the air along its discharge pathway from a mine (Ptaszyński, Łuczak, Życzkowski, & Kuczer, 2018). Therefore, for the known parameters of thermodynamic air streams (defined by standard ventilation measurements in underground excavations), it is possible to determine whether there exist sections of air discharge ducts where the phenomenon of water vapor condensation occurs, and what the mass flow rate of water is, if referred to the unit of length in a given duct. From the considerations presented in the paper (Ptaszyński et al., 2018), it can be concluded that positive water mass sources from water vapor condensation, which are distributed sources on the finite sections of the ducts, as well as locally located water mass sources or their outlets, resulting from the condensation of water or evaporation of water droplets from the mixing of wet air streams and haze air streams containing water mist, may occur on the pathways of air discharge through the ducts. Some of these values of water mass flow rate depend on the time of the year (summer/winter). In the course of a year, the character of the source may change, which in one period may be positive (source) and in another negative (outlet). The author also identified those ducts on the analyzed air pathway, whose only function was to carry a fluid, i.e. those with no source or outlet of water mass from the condensation of water vapor.

In the discussed example, the output values of water mass sources from water vapor condensation, on the analyzed pathway of air discharge to the surface, were determined for the extreme seasons. The flow of air through a duct as a single-phase fluid is subject to considerations in mining aerology. The concepts of resistance (specific or equivalent) of a duct, its characteristics, and loss of pressure are utilized then. The fixed flow of the two-phase medium in the transportation duct makes it difficult to apply some of the previously mentioned terms. This mainly concerns the characteristics of the duct, its graphical representation and the method of determining resistance of the duct (Ptaszyński, 2015). The notion of flow characteristics of the duct has been introduced and specified, where a fixed, co-current and no-slip two-phase flow occurs. This notion contains the value of $A_{m}$, which is the pressure that the force of gravity of water droplets contained in the flowing air exerted on the plane perpendicular to the velocity vector. Due to the fact that on the discharge pathway of the wet from the mine, the phenomenon of water vapor condensation occurs, water droplets are transported with the air to the surface.

The objective of this research study is to determine the value of the above-mentioned $A_{m}$, occurring in flow characteristics of ducts, through which a two-phase medium, composed of air and water, flows along a discharge pathway from the mine.

The example of a pathway and the results of calculations presented in (Ptaszyński, 2015) were used in this study.

FLOW CHARACTERISTICS OF THE DUCT TRANSPORTING TWO-PHASE MEDIUM

With the fixed two-phase flow of air and water through a vertical duct in the upward direction, where the phases move in the same direction and with the same velocity (no velocity of the slip), it is insufficient to use the term of the duct resistance in order to correctly determine the mechanical energy loss in it (Ptaszyński, 2015). It is
important in such a case to use the concept of flow characteristics of the duct which, according to (Ptaszyński, 2015), can be presented in the form of the value \( w^* \) being the following function:

\[
\begin{align*}
w^* &= f(A_{m}, R_{m}, \dot{m}) \quad \text{N/m}^2
\end{align*}
\]  

(1)

The characteristics of flow can be represented by the following equation:

\[
\begin{align*}
w^* &= A + R_{m} \cdot \dot{m}^2 \quad \text{N/m}^2
\end{align*}
\]  

(2)

In case of a one-phase air flow, the value of \( A = 0 \), and the value of \( R_{m} \) represents the specific resistance of the duct with a fully developed turbulent flow in it. If the characteristics referred to the flow of water through this duct (one-phase medium), then the value of \( A \) would be the pressure that the water exerts on the plane perpendicular to the direction of the flow. This amount does not depend on the mass output of the water flowing in it. In this case, the value of \( R_{m} \) is also the specific resistance of the duct, but with the turbulent flow of water through it. Regarding the flow characteristics of this duct for a two-phase medium with constant mass concentration flowing through it, equation (2) still defines it correctly, but the value of \( A = A_{m} \) represents the pressure exerted on the plane perpendicular to the flow direction of water contained in this duct. In this case, however, the value depends on the mass output \( \dot{m} \) of the mixture with the specific mass concentration of water, and it can be calculated from the following dependence (Ptaszyński, 2015):

\[
\begin{align*}
A_{m} = \dot{w}_{m=0} = (z_2 - z_1) \rho_w g c \quad \text{N/m}^2
\end{align*}
\]  

(3)

It is similar with the a value of \( R_{m} \), which also depends on the mass output of the mixture \( \dot{m} \) and on the specific mass concentration of water in it.

As it was already mentioned, the objective of this research study is to determine the value of \( w_{m=0} = A_{m} \) for the ducts located in the air discharge pathway and therefore also for those in which there are internal mass sources of the heavier phase (water) supply from the condensation of water vapor in the cooling wet air stream, and for those in which the process of mixing the wet air streams occurs.

For this purpose, ventilation and thermodynamic data on the air discharged to the surface from the underground mine, obtained in the paper (Ptaszyński et al., 2018), were used to determine:

- location and character (distributed, local) of the occurrence of the phenomenon of water vapor condensation on the air discharge pathway,
- mass output of internal water sources from the condensation of water vapor on the air discharge pathway,
- mass output of water transported with the air through individual sections of the air discharge pathway.

Using such values, the article specifies the water mass contained in the air and discharged with the air, with no slip, through the ducts, to the surface. The pressure \( A_{m} \) was also calculated, with which the weight of this water affects the air at a given section of the duct being a part of the air discharge pathway, and it was indicated what this value depends on. These values were specified for a predetermined flow of the analyzed medium at each section of the analyzed flow discharge pathway.

**EXEMPLARY PATHWAY OF AIR DISCHARGE FROM THE MINE**

Figure 1 illustrates a discussed pathway of air discharge from the mine. It consists of a two-level ventilation shaft, a ventilation pipe and a ventilation duct.

![Figure 1 - Pathway of air discharge from the mine with marked points assigned to the corresponding cross-section of the flowing air stream (Ptaszyński et al., 2018)](image-url)
The ventilation shaft has a circular cross-section, a diameter of 7 m, cross-sectional area \( F = 38.48 \text{m}^2 \) and a depth of 1030 m. The bottom of the lower level with the air stream inlet is located at 1023 m, and the diameter of the ventilation pipe is 5 m. The length of the ventilation pipe is \( l_{\text{pipe}} = 28 \) m, and its angle of inclination is \( \alpha = 45^\circ \). The ventilation duct has a length of 45 m, a rectangular cross-section measuring \( 4 \times 5 \) m and its angle of inclination \( \beta = 5^\circ \). The results of measurements and calculations of ventilation as well as thermodynamic parameters are as follows (Ptaszyński, 2016; Ptaszyński et al., 2018):

- Air stream flowing in at the 1023 level: barometric pressure \( p_1 = 110,000 \) Pa; dry-bulb temperature \( t_{1a} = 29 \) °C; wet-bulb temperature \( t_{w1} = 28.5 \) °C; specific humidity of wet air \( X_1 = 0.02258 \text{kg/kg} \); wet air density \( \rho_1 = 1.2517 \text{kg/m}^3 \); volumetric flow rate of wet air flowing into the shaft \( \dot{V}_1 = 166.67 \text{m}^3/\text{s} \); wet air mass flow rate \( \dot{m}_1 = 208.62 \text{kg/s} \); dry air mass flow rate \( \dot{m}_{1a} = 204.01 \text{kg/s} \).

- Air stream flowing in at the 423 level: barometric pressure \( p_2 = 98,500 \) Pa; dry-bulb temperature \( t_{1a} = 26 \) °C; wet-bulb temperature \( t_{w2} = 25 \) °C; specific humidity of wet air \( X_2 = 0.02024 \text{kg/kg} \); wet air density \( \rho_2 = 1.1336 \text{kg/m}^3 \); volumetric flow rate of wet air flowing into the shaft \( \dot{V}_2 = 166.67 \text{m}^3/\text{s} \); wet air mass flow rate \( \dot{m}_2 = 188.93 \text{kg/s} \); dry air mass flow rate \( \dot{m}_{2a} = 185.18 \text{kg/s} \).

- The outside air is sucked into the shaft through the sealed-off outset of the shaft. The barometric pressure at the surface is \( p_T = 98,600 \) Pa. The conducted research took into account air parameters characteristic for winter and summer periods, which are as follows:
  - In winter: dry-bulb temperature \( t_1 = -20 \) °C; wet-bulb temperature \( t_{w1} = -20 \) °C; specific humidity of wet air \( X_1 = 0.0079 \text{kg/kg} \); wet air density \( \rho_1 = 1.35645 \text{kg/m}^3 \); volumetric flow rate of external air intake accounts for 10% of the volumetric flow rate of the air discharged through the shaft (section 4-5), i.e. \( \dot{V}_1 = 0.1 (\dot{V}_1 + \dot{V}_3) \).
  - In summer: dry-bulb temperature \( t_1 = 24 \) °C; wet-bulb temperature \( t_{w1} = 21 \) °C; specific humidity of wet air \( X_1 = 0.01484 \text{kg/kg} \); wet air density \( \rho_1 = 1.14597 \text{kg/m}^3 \); volumetric flow rate of external air intake accounts for 10% of the volumetric flow rate of the air discharged through the shaft (section 4-5), i.e. \( \dot{V}_1 = 0.1 (\dot{V}_1 + \dot{V}_3) \).

Additional measurement results at the end of sections (1-2 and 4-5), at the cross section of the shaft below the air inlet at a higher level (cross-section 2) as well as below the ventilation pipe inlet (cross-section 5) are as follows:

- Barometric pressure \( p_2 = p_3 = 98,500 \) Pa; dry-bulb and wet-bulb temperatures \( t_{2a} = t_{w2} = 24 \) °C; wet air density \( \rho_2 = 1.14332 \text{kg/m}^3 \); volumetric flow rate of wet air \( \dot{V}_2 = \dot{m}_2/\rho_2 = 182.469 \) m³/s; barometric pressure \( p_3 = p_4 = p_6 = 93,600 \) Pa; dry-bulb and wet-bulb temperatures \( t_{w3} = t_{w5} = 22 \) °C; wet air density \( \rho_3 = 1.0941 \text{kg/m}^3 \); volumetric flow rate of wet air \( \dot{V}_3 = (\dot{m}_2 + \dot{m}_3)/\rho_3 = 363.358 \) m³/s.

If the airflow at the cross-sections 2 and 5 was a single-phase one (saturated wet air), then the average air velocity in those cross-sections would be equal to \( v_2 = V_2/F = 4.74 \text{m/s} \) and \( v_3 = V_3/F = 9.44 \text{m/s} \).

The measurements recorded at the cross-sections of the shafts 2 and 5 do not determine unambiguously whether there is a single-phase flow of saturated wet air or a two-phase one, where in addition to the vapor-saturated air stream constituting the carrier gas phase, there is a stream of fluid produced in the process of water vapor condensation in the air pathway. Ventilation measuring instruments commonly used in mining are not capable of solving this problem. The method involving the Mollier diagram presented in the research paper (Ptaszyński, 2016) allowed to determine locations of water mass sources or water mass outlets formed from water vapor condensation in the air stream, and to specify them quantitatively. It was found that in the discussed pathway of air discharge from the mine, there are water mass sources and water mass outlets resulting from water vapor condensation. The condensation phenomenon occurring in the duct was described by the unit mass flows of water condensed in a given duct, occurring in the duct was described by the unit mass flows of water condensed in a given duct, referenced to the unit length of the condensation zone in the duct, is the unit mass flow rate of water and its physical quantity is \( \text{kg/(s m)} \). The knowledge of these values will allow to determine the aforementioned loss of pressure caused by the component of weight of the water droplets resulting from the vapor condensation, flowing with air at the same velocity. For this purpose, for each duct located within the discharge pathway, mathematical forms of the mass flows of water sources or outlets existing in the ducts must be provided as functions of the independent variable “s”, which is the distance.

The analyzed ducts include those, where:

- There is a local water mass source or local water mass outlet located at the inlet cross-section of the duct, as a result, e.g. of water condensation when wet air streams are mixing. Examples include the ducts: (5-10) with the local source (in winter) and (2-4) with the local outlet, located within the air discharge pathway analyzed in this paper (Ptaszyński et al., 2018).
b) there is a water mass source that is distributed throughout the length of the duct, resulting from the condensation of water vapor in the air flowing through this duct. Examples of such ducts located within the analyzed air discharge pathway include: (4-5), (10-12),

c) on a finite section of a given duct there is a distributed water mass source resulting from the condensation of water vapor in the air flowing through this duct. An example of such a duct located within the analyzed air discharge pathway is: (1-2), in which the condensation phenomenon occurs in the section (W-2), which is the above-mentioned distributed water mass source, whereas in the section (1-W) of this duct there is no water mass source.
d) the condensation of water vapor contained in the flowing air does not occur throughout the entire length of the duct, and therefore, there is no water mass source in it. An example of such a duct located within the analyzed air discharge pathway is: (12-K).

The mass flow rate, flowing through a given duct, depends on the presence of water sources from the condensation of water vapor contained in the flowing air, and from the position of a specific duct on the air discharge pathway, which determines which water mass flow flows into this duct from the preceding duct. Therefore, in the calculation of the water mass flow rates, it is important to assign them to the specific ducts located within the discharge pathway. Figure 2 illustrates the air discharge pathway from the mine, whereas the position of a given duct within this pathway is specified by the number of the initial point (cross-section) and the number of the terminal point of the duct, separated by a hyphen, provided in brackets. Such designation of the duct is present in the physical quantity subscripts that apply to these ducts.

In this research paper, the value of unit mass flow of condensed water \( m_{w(j)} \) in the local source/outlet is determined mathematically using the Dirac pseudo function \( \delta(s) \), hence it is mathematically written as

\[
m_{w(j)(a-b)}(s) = \pm c \cdot \delta(s - a), \quad \text{kg/(s·m)},
\]

where the subscript (a-b) of \( m_{w(j)}(s) \) means that the unit mass flow of the condensed water being the function of distance “s” refers to the duct with the nodes (a-b), of which the “a” node is the initial node, and the other one is the terminal node. The physical unit of the quantity \( m_{w(j)(a-b)}(s) \) is kg/(s·m). The value of “c” on the right of the formula (4) denotes the mass flow rate of this local source expressed in kg/s when preceded by the “+” sign, and when “c” is preceded by the “−” sign, it denotes the local outlet mass flow rate. It should be noted that if local sources or outlets are present on the analyzed discharge pathway, they are located in the initial node \( s = a \) of a specific duct (a-b). If, in addition to the already discussed local source (outlet) located in the “a” node, there is a continuous water source from the condensation of water vapor in the flowing and cooling air in the duct (a-b), then knowing the unit water mass flow “d” of this source distributed on the pathway, the constant value on the pathway in a given duct, expressed in kg/(s·m), the resultant unit water mass flow rate from the sources acting in the analyzed duct, can be determined in the form of the formula (5):

\[
m_{w(j)(a-b)}(s) = \pm c \cdot \delta(s - a) + d, \quad \text{kg/(s·m)},
\]

Formula (5) demonstrates that the value of the unit water mass flow rate in such a duct is the algebraic sum of the individual water mass flow rates of the sources and outlets acting in the duct.

For the ducts examined in (Ptaszyński et al., 2018), in general, the unit mass output of a water source in a duct can be described by the formula (5), but depending on the position of the duct on the discharge pathway and whether there are water mass sources or outlets in the duct, “c” and “d” may take different values. As an example, let us consider the duct (a-b), deprived of mass water sources, but located in the discharge pathway in such a way that there is a water stream in the flowing air in the preceding duct. This means that the duct (a-b) has only a transportation function for the wet air stream and for the water stream. This means that in Formula (5) describing the unit water mass source in this duct, the value of “d” = 0 (no sources in the duct), while the value of “c” is equal to the mass flow rate (kg/s) of the water flowing from the preceding duct to the duct (a-b).

Then, the mass flow rate of the water that is fed to the analyzed duct by the duct (or a sequence of connected ducts) preceding it, may be considered to be the mass flow rate of the local source located in the studied duct at its initial section (point) “a”. Knowing the value of the unit mass flow rate of water in the exemplary duct (a-b), i.e. the value of \( m_{w(j)(a-b)}(s) \), it is necessary to determine the water mass flow rate \( M_{w(a-b)}(s) \) in this duct as a function of the distance “s”. The mass flow rate of the water flowing in the exemplary duct (a-b), at the cross-section located in a position described by the coordinate “s”, is determined by the following Formula:

\[
M_{w(a-b)}(s) = \int_{a}^{s} m_{w(j)(a-b)}(s) \, ds, \quad \text{kg/s}
\]

where the variable “s” \( \in (a, b) \). The quantity \( M_{w(a-b)}(s) \) is a function which is continuous on an interval in definition interval, and its jump at the point of discontinuity is finite.

The above-mentioned quantity \( M_{w}(s = b) \) of the mass flow rate of water from condensation at the end of the duct <a, b> is determined using the relationship (6) as:
For the discussed example of air discharge from the mine, it can be written that:

1. Condensation of water vapor in the wet air stream, flowing in the vertical duct 1-2, occurs at the last 520 meters of its length. The unit flow rate of the distributed water mass source \( m_{W(1-2)}(s) \) is equal to 0.001240 kg/(s·m) throughout the entire length of 520 meters (Ptaszyński et al., 2018).

For such a value of the unit water flow rate, it is possible to determine a form of the water mass flow \( M_{W(1-2)}(s) \), flowing concurrently in the analyzed duct, using the Formulas (6) and (7):

\[
M_{W(1-2)}(s) = \int_{s_{1}}^{s} m_{W(1-2)}(s) \, ds = \int_{s_{1}}^{s} m_{W(1-2)}(s) \, ds + \int_{s_{0}}^{s} m_{W(1-2)}(s) \, ds = 0 + 0.001240 \, (s - s_{w}), \text{ kg/s (8)}
\]

\[
M_{W(1-2)}(s = s_{2}) = 0.001240 \, (s_{2} - s_{w}) = 0.001240 \cdot 520 = 0.6448 \text{ kg/s (9)}
\]

This quantity is the function of distance, but it is independent of the season since thermodynamic parameters of this wet air stream do not change throughout the year. At the initial cross-section 1, the mean air velocity is \( v_{1} = 4.33 \text{ m/s} \), in the cross-section W, where water vapor condensation begins, it is \( v_{w} = 4.38 \text{ m/s} \), and at the end point 2: \( v_{2} = 4.73 \text{ m/s} \). The wet air density at the cross-section 1 is equal to \( \rho_{1} = 1.2517 \text{ kg/m}^3 \), in the cross-section W: \( \rho_{w} = 1.2370 \text{ kg/m}^3 \), and in the cross-section 2: \( \rho_{2} = 1.1433 \text{ kg/m}^3 \). The mean values of wet air velocity and density at the section W-2 are, respectively:

\[
v_{ir(W-2)} = \frac{v_{w} + v_{2}}{2} = 4.56 \text{ m/s}, \rho_{ir(W-2)} = \frac{\rho_{w} + \rho_{2}}{2} = \frac{1.2370 + 1.1433}{2} = 1.1902 \text{ kg/m}^3
\]

2. In the 6-meter-long duct 2-4, there is a mixing of the two-phase stream of saturated wet air and water droplets (water mist) with a stream of wet air, resulting in a two-phase mixture. However, as demonstrated in the calculations in (Ptaszyński et al., 2018), at the initial cross-section of this short duct, there is a water mass outlet with a unit mass flow rate \( m_{W(2-4)}(s) = -0.099777 \delta(s - s_{2}) \), where \( \delta(s - s_{2}) \) is Dirac’s pseudo-function. In this duct, the mass flow rate \( M_{W(2-4)}(s) \) of the water flowing co-currently, with no slip, is expressed by the following Formulas:

\[
M_{W(2-4)}(s) = M_{W(1-2)}(s_{2}) + \int_{s_{2}}^{s} (-0.099777 \delta(s_{2})) \, ds = 0.6448 - 0.099777 \mathcal{H}(s - s_{2}), \text{ kg/s (10)}
\]

where \( \mathcal{H}(s - s_{2}) \) is a Heaviside’s function.

\[
M_{W(2-4)}(s = s_{4}) = 0.6448 - 0.099777\mathcal{H}(s_{4} - s_{2}) = 0.54503 \text{ kg/s (11)}
\]

The quantity \( M_{W(2-4)}(s) \) is constant on the pathway 2-4, because the outlet is located at the inlet cross-section of the duct, and its value does not depend on the season, since thermodynamic parameters of the wet air stream affecting it do not change during the year. In any cross-section of this section, the mean air velocity is \( v_{4} = 9.055 \text{ m/s} \), and the wet air density is \( \rho_{4} = 1.1394 \text{ kg/m}^3 \).

3. Condensation of water vapor in the wet air stream, flowing in the vertical duct 4-5, occurs at its entire length. The unit flow rate of the distributed water mass source \( m_{W(4-5)}(s) \) is constant throughout this 400-meter-long section, and it is equal to 0.00185 kg/(s·m) (Ptaszyński et al., 2018). The form of the water mass flow rate \( M_{W(4-5)}(s) \), flowing concurrently in the considered duct, is expressed by the following Formulas:

\[
M_{W(4-5)}(s) = \int_{s_{4}}^{s} m_{W(4-5)}(s) \, ds = \int_{s_{4}}^{s} [0.54503 \delta(s_{4}) + 0.001850] \, ds = 0.54503 \mathcal{H}(s - s_{4}) + 0.001850 \, (s - s_{4}) \text{ kg/s (12)}
\]

\[
M_{W(4-5)}(s = s_{5}) = 0.54503 + 0.001850 \, (s_{5} - s_{4}) = 1.28503 \text{ kg/s (13)}
\]

The quantity \( M_{W(4-5)}(s) \) is the function of distance but it is independent of the season, since thermodynamic parameters of this wet air stream do not change over the year. At the initial cross-section 4, the mean air velocity is \( v_{4} = 9.055 \text{ m/s} \), and at the cross-section 5, it is \( v_{5} = 9.420 \text{ m/s} \). The mean air velocity at this section of the shaft is equal to \( v_{ir(4-5)} = \frac{v_{4} + v_{5}}{2} = 9.238 \text{ m/s} \), and the mean wet air density at this section is:

\[
\rho_{ir(4-5)} = \frac{\rho_{4} + \rho_{5}}{2} = \frac{1.1394 + 1.0932}{2} = 1.1163 \text{ kg/m}^3
\]
At the further sections of air discharge pathway in the example provided in the research paper (Ptaszyński et al., 2018), the flow rate parameters of the medium depend on the time of the year. Therefore, in the paper (Ptaszyński et al., 2018) two periods were considered: summers and winters. In the summer season, further ducts were described by the numbers of the points representing the thermodynamic states of the discharged stream, denoted on the Mollier diagram as: 5-8, 8-11, 11-K. In the winter period, further ducts were denoted in the Mollier diagram with the numbers of the points representing the thermodynamic states of the discharged stream: 5-10, 10-12, 12-K. Point K denoted the thermodynamic state of the stream at the inlet of the main ventilation fan.

4. In summer, in the 6-meter-long duct 5-8, there is a mixing of the two-phase stream of wet air and water droplets (water mist) with the wet air stream drawn through the shaft outset seal. In the analyzed case, external losses accounted for 10% of the volumetric output of the air discharged from the mine, expressed as dry air stream. As a result, a two-phase mixture was produced, but as demonstrated in the calculations included in (Ptaszyński et al., 2018), at the initial cross-section of this short duct, there was a water mass outlet with a unit mass flow rate equal to \( \dot{m}_{w(5-9)}(s) = -0.04281 \delta(s - s_9) \). In this duct, the mass flow rate \( \dot{M}_{w(5-9)}(s) \) of the water flowing co-currently, without slip in the examined duct, is expressed by the following Formulas:

\[
\dot{M}_{w(5-9)}(s) = \dot{M}_{w(4-5)}(s_5) + \int_{s_5}^{s_9} [-0.04281 \delta(s)] ds = 1.28503 - 0.04281 H(s - s_5), \text{ kg/s} \tag{14}
\]

\[
\dot{M}_{w(5-9)}(s = s_9) = \dot{M}_{w(4-5)}(s_9) + \int_{s_9}^{s_9} [-0.04281 \delta(s)] ds = 1.28503 - 0.04281 H(s_9 - s_9) = 1.24222 \text{ kg/s} \tag{15}
\]

The quantity \( \dot{M}_{w(5-9)}(s) \) is constant on the pathway 5-8, because the outlet is located at the inlet cross-section of the duct, and the parameters of the outside air were assumed to remain constant during the summer season. At any cross-section of this section, the mean air velocity is \( v_9 = 10.173 \text{ m/s} \), and the wet air density is \( \rho_w = 1.0936 \text{ kg/m}^3 \).

In winter, in the 6-meter-long duct 5-10, there is a mixing of the two-phase stream of wet air and water droplets (water mist) with the wet air stream drawn through the shaft outset seal. As a result, a two-phase mixture is produced, but as demonstrated in the calculations included in (Ptaszyński et al., 2018), at the initial cross-section of this short duct, there is a local water mass source with a unit mass flow rate \( \dot{m}_{w(5-10)}(s) = 0.29968 \delta(s - s_9) \). In this duct, the mass flow rate \( \dot{M}_{w(5-10)}(s) \) of the water flowing co-currently in the examined duct, is expressed by the following Formulas:

\[
\dot{M}_{w(5-10)}(s) = \dot{M}_{w(4-5)}(s_5) + \int_{s_5}^{s_9} [0.29968 \delta(s)] ds = 1.28503 + 0.29968 H(s - s_5), \text{ kg/s} \tag{16}
\]

\[
\dot{M}_{w(5-10)}(s = s_9) = \dot{M}_{w(4-5)}(s_9) + \int_{s_9}^{s_9} [0.29968 \delta(s)] ds = 1.28503 + 0.29968 H(s_9 - s_9) = 1.58471 \text{ kg/s} \tag{17}
\]

The quantity \( \dot{M}_{w(5-10)}(s) \) is constant on the pathway 5-10, because the local water mass source is located at the inlet cross-section of the duct, and the parameters of the outside air were assumed to remain constant during the winter season. At any cross-section of this section, the mean air velocity is \( v_9 = 10.447 \text{ m/s} \), and the wet air density is \( \rho_w = 1.0805 \text{ kg/m}^3 \).

5. In the ventilation pipe, the flowing two-phase stream gets cooled in summer. Therefore, on the pathway 8-11, there is a fixed distributed unit water mass source equal to \( \dot{m}_{w(8-11)}(s) = 0.0306 \text{ kg/(s \cdot m)} \) (Ptaszyński et al., 2018). The volume of the water mass flow rate \( \dot{M}_{w(8-11)}(s) \), flowing out of this duct, co-currently with the air, can be calculated from the Formula:

\[
\dot{M}_{w(8-11)}(s) = \int_{s_9}^{s} [\dot{M}_{w(5-9)}(s = s_9) + \dot{m}_{w(8-11)}(s)] ds = \int_{s_9}^{s} [1.24222 \delta(s - s_9) + 0.0306] ds = 1.24222 H(s - s_9) + 0.0306 (s - s_9), \text{ kg/s} \tag{18}
\]

\[
\dot{M}_{w(8-11)}(s = s_{11}) = \int_{s_9}^{s_{11}} [\dot{M}_{w(5-9)}(s = s_9) + \dot{m}_{w(8-11)}(s)] ds = \int_{s_9}^{s_{11}} [1.24222 \delta(s - s_9) + 0.0306] ds = 1.24222 H(s_{11} - s_9) + 0.0306 (s_{11} - s_9) = 2.0990 \text{ kg/s} \tag{19}
\]

The quantity \( \dot{M}_{w(8-11)}(s) \) is a function of distance and relates to the summer period. In the initial cross-section 8 of the ventilation pipe, the mean air velocity is \( v_9 = 20.293 \text{ m/s} \), and in the cross-section 11: \( v_{11} = 20.539 \text{ m/s} \).
The mean air velocity at this section of the shaft is equal to \( v_{10(\theta-11)} = \frac{v_{10} + v_{11}}{2} = 20.416 \text{ m/s} \), and the mean wet air density at this section is \( \rho_{10(\theta-11)} = \frac{\rho_{10} + \rho_{11}}{2} = \frac{1.0936 + 1.0805}{2} = 1.0871 \text{ kg/m}^3 \).

In winter, in the ventilation pipe, the flowing two-phase stream also gets cooled. Therefore, on the pathway 10-12, there is a fixed distributed unit water mass source equal to \( m_{w,j(10-12)}(s) = 0.0275 \text{ kg/(s·m)} \) (Ptaszyński et al., 2018). The volume of the water mass flow rate \( M_{w(10-12)}(s) \), flowing with no slip, co-currently with the air, can be calculated from the formula:

\[
M_{w(10-12)}(s) = \int_{s_{10}}^{s} \left[ M_{w(5-10)}(s = s_{10}) \delta(s - s_{10}) + \dot{m}_{w,j(10-12)}(s) \right] ds = \int_{s_{10}}^{s} [1.58471 \delta(s - s_{10}) + 0.0275] ds = 1.58471 \mathcal{H}(s - s_{10}) + 0.0275 (s - s_{10}), \text{ kg/s} \tag{20}
\]

\[
\dot{M}_{w(10-12)}(s = s_{12}) = \int_{s_{10}}^{s_{12}} [M_{w(5-10)}(s = s_{10}) \delta(s - s_{10}) + \dot{m}_{w,j(10-12)}(s) ] ds = \int_{s_{10}}^{s_{12}} [1.58471 \delta(s - s_{10}) + 0.0275] ds = 1.58471 \mathcal{H}(s_{12} - s_{10}) + 0.0275 (s_{12} - s_{10}) = 2.35471 \text{ kg/s} \tag{21}
\]

The quantity \( M_{w(10-12)}(s) \) is a function of distance and relates to the winter period. In the initial cross-section 10 of the ventilation pipe, the mean air velocity is \( v_{10} = 20.474 \text{ m/s} \), and in the cross-section 12: \( v_{12} = 19.994 \text{ m/s} \). The mean air velocity at this section of the shaft is equal to \( v_{10(\theta-11)} = \frac{v_{10} + v_{11}}{2} = 20.234 \text{ m/s} \), and the mean wet air density at this section is \( \rho_{10(\theta-11)} = \frac{\rho_{10} + \rho_{11}}{2} = \frac{1.0905 + 1.1065}{2} = 1.0935 \text{ kg/m}^3 \).

6. In summer, the two-phase stream 11-K in the ventilation duct is not cooled. Therefore, the unit distributed water mass source from the condensation is \( m_{w,j(11-K)}(s) = 0 \). In winter, the situation is the same: \( m_{w,j(12-K)}(s) = 0 \). There is no internal water mass source in the duct, but the water from the condensation occurring in the preceding ducts is discharged through it. The water mass flow rate \( M_{w(11-K)}(s), M_{w(12-K)}(s) \), flowing in the air stream in the duct is constant over its entire length and the value depends on the period of the year. For the summer:

\[
M_{w(11-K)}(s) = \int_{s_{11}}^{s} \left[ M_{w(8-11)}(s = s_{11}) \delta(s - s_{11}) + \dot{m}_{w,j(11-K)}(s) \right] ds = \int_{s_{11}}^{s} 2.0990 \delta(s - s_{11}) ds = 2.0990 \mathcal{H}(s - s_{11}), \text{ kg/s} \tag{22}
\]

\[
\dot{M}_{w(11-K)}(s = s_{K}) = \int_{s_{11}}^{s_{K}} \left[ M_{w(8-11)}(s = s_{11}) \delta(s - s_{11}) + \dot{m}_{w,j(11-K)}(s) \right] ds = 2.0990 \mathcal{H}(s_{K} - s_{11}) = 2.0990 \text{ kg/s} \tag{23}
\]

The air velocity in the shaft duct in summer is equal to \( v_{3r(11-K)} = 20.165 \text{ m/s} \), and the wet air density at this section is \( \rho_{3r(11-K)} = 1.0805 \text{ kg/m}^3 \).

For winter:

\[
M_{w(12-K)}(s) = \int_{s_{12}}^{s} \left[ M_{w(10-12)}(s = s_{12}) \delta(s - s_{12}) + \dot{m}_{w,j(12-K)}(s) \right] ds = \int_{s_{12}}^{s} [2.35471 \delta(s - s_{12}) + 0] ds = 2.35471 \mathcal{H}(s - s_{12}), \text{ kg/s} \tag{24}
\]

\[
\dot{M}_{w(12-K)}(s = s_{K}) = \int_{s_{12}}^{s_{K}} \left[ M_{w(10-12)}(s = s_{12}) \delta(s - s_{12}) + \dot{m}_{w,j(12-K)}(s) \right] ds = \int_{s_{12}}^{s_{K}} [2.35471 \delta(s - s_{2}) + 0] ds = 2.35471 \mathcal{H}(s_{K} - s_{12}) = 2.35471 \text{ kg/s} \tag{25}
\]

The air velocity in the shaft duct in winter is equal to \( v_{3r(12-K)} = 19.629 \text{ m/s} \), and the wet air density at this section is \( \rho_{3r(12-K)} = 1.1065 \text{ kg/m}^3 \).

Figs. 2a and 2b illustrate how the mean wet air velocity changes in the cross-section of the pathway of the air discharged in summer and winter, respectively. Figs. 3a and 3b illustrate the change in the water mass flow rate from the condensation of water vapor carried away to the surface along the air discharge pathway in summer and in winter.
In a co-current flow upward, in which the velocity of water droplets and the velocity of the air stream carrying them away are the same (no-slip flow), the water mass $m_w$ in the vertical duct (W-2) is defined by the following formula:

$$m_{w(1-2)} = \int_{s_1}^{s_2} \frac{\dot{m}_{w(1-2)}(s)}{v_{ir(1-2)}} ds = \int_{s_1}^{s_2} \frac{0.001240(s-s_w)}{v_{ir(w-2)}} ds = \frac{0.001240}{2v_{ir(w-2)}} (s_2 - s_w)^2, \text{ kg}$$

Having substituted appropriate data, it was obtained as follows: $m_{w(1-2)} = 36.81 \text{ kg}$

In the next duct (2-4) located on the discharge pathway, the mass of water droplets is determined by the relationship:

$$m_{w(2-4)} = \int_{s_2}^{s_4} \frac{\dot{m}_{w(2-4)}(s)}{v_{ir(2-4)}} ds = \int_{s_2}^{s_4} \frac{0.54503 \mathcal{H}(s-s_4)}{v_{ir(2-4)}} ds = \frac{0.54503 (s_4-s_2)}{v_{ir(2-4)}}, \text{ kg}$$

Having substituted appropriate data, it was obtained as follows: $m_{w(2-4)} = 0.36 \text{ kg}$

In the vertical duct (4-5), the mass of water droplets is determined by the relationship:

$$m_{w(4-5)} = \int_{s_4}^{s_5} \frac{\dot{m}_{w(4-5)}(s)}{v_{ir(4-5)}} ds = \int_{s_4}^{s_5} \frac{0.54503 \mathcal{H}(s-s_4)+0.00185(s-s_4)}{v_{ir(4-5)}} ds = \frac{0.54503 (s_5-s_4)}{v_{ir(4-5)}} + \frac{0.00185}{2v_{ir(4-5)}} (s_5 - s_4)^2, \text{ kg}$$

Having substituted appropriate data, it was obtained: $m_{w(4-5)} = 39.62 \text{ kg}$

In the vertical duct (5-8), the mass of water droplets can be calculated from the formula:

$$m_{w(5-8)} = \int_{s_5}^{s_8} \frac{\dot{m}_{w(5-8)}(s)}{v_{ir(5-8)}} ds = \int_{s_5}^{s_8} \frac{1.28503-0.0481 \mathcal{H}(s-s_5)}{v_{ir(5-8)}} ds = \frac{1.24222 (s_8-s_5)}{18.173}, \text{ kg}$$
Having substituted appropriate data, it was obtained: $m_{w(8-11)} = 0.74$ kg.

In the duct (ventilation pipe) (8-11) inclined by the angle $\alpha$ to the horizontal plane, the mass of water droplets in summer can be calculated from the Formula:

$$m_{w(8-11)} = \int_{s_{g}}^{s_{11}} \frac{m_{w(8-11)}(s)}{v_{pr(8-11)}} \, ds = \int_{s_{g}}^{s_{11}} \frac{1.24222 \cdot 9.81(s-s_{g}) + 0.0306 (s-s_{g})}{20.416} \, ds = \frac{1.24222 (s_{11}-s_{g})}{20.416} + \frac{0.0306 (s_{11}-s_{g})^{2}}{2 \cdot 20.416}, \text{ kg} \quad (30)$$

Having substituted appropriate data, it was obtained as follows: $m_{w(8-11)} = 2.29$ kg.

In the duct (ventilation pipe) (11-K) inclined by the angle $\beta$ to the horizontal plane, the mass of water droplets in summer can be calculated from the Formula:

$$m_{w(11-K)} = \int_{s_{11}}^{s_{K}} \frac{m_{w(11-K)}(s)}{v_{pr(11-K)}} \, ds = \int_{s_{11}}^{s_{K}} \frac{2.0990 \cdot 9.81 (s-s_{11})}{20.165} \, ds = \frac{2.0990 (s_{K}-s_{11})}{20.165}, \text{ kg} \quad (31)$$

Having substituted appropriate data, it was obtained: $m_{w(11-K)} = 4.68$ kg.

Fig. 2b and Fig. 3b demonstrate that during the winter season, in the three final ducts of the discharge pathway, the water mass flow rate from the water vapor condensation on the air discharge pathway is greater than that for the summer period. Therefore, the water mass present in these ducts in winter will be different than previously calculated. The water mass in these ducts is determined as follows:

In the vertical duct (5-10), the mass of water droplets can be calculated from the Formula:

$$m_{w(5-10)} = \int_{s_{5}}^{s_{10}} \frac{m_{w(5-10)}(s)}{v_{pr(5-10)}} \, ds = \int_{s_{5}}^{s_{10}} \frac{1.28503 + 0.29968 \cdot 9.81(s-s_{5})}{10.447} \, ds = \frac{1.58471 (s_{10}-s_{5})}{10.447}, \text{ kg} \quad (32)$$

Having substituted appropriate data, it was obtained: $m_{w(5-10)} = 0.91$ kg.

In the duct (ventilation pipe) (10-12) inclined by the angle $\alpha$ to the horizontal plane, the mass of water droplets in winter can be calculated from the Formula:

$$m_{w(10-12)} = \int_{s_{10}}^{s_{12}} \frac{m_{w(10-12)}(s)}{v_{pr(10-12)}} \, ds = \int_{s_{10}}^{s_{12}} \frac{1.58471 (s-s_{10}) + 0.0275 (s-s_{10})}{20.234} \, ds = \frac{1.58471 (s_{12}-s_{10})}{20.234} + \frac{0.0275 (s_{12}-s_{10})^{2}}{2 \cdot 20.234}, \text{ kg} \quad (33)$$

Having substituted appropriate data, it was obtained: $m_{w(10-12)} = 2.73$ kg.

In the duct (ventilation pipe) (12-K) inclined by the angle $\beta$ to the horizontal plane, the mass of water droplets in summer can be calculated from the Formula:

$$m_{w(12-K)} = \int_{s_{12}}^{s_{K}} \frac{m_{w(12-K)}(s)}{v_{pr(12-K)}} \, ds = \int_{s_{12}}^{s_{K}} \frac{2.35471 \cdot 9.81 (s-s_{12})}{19.629} \, ds = \frac{2.35471 (s_{K}-s_{12})}{19.629}, \text{ kg} \quad (34)$$

Having substituted appropriate data, it was obtained as follows: $m_{w(12-K)} = 5.40$ kg.

The consequence of the occurrence of the water mass in the discussed vertical and variously inclined ducts is the loss of static pressure of the air caused by the weight of the calculated water mass. These pressure losses, determined for the individual ducts, constitute the quantities $A_{m}$, sought in this research paper, describing the flow characteristics of each of the analyzed ducts. They can be determined from the following relationships:

$$A_{m(1-2)} = \frac{m_{w(1-2)} \cdot 9.81}{F(1-2)} = \frac{36.81 \cdot 9.81}{34.84} = 10.36 \text{ Pa} \quad (35)$$

$$A_{m(2-4)} = \frac{m_{w(2-4)} \cdot 9.81}{F(2-4)} = \frac{0.36 \cdot 9.81}{34.84} = 0.10 \text{ Pa} \quad (36)$$

$$A_{m(4-5)} = \frac{m_{w(4-5)} \cdot 9.81}{F(4-5)} = \frac{39.62 \cdot 9.81}{34.84} = 11.16 \text{ Pa} \quad (37)$$

$$A_{m(5-8)} = \frac{m_{w(5-8)} \cdot 9.81}{F(5-8)} = \frac{0.74 \cdot 9.81}{34.84} = 0.21 \text{ Pa} \quad \text{(in summer)} \quad (38)$$

$$A_{m(5-10)} = \frac{m_{w(5-10)} \cdot 9.81}{F(5-10)} = \frac{0.91 \cdot 9.81}{34.84} = 0.26 \text{ Pa} \quad \text{(in winter)} \quad (39)$$

$$A_{m(8-11)} = \frac{m_{w(8-11)} \cdot 9.81 \cdot \sin \alpha}{F(8-11)} = \frac{2.29 \cdot 9.81 \cdot 0.7071}{19.63} = 0.81 \text{ Pa} \quad \text{(in summer)} \quad (40)$$

$$A_{m(10-12)} = \frac{m_{w(10-12)} \cdot 9.81 \cdot \sin \beta}{F(10-12)} = \frac{2.73 \cdot 9.81 \cdot 0.7071}{19.63} = 0.96 \text{ Pa} \quad \text{(in winter)} \quad (41)$$
\begin{align*}
A_{m(11-K)} &= \frac{m_{w(11-K)} \beta \sin \beta}{F_{(11-K)}} = \frac{4.68 \cdot 9.81 \cdot 0.0872}{20} = 0.20 \text{ Pa (in summer)} \quad (42) \\
A_{m(12-K)} &= \frac{m_{w(12-K)} \beta \sin \beta}{F_{(12-K)}} = \frac{5.40 \cdot 9.81 \cdot 0.0872}{20} = 0.23 \text{ Pa (in winter)} \quad (43)
\end{align*}

The values of $A_{m(a-b)}$ calculated above, in the exemplary ducts (a-b), which represent the air discharge pathway, refer to a no-slip flow in which all the condensed water is brought to the surface. These are the theoretically smallest values possible.

**CONCLUSIONS**

1. The procedure and calculation Formulas presented in this research paper allow to determine the value of $A_{m}$ for the analyzed ducts. This quantity is essential for determining the flow characteristics of the duct in which a fixed no-slip co-current two-phase flow occurs.
2. The calculations were carried out in the following ducts:
   a) vertical, in which the water mass flow rate was constant over their entire lengths (transporting ducts),
   b) vertical, in which the mass sources of the liquid phase were distributed over the length of the flow pathway,
   c) in which the mass sources resulted from the condensation of water vapor contained in the air. As a consequence, the mass flow of the liquid phase changed along the flow pathway,
   d) inclined at a certain angle, in which the water mass flow rate remained constant throughout its entire length,
   e) inclined at a certain angle, in which the water mass flow rate was changing along the flow pathway in the duct.

In each of the above-mentioned cases, the calculation Formulas for the water mass contained in the air in the analyzed duct and for the quantity $A_{m}$ were different, and were provided in the paper.
3. In all types of the studied ducts, the values of the water mass and pressure losses $A_{m}$, calculated for fixed two-phase no-slip flows are theoretically the smallest.
4. Theoretically, the smallest pressure loss $A_{m}$ on the air discharge pathway, expressed in Pa, is the sum of the calculated pressure losses $A_{m}$ for the individual ducts of this pathway, and its value (as indicated by the performed calculations for the presence of the liquid phase in the discharge pathway only due to the phenomenon of water vapor condensation) does not exceed several dozen Pa. However, one should be aware that, in fact, these values can be much larger.
5. The largest contribution to the total pressure losses $A_{m}$ occurring on the discharge pathway, have those long sections of vertical ducts in which the water mass in the air is the greatest.

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**NOMENCLATURE**

- $A_{m}$ - pressure exerted by the weight of water droplets in an inclined duct being a pathway for the discharge of air from the mine to the surface, N/m²,
- $c$ - volumetric concentration (volume) of water in air (constant value throughout the entire length of the duct), m³/m³,
- $m$ - mass flow rate of air and water in a duct, kg/s,
- $m_{w}(s)$ - the unit mass flow of the condensed water being the function of distance “s”, kg/(s m),
- $M_{W}(s)$ - the mass flow rate of the water flowing in the exemplary duct being the function “s”, kg/s,
- $H(s - s_{2})$ — the Heaveside’a function,
- $R_{m}$ - duct resistance for a two-phase mass flow rate, with constant mass concentration of water at the length of the duct, (m kg)⁻¹,
- $w^{*}$ - mechanical energy loss, N/m²,
- $(z_{1}-z_{2})$ - difference between height of inlet and outlet of the duct, m,

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GEOMECHANICAL ZONING WITHIN DIGITAL ECONOMY FOR THE PURPOSE OF OPTIMIZATION OF MINING CONSTRUCTION PARAMETERS

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ABSTRACT

Application of modern geotechnologies for the purpose of geomechanical processes in the rock mass and its influence on the production process are considered in the article. The role of digital technologies in the problem solution of management of mining pressure and predictions of its manifestations are estimated. Location of geomechanics is determined within «Digital Pit» and «Digital Mine» programs.

KEYWORDS

Geomonitoring, Geomechanical Model, Physical-Mechanical Properties, Stress-Strain State

INTRODUCTION

The key trend of the mining industry of Kazakhstan is currently a transition towards new technological levels according to the concept of Industry 4.0. Despite the fact that a portion of the fields in the country are equipped with modern equipment and with developed data transmission networks (21%), an essential portion of fields are in need of modernization (Ministry of information and communications of Republic of Kazakhstan).

Advent of the automated process control system (APCS) within digitalization of the field leads to: essential reduction of human presence at working spaces (Hormazabal, 2009), to a remote control system of the equipment (Pareira, 2011) (Urazbekov, 2004) (Urazbekov, Complex system of diagnostics of technical condition of railway track elements, 2005), to safety improving and efficiency of development, etc. An integral part of the automated process control system creation is development of information and navigation provisions as the base for formation of the automated systems for integrated mining control.

Development of geoinformational technologies at the present stage solves a problem of management of rock pressure and prediction of its manifestations more precisely and more fully with the use of digital technologies. During this work, estimation is performed of the role of digital technologies in a problem solution of management of rock pressure and predictions of its manifestations. Also, location of geomechanics is determined within the "Digital Pit" and "Digital Mine" programs. (Figure 1)

Figure 1 - Structural model of “Digital Mine”

During development of mine, technology of mining illustrates close connection with geomechanical processes dynamics. The recent period of mineral deposits working offs is characterized by increase in development depth and complication of bedding conditions, involvement in working off of temporarily inactive stocks. The possibility of working off of these stocks demands the analysis of geological conditions of the field and forecast of changes of geomechanical conditions during mining operations. Mining designs can have significant differences in geology, geometry, development duration and geotechnical tasks. Therefore, it is rather difficult to determine the level of monitoring accuracy without studying of mechanics of deformations. To achieve that, geomechanical zoning of the field is carried out and the system of uniform geomonitoring is defined.
Geomonitoring is understood as the system of the regular observations, collection, accumulation, processing and the analysis of information, assessment of the geological environment condition and the forecast of its changes under the influence of natural factors and technogenic loading (Hormazabal, 2009).

Geomonitoring and prediction of geomechanical processes include researches of displacement process of the land surface; studying of geological features, hydrogeological, physical and mechanical rock properties, a tentative stress-strained state of the rock mass and its change in the course of conducting mining operations; creation of 3D geomechanical model with the stability ratings; model operation of geomechanical processes, etc.

**Development of field geomechanical models.** While developing geomechanical models within uniform geomonitoring, the main objective is obtaining the characteristics reflecting a qualitative condition of the massif for identification of potentially dangerous sites and division into districts of the field (zoning) according to a stability criterion for safety of conducting mining operations. Geomechanical models have to consider an actual stress-strained state, a block structure and physical-mechanical properties of rocks, qualitative characteristics of the massif, value of a stress-strained state, geometrical parameters of ledges etc. is possible on the basis of such models which were created oriented on the actual information of the massif.

The 3D block geomechanical model of the field is created in terms of a geological model including the lithologic differences of rocks, tectonic violations, the main systems of cracks (see Figure 2). The geomechanical model includes geo specifications of fields: structure of the massif, orientation of cracks, their parameters, cracks surface pattern, physical and mechanical properties of ore and the containing breeds, hydrogeological parameters, a tentative stress-strained state of the massif and its change in process of conducting mining operations, and classification of quality of the massif etc. (Sjedina, 2017) (see Figure 3).

![Figure 2 - 3D geological model of the field](image)

The received model allows model operation of any changes of the massif stress-strained state and forecast of various geomechanical situations at field working off.

The whole massif is zoned on categories of stability by results of the analysis and a generalization of all criteria brought into the geomechanical model.

Thus, geomechanical zoning on several categories depending on physical and mechanical properties, hydrogeology, durability of the massif, its joining, coverage of studies etc. makes possible to generally assess situation on a stable state of the massif fulfilled and planned to working off. Also, it allows taking in advance measures for ensuring stability of mining designs and conducting safe mining operations, optimization of development system parameters and volume of mining operations.

![Figure 3 - Stages of geomechanical models development](image)
Development of the system of high-precision satellite positioning. Creation of a system of high-precision positioning of the technical means which are logging in a uniform geomonitoring is necessary for high-precision determination of geodetic coordinates and heights by means of the modern satellite navigation technologies in the modes of real time and post-processing. Besides, this system will allow carrying out: the current surveying works on the field; monitoring of mining-transport inventory and personnel; geodetic works for the production, administration, and road construction on the territory of mining allotment of fields.

Realization of a system of high-precision positioning allows the industry to essentially change the current monitoring system of deformations. Moreover, it allows the supervision on the fields using the unified approach to the problem solution of high-precision positioning of the used technical means and data exchange between these sensors and the situational center.

The technology which allows solving problems of high-precision satellite positioning is the technology of differential correction of GNSS signals in the form of base stations of differential correction (Genike, 2004).

The mobile and stationary terminals integrated into an inventory design (on monitoring of deformations) have to be developed at the tentative stage. They will provide full process automation of high-precision positioning and data transmission in the situational or dispatching center. At the same time, development of technical documentation and geodetic points connection of basic geodetic network on the area is necessary. It will allow to differentiate the causes of shift of the phase centers. Determination of normal heights has to be automatic on the basis of use of model of quasigeoid heights and satellite navigation signals.

With a reference to the received basic geodetic network, the digital model of a field surface is under construction. It reflects the measurement results of deformations and displacement of the land surface received in various ways and by the use of various methods. The digital model of a surface (see Figure 3) makes possible the division of the field surface into “stable” and “inclined to deformations” sites, which analysis will then allow to predict a condition of the land surface of the field (Baltiyeva, 2017).

![Digital model of the pit surface](image)

**CONCLUSIONS**

Main objective of the development of uniform geomonitoring system and geomechanical zoning – safety improvement while conducting mining operations, prevention of accidents bounded to geomechanical processes, optimization of parameters of development systems and mining operations volume.

The algorithm of research given in this work is directed to implementation of the state program "Digital Kazakhstan" for the period 2018-2022. In particular, development of a software and hardware complex of system of high-precision satellite positioning and the unified system of collection and data transmission will be coordinated with a state program task of industry digitalization.

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PROSPECTS FOR DEVELOPMENT OF WORLD COAL MINING AND THERMAL POWER GENERATION

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ABSTRACT

The article deals with the problem of choosing a strategy for managing the coal industry and coal enterprises in the context of various economies of the world. The factors of the economic and ecological justification for either extension or termination of the coal enterprises’ functioning are analyzed. A comparison of key indicators for energy generation based on coal combustion and the use of alternative energy sources is given. The concept of synchro-mining for extending the life cycle of a coal mine in a market economy is proposed.

KEYWORDS

Synchro-mining, industrial park, coal mining, thermal power generation.

INTRODUCTION

In recent decades, the world has been concerned with the problem of environmentally responsible management, for the solution of which one of the fundamental concepts of modernity has been developed – the paradigm of sustainable development, embodied mainly in “green technologies” for producing energy from alternative sources [1].

Actually, the alternative to energy production opportunities led to a serious global discussion about the need for coal as a resource, the feasibility of closing coal mines and abandoning the coal industry as a whole. Experts’ views, however, vary from the most radical to the most loyal, which ultimately determines the type of coal strategies that are being implemented in different countries around the world. So, if in the 20th century the problem of the coal industry restructuring arose only in the UK, then in the 21st century the dilemma of "whether to use coal or not" as well as "whether to close mines or not close" is faced by all countries of the world, including those that have never had coal industry in its economic complex.

The English Mining Revolution had a clearly expressed economic justification for the closure of mines, which consisted in a shortage of budgetary funds for their maintenance. At the same time, the plan to abandon the coal industry was first promulgated in 1984. Nevertheless, the last UK mine was closed only 31 years later in winter 2015. Even with the strong political Margaret Thatcher’s will, the closure of the coal mines took a long time to implement the plan [2]. This shows that the coal industry is not so easy to remove from the structure of the economy.

Today, the problem of coal mines closing has become ecologically and economically oriented. In the economic context, it is largely about the price loss of coal to alternative sources of energy. In the environmental - "cleaner" alternative technologies in the energy sector, many technical problems of which have been still far from being solved. Obviously, the two above marked directions indicate the presence of a certain competition of different economic interests. They also give the coal topic some political component, which in the end also affects the attitude towards coal in different states.

Briefly review the state of affairs in the coal industries against the backdrop of trends in the energy market in some countries of the world.

RESULTS

The United States of America Perspective

The period of Barack Obama’s presidency is characterized by the bankruptcy of 50 coal companies and the drop in coal production in the US by 40% from 2008 to 2016 [3]. This was a goal-oriented and intentional policy in the trend of an environmentally friendly economy, which was supported by the relevant regulatory enactments. Among these acts were a moratorium on the new federal mines opening and a strict system for controlling emissions into the atmosphere. Obama’s opponents said that the president literally buried tens of billions of dollars in the land in the form of unextracted coal. As a result, if the resources mix providing electricity in the US in 2008 consisted of coal by 50%, then in 2016 - only by 30% [3].

Robert Murray - the head of the largest coal corporation in the US and the closest associate of the current president, Donald Trump - says that the best result for the coal industry of the States in the coming years will be to retain the above-mentioned 30% of the resource market [4]. In reality, it is predicted, albeit not a sharp, but a decline...
in demand for coal. And this despite the fact that the so-called recovery of the coal industry was one of the major points of Trump’s election program, because coal is mined in half of the US states. However, as Mr. Murray argues, Trump will not be supported by the Senate in repealing the above-mentioned regulations severely restricting the coal industry, and investors will not want to return to the resource portfolio with a coal dominant and prefer to balance and diversify it as much as possible [4].

The opinion expressed by Mr. Murray about the outflow of investors from the coal industry can hardly be extrapolated to the global level. According to the Bloomberg agency, in fact, in the projected investment structure in the energy sector until 2025, there is no capital investment in the US coal industry, just as they are absent in the proposed investment structure in Europe (Figure 1). But simultaneously, in the countries of the Asia-Pacific partnership (APAC), the Middle East region (MEA), and in other countries of the world (RoW), investments in coal of about 400 billion US dollars are planned. Only APAC partnership intends to invest almost 4 times more funds in energy than the US, Europe and other groups of countries combined [5].

Figure 1 – Structure of World Investment in Energy Generation in 2016 – 2025, $b [5]

The experts of the Institute for Economics of Energy and Financial Analysis (IEEFA) also comment on the reduction of the coal industry in the USA arguing that the only competitive advantage of coal is the growing price of gas. In January 2017, the Institute predicted that by 2050 most of the coal-fired power plants would be closed due to the fact that the business will steadily withdraw capital from the industry [6]. However, if you look at the US stock exchange market, the rating agency Zaks in February 2017 placed the coal industry on the 65th place out of 258 in terms of shares and bonds, which indicates a good position of the coal companies in the financial market: their shares rose by 153% this year compared to the growth of 24% last year, and revenues for them increased by 7.5% compared to an increase of only 4% last year. But experts of the agency do not predict the growth of shares of coal companies in the long term [7].

In our opinion, it is interesting to consider the opinion of the Standrd & Poor's, which believes that coal companies will retain their market share in the future and can be even more effective, but only on the condition of severe cut of wages. From time to time, the US press already has a formula: “more natural gas and alternative energy = fewer jobs in the coal industry.” Meanwhile, the US Department of Energy in March 2017 announced that this year it has been planned to add 26 gigawatts of power, of which 9.5 - due to solar energy; 8 - natural gas; 6.8 - wind power [8].

**Poland Perspective**

According to the results obtained by the researchers of the Polish Academy of Sciences, in 2016 coal accounted for 89%, natural gas - 3%, wind - 6.6% in the power structure of Poland (Figure 2). Out of 40 gigawatts of power installed in 2014, 31 is provided due to coal; 0.9 - gas; 2.2 - hydro resources and 4.2 - alternative sources. In
2050, Poland is planning only to begin closing coal-fired power plants with an approximate total capacity of 18 gigawatts, and today it is consistently modernizing existing coal-fired power plants. For example, some of them have been already equipped with high-performance boilers and turbines with supercritical parameters - temperature at 620°C and pressure up to 30 MegaPascals [9].

Figure 2 – Structure of Energy Sources in Poland and USA in 2014, 2016, 2017 [9]

In the context of the prospects for the coal industry, the United States and Poland represent fundamentally different strategies: The United States declares a rejection of coal for the sake of a green economy. On the contrary, Poland puts energy security and independence above the goals of sustainable development. This is confirmed by the fact that the import of coal to Poland is only 25% of the required volume, in the EU 53% despite the fact that the cost of coal imported to Poland is lower (USD 50-52) than that of the extracted coal (USD 76). But the Polish government believes that it is better for the national economy to leave domestic, albeit subsidized, coal for supplying existing consumers than to expand the market for importers. In this case, the amount of subsidies from the budget will be less than the total cost of imported coal [9].

Nowadays, the priority areas in the development of coal enterprises in Poland are the development of coal gasification technology for highly efficient production of fuel and electricity; production of hydrogen-rich gas during chemical cyclic combustion of coal; gasification of coal with CO₂ absorption.

Europe Perspective

The role of coal in the process of the European transition to alternative energy sources is best identified in the report by the President of the European Association for Coal and Lignite “Euracool” Mr. Wolfgang Cieslik in 2016. From the statistical data given in the report, we can draw the following conclusions.

1. In 2015, there was no country in Europe that would not have used coal for its own energy supply in any way (Figure 3). Even Britain, having closed its last mine, continued to import coal in the amount of 25.5 million tons. A total of 28 EU member states produced about 500 million tons of coal in 2015 and imported about 100 million tons for their own needs [10].

2. Coal is imported by all EU countries. From this, we can conclude that those countries that do not have their own coal mining (France, Italy, Portugal, Finland, Ireland, Belgium, the Netherlands and others) can not abandon this resource even with sufficiently developed innovative technologies of alternative energy, and so countries that independently produce coal (Germany, Poland, Turkey, Bulgaria, Greece, Ukraine) are yet lacj of it. The largest importers of coal in the EU in 2014 were the United States and Colombia (42% overall), Russia (30%) and Africa (10%) [10].

The share of coal in the EU’s electric generation in 2014 was 26.4% on a par with the atom (27.5%), water and gas (27%) (Figure 4). The share of alternative energy sources (wind, sun, tides, biofuel, waste) was slightly higher...
- 17%. We see the resource-rich diversified energy portfolio of the EU in the transition period with a stable dominant of traditional energy sources [10].

Figure 3 – Coal Production and Imports in Europe in 2015, million tones [10]
The important role of coal in the current energy balance Mr. Cieslik emphasizes in the example of Germany. In 2013, electric power was generated there in two types of power plants - fuel and renewable. Fuel power stations consistently provided about 90 GW of power (coal - 50, gas - 30 and atom - 10) (Figure 5). On the contrary, out of 71 GW of installed capacity of renewable wind-solar power stations, never more than 35 GW were provided, and in some nights the power of these power plants fell to 0.1 GW. "It's still impossible to maintain a light bulb burning on a calm night without traditional fuel-fired power plants," notes Mr. Cieslik [10].

Figure 4 – Share of Coal in Power Electricity Generation in EU in 2014, % [10]

Synchro-mining Concept

The foregoing confirms that the attitude to coal as a resource is different in the world. It is maintained by economic reasons for its high cost compared to alternative sources, other economic reasons for the large reserves of this resource that can not be conserved and environmental reasons against coal.

In our opinion, technological reasons are the most objective. Combustion of coal has been a constant and controlled process throughout decades, whereas the wind, water, and tides are not permanent and still poorly controlled natural phenomena, for which it is necessary to have impressive storage to smooth out the instability of operations based on their use. Therefore, decisions on coal should be weighed and objective. Talking about the closure of the industry is clearly premature, but thinking about how to use coal mines and the resources available due to them is a topical issue.
In this regard, the National Mining University offers its vision of the coal problem, which consists, firstly, in a differentiated approach to each individual mine, and secondly, in the application of three decisions - "Whether or not to close as well as give a second life" instead of only two ones - "whether to close or not".

What does the differentiated approach to each individual mine mean and what can its second life be related to?

A mine can be closed in two cases. The first case is characterized by an objective physical exhaustion of the mineral. The second case lies in the economic plane and is associated with the high cost of coal, its environmental insolvency, and, in fact, the reluctance to redistribute investments for the purpose of "greening" the mining process and choosing an easy path of importing coal from other interested countries. However, such mines can be given a second life through the introduction of innovative environmentally friendly technologies. Such examples of successful combination of science and production exist in Germany, Poland and the Netherlands.

One should not forget about those mines that can not be physically closed due to difficult hydro-geological conditions and the need for their constant operating in the mode of water pump to avoid flooding of adjacent territories. There are many examples of such mines, and not only could but must they be given a second life.

But for this it is necessary to develop a complex multi-industry project to prolong their life cycle.

Such a complex multi-industry project, adapted to the development objectives of mining enterprises and regions, should

- be implemented within the framework of a sufficiently powerful mechanism of public-private partnership widely employed in developed countries,

- be multi-industry and based on a platform of the latest technological solutions in different fields,
- have clear business orientations,
- contribute to the improvement of the investment climate of the region where a mining company operates,
- fit into the organizational models having already been established by law.

It is very important to understand that the problem of giving the second life to a mine through the implementation of complex multi-industry projects must be solved not at the moment of its closure, but in the period of stable operations.

Since 2010, the National Mining University has been conducting research to formulate a concept of Synchro-mining for the development of mining enterprises and regions, which has been brought to the level of an integrated multisectoral project. This project, with some refinements with representatives of governmental authorities and private companies, could contribute to solving the problems of mining enterprises and regions development on the principles of public-private partnership [11-15].

The Synchro - mining project is based on the idea that parallel to the main activity of a mine, which is mining, for example, coal mining, independent business projects can be implemented on the same open technology platform.

This technological platform represents a set of engineering innovative technologies adapted to the conditions of mining enterprises.

Taking into account the need for active resource and energy saving, all technologies of the Synchro-mining platform allow using the available natural resources of the mine, such as remains of undrawn coal, water, gas, solar radiation, wind, as well as mine assets, including underground and ground-based complexes and surroundings. As a result, it is possible to obtain additional strategic products in the form of synthesis gas, energy, clean water, agricultural products on a lucrative basis. Then the potential of the mine can be estimated not only from the perspective of available coal reserves, but also from the standpoint of other economic opportunities that this mine can provide [11-13].

In this case, with stopping extracting mineral resources, a mine’s economic attractiveness does not radically decrease, its life cycle does not end, and the regional economy does not experience socio-economic shocks.

In countries with numerous state mines, attracting private companies not only for coal mining, but also for managing and renting mine assets in the new context of the second life of the mine on the principles of public-private partnership can make an alternative to their privatization under normal conditions. For private companies, to provide a second life to the mine means to show their commitment to the ideas of corporate social responsibility. A successfully
functioning mine in a post-mining period is the best payment to future generations for the natural resources currently being used, which fully meets the principle of sustainable development of the territories.

The openness of the technological platform is manifested in the possibility of forming a diversified consortium of participants (developers, owners, customers, users of innovative technologies) at national and international levels [11-13].

Thus, the potential of a possible multi-industry project of public-private partnership "Synchro-mining" consists of the potentials of individual business projects that start from an open technology platform and are able to generate economic, social and environmental effects for the enterprise and the regional economy.

The tasks to be accomplished when implementing this project are as follows.

In the social field: creating new jobs and modernizing social infrastructure, reducing the level of migration, especially for young people, developing the business sector and, as a result, aligning the demographic situation in the region.

In the ecological direction: the restoration of the surface landscape damaged by long-time mining, the soil, water and air cleaning, the introduction of a people’s health monitoring system, the implementation of environmental projects related to a "wasteless management".

In the economic direction: activation of innovative processes based on the implementation of innovative business projects, the transfer of technology, the formation of enabling investment climate, the creation of scientific and production clusters.

Benefits for investors consist in prolongation of the period of profitable mine’s operations, diversification of the investment portfolio and distribution of risks between different projects, obtaining the effect of economy of scale due to the implementation of projects in the cluster system.

Benefits for the region are reduced to an extension of a city-forming enterprise’s life cycle and the avoidance of local socio-economic crises when it is closed, ensuring stable revenues to the budget, reducing unemployment, and creating an entrepreneurial environment.

CONCLUSION

At present, the following business projects can be included in the multi-sectoral project of the public-private partnership "Synchro-mining": complex for underground generation of synthesis gas, a complex for the generation and distribution of green energy and electricity, a water treatment complex, an agro-complex with greenhouse facilities, complex for the materials recycling and the synthesis of useful chemical elements, entertainment and service center for industrial tourism "Park-Museum" Technoland", logistics-warehouse complex in the cavities of underground workings and buildings on the surface, and other projects [14].

The organizational model of the project of public-private partnership "Synchro-mining" can integrate the organizational and legal mechanisms of the science and industrial parks.

The Science Park regulates the relations of science and business in matters of intellectual property while creating and transferring innovative technologies. The Industrial Park regulates the relations between developers of technologies, business and local authorities in matters of land allocation and granting privileges for doing business on its territory.

Such benefits provided by the Law of Ukraine "Industrial Parks" include: the possibility of obtaining state financial support for the arrangement of the park (in the form of targeted financing on a non-repayable basis), obtaining interest-free loans from the State budget, tax incentives, the right to rent land for an industrial park for at least 30 years and exemption from import duties on equipment, materials, etc. for the operation of the park.

In 2017 in Pavlograd, the 18th largest industrial park opened in Ukraine by the initiative of the Dnieper Regional State Administration. It occupies the largest area of 250 hectares in Ukraine and is now at the stage of collecting applications from potential investors. According to Alexander Kolomytsev, the head of the company managing the Pavlograd industrial park, the first investment project will be aimed at obtaining synthesis gas from the coal of the gas group [15].

Thus, industrial parks are rather prospective. At the same time, a mining company could also be simultaneously both a co-founder of the scientific park and the company managing the industrial park on its own or
communal land, monitoring the creation and transfer of new technologies, which only increases the attractiveness of public-private partnerships for it. In turn, the state thus transfers part of the risks associated with the formation of socio-economic depression in the territories of intensive mining to a private partner.

On balance, in the transition period from carbon to alternative energy sources, coal needs to be considered as a strategic resource along with active development of other types of renewable resources. The mines, which can still produce coal and those that can not be physically closed, must be given a second life. At present, authorities and private companies face the important task of developing a systematic and mutually beneficial project for the prolongation of the mine’s life cycle on the principles of public-private partnership.

The National Mining University sees the possibility of implementing a multi-industry project of public-private partnership "Synchro-mining" for the development of mining enterprises and regions. The existing conditions for its implementation can be described as unique, without exaggeration, because there are proven innovative technologies, legislative framework, socially responsible private companies ready for dialogue, commitment of government bodies, it is only necessary to inform each other about existing opportunities and initiative push to open negotiation processes between the participants.

REFERENCES


WEAR BEHAVIOR OF LINER OF LIGHTWEIGHT METALLIC SKIP

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Abstract:

The mineral deposits are exploited on deeper and deeper levels. In order to improve the efficiency of haulage hoist system for the super-deep shaft mine, it is necessary to have stricter requirement for the lightweight of large mining skip. As the major component of mining skip, the liners endure impacting force and abrasion during loading and unloading the iron ore bulk. The weight ratio of liner to the total mining skip is rather large for the thickness of liner plate. The abrasion-resistant material is chosen to minimize the thickness of the liner plate and extend its life service. Three kinds of lining material NM500, NM400 and 16Mn are selected to test their wear performance based on skip loading condition. The wear rate and wear coefficient of these materials are calculated according to the experiment results. The EDEM model of metallic skip is built to simulate the wear behavior of liner during the loading and unloading process. The wear distribution of liner using different lining material can be predicted, the reasonable liner thickness and their layout scheme are suggested for different region of liner.

Keyword: metallic skip, lightweight, liner, wear, DEM

1. Introduction

As mineral resources exploiting deeper and deeper, a large mining skip is needed to lift the more iron ore in order to improve the efficiency of lifting. Balanced-hoisting system at relatively shallow depth (< 1,600 m), relying on one skip to assist the other while still moving acceptable payload, is rather efficient. However, due to the fact that the weight of extended hoist rope accounts for a great proportion in the total hoisting system weight, this lifting efficiency drops away rather quickly as hoisting depths approach 2,000 to 2,500 meters (S. Gorzalczynski, 2015).

The increasing demand for iron ore is placing greater emphasis on the need for higher mining production capacity coupled with the objectives of more efficient, low cost operations. For the deep mining hoisting system, it is necessary to improve the mineral output to guarantee the suitable mining economic. Thus, it demands that skip should contain more mineral during a single lifting cycle.

Lightweight mining skip is the trend of the mining equipment. Reducing the weight of the mining skip is equal to increase the effective loading weight, thus increase the lift ability of the mining hoisting system. Moreover, a lightweight mining skip can supply more space to the weight of longer hoist steel rope. The mining lifting height can be longer under the same rope diameter. It is helpful to increase the mining depth further in the future.

For the bail-type bottom dump mining skip, the weight of wear liners is about 12% of the skip tare weight (S. Gorzalczynski, 2015). Lightening the liner is usually considered through using wear resistance steel and reducing liner thickness.

The skip liner is composed of many steel plates while their thicknesses and sizes may be different (Fig.1). The liner of the metallic skip is easy to wear during the skip loading and dumping the iron ore bulks. Considering the density and hardness of iron ore are much higher than that of coal, the process of loading and charging iron ore will cause the severe wear behavior on the skip container wall compared with the coal mining skip. The performance of mine hoisting skip will deteriorate due to continuous usage and harsh working condition. The poor of the wear resistance of skip container (shown in Fig.2) will aggravate the frequency of equipment downtime caused by the liner failure. It is necessary to extend the life usage of liner, and reduce the maintenance cost.
A lot of studies have been done to use the discrete element method to predict the worn profile shapes of liners and lifters in mills. The wear resistance of different lining materials in bins and chutes were tested by purposely designed experimental apparatus, and the service life of lining materials were discussed (Wei Chen, 2017). DEM model is used to predict the mill liner evolution and the operational changes over the full life cycle of the liner (Cleary and Owen, 2010). The liner wear evolution in ball mills is simulated, and the liner profile evolution with time based on the wear model is predicted (Powell, M.S., 2011). The sensitivity of wear prediction subjected to DEM parameters was analyzed at the single iron ore particle level (Chen G.M., 2017). Therefore, DEM model provides a powerful tool for simulating the liner wear of metallic skip and predicting the liner service life.

The purpose of this paper is to experimentally investigate the abrasive wear performance of three kinds of liner materials against iron ore. The DEM model of metallic skip is built to simulate the loading and dumping process of iron ore, from which the wear performance of liner using the different liner materials can be compared by their material properties. It is an efficient way to quantitatively determine the thickness of liner plate locating on different area of skip wall.

2. Wear resistance material used for metallic skip liner

The liner of metallic skip should possess higher surface hardness to resist the chipping and cutting of the iron ore. And it also needs the liner possess the strength and ductility performance to endure the impacting force. The lining material properties, including wear rate, wear coefficient and hardness, vary as the abrasive wear propagates into the material from frictional rubbing. It can be used to evaluate the abrasive wear resistance of lining material.

2.1 Abrasion wear test of lining material

Three kinds of lining materials (NM400, NM500, and 16Mn) used as internal wear liner are selected to test their wear resistance performance. The ball-on-disc wear test shown in Fig.3 is conducted based on the testing material and ball. The range of hardness of iron ore is 6~14. The testing wear ball with diameter Φ5mm is made of silicon carbide (hardness 6.5) and corundum. The testing liner material is processed as disc shape with dimension Φ35×10mm. This particle ball is positioned at a distance r=16 mm with regard to the central axis OO’.

Two factors (load and sliding velocity) which affecting the liner wear behavior are tested for calculating the wear properties. The angular velocity of the disc geometry is with respect to its central axis (OO’) and F_N shown in
Fig. 3 is the indentation force acting on the ball particle. This test is useful for analyzing a wear loss measurement.

The mass loss of lining material is measured and weigh every 20 minutes during the wear testing 60min. The value of mass loss is averaged with 5 weighing results.

The wear rate of lining material is used as the fundamental liner characteristics which representing the mass loss varying with the time. The wear process of friction pair considering in this study lasts from the running-in to quasi-steady state. The tendency of wear rate can be depicted as following equation (Kumar R, 2002):

$$\omega_m = (\omega_0 - \omega_s) e^{-bt} + \omega_s$$  \hspace{1cm} (1)

where, $\omega_m$ is wear rate of mass loss, g/min; $\omega_0$ is the initial wear rate, g/min; $\omega_s$ is the steady-state wear rate, g/min; b, negative correlation nonlinear coefficient during running in, min$^{-1}$.

![Graph showing wear rate of lining material with different conditions](image)

**Fig.4 Wear rate of lining material**

The experiment results of mass loss of lining material are plotted in the Fig.4 while the wear rate varying with time is fitted with the equation (1). As shown in Fig.4(a), the wear rate of NM400, 16Mn is much higher than that of NM500 under the condition of the same force and spinning velocity. The wear rate of NM400, 16Mn will decrease sharply within the 40min in which the wear process may be in the period of running in. The test results in Fig.4(b) indicates that the wear rate of NM500 will vary with the spinning speed and indentation force. The wear rate will become large with the increasing of spinning velocity.

### 2.2 Wear coefficient of lining material

Wear coefficient is dimensionless coefficient which reflecting the wear resistance of lining material. The larger the wear coefficient, the better the material wear resistance will be. The volume loss of material is proportional to the load, sliding velocity and wear time, and varies inversely with the material hardness of friction pair. The wear coefficient K is represented as following (Bartier O, 2010):

$$V = K \frac{FS}{H^*}$$  \hspace{1cm} (2)

$$H^* = \begin{cases} 
\left( \frac{1}{H_1} + \frac{2}{H_2} \right)^{-1}, & R_1 < R_2 \\
\left( \frac{2}{H_1} + \frac{2}{H_2} \right)^{-1}, & R_1 = R_2 
\end{cases}$$

where, V is the wear volume loss, m$^3$; S is the sliding distance of ball, mm; K is the dimensionless wear coefficient; F is testing load, N; $H^*$ is relative hardness of contact face, HB; $H_1$, $H_2$ is the material hardness of friction pair respectively; $R_1$, $R_2$ is the curvature radius of two contact face respectively, mm.
The wear coefficients of three tested lining materials are shown in Fig. 5. The wear coefficient of NM400 and 16Mn is closely to each other for the wear coefficient in equation (2) considering the hardness of contact face. The hardness of NM400 material used in this wear experiment is a little lower than the standard NM400 material. Comparing with the lining material of NM400 and 16Mn, experiment results show that the material NM500 displays the better wear resistance characteristics when it used as the skip liner.

3. Liner Wear Simulation during loading and dumping

3.1 EDEM model of mining skip

The discrete element method (DEM) simulations have been utilized to predict the abrasive wear caused by iron ore. Large-scale handling of iron ore bulk-solids causes a high amount of volume loss from the surfaces of metallic skip liner. Predicting the sliding wear depth of liner surfaces is beneficial for efficient maintenance of metallic skip.

EDEM, one of the famous DEM software, is used to simulating the wear behavior of skip liner. The Hertz-Mindlin with Archard Wear model in EDEM extends the standard H-M model to give an estimation of wear depth for geometry surfaces (EDEM, 2016). The model is based on Archard model and uses the idea that the amount of material removed from the surface will be proportional to the frictional work done by particles moving over the surface.

Based on Archard’s wear model (Archard, 1953), the wear volume loss $Q$ for the sliding wear model is estimated by:

$$Q = W F_n d_t$$  \hspace{1cm} (3)

where, $Q$ is the volume of material removed, $F_n$ is the indentation force, $d_t$ is the tangential distance moved and $W$ is a wear constant originally:

$$W = \frac{K}{H}$$  \hspace{1cm} (4)

where, $K$ is a dimensionless constant which is calculated in equation (2), and $H$ is a hardness measure of the softest surface. The input to EDEM model is the wear constant $W$ with units $1/\text{Pa}$. The calculated wear constant $W$ of lining materials NM500, NM400 and 16Mn is $1.58 \times 10^{-11}, 4.73 \times 10^{-11}, 1.23 \times 10^{-10}$ (1/Pa) respectively.

As the equation (3) predicts a volume of material to be removed, the wear depth per element is used in EDEM model:

$$d_e = \frac{Q}{A}$$  \hspace{1cm} (5)

The parameters in the EDEM skip model are determined using the above calculated experimental wear results and the related iron ore characteristics.
3.2 Simulation of mining skip loading and charging

The virtual 3D model of metallic skip is built in solidworks and imported in EDEM software, from which the iron ore generated by the particle plant with the total loading weight 50t. In the initial structure design of this metallic skip, the liner thickness of impacting area and its neighbor region are 40mm with liner thickness of other area 25mm.

The iron ore particle is shaped with irregular surface to reflecting the mining field reality. For the underground mining lifting system, the huge iron ore block will firstly be crushed into the smaller one before it loading into the skip container. The diameter of particle in the EDEM model is set as random variable with the mean value of diameter 200mm. The particle size of bulk materials is approximately treated as the normal distribution with the upper and lower limit of particle diameter 300mm and 100mm respectively. The angle of loading chute is 45° for creating streamline flow during loading the bulk materials.

The wear behavior of liner can be simulated during skip loading and dumping iron ore. The region of the liner facing to the particle flow will be hit by large impacting force which causes the severe wear of liner. The particle flow of iron ore appears in scattered state when loading the particle from chute into the skip container. Thus, the two liner plates adjacent to the loading chute also have the severe wear status which the scattered particle may impact on this region. The majority particles will move along the vertical skip wall and pile up in the skip container.

![Iron ore loading](image1)
![Liner wear](image2)
![Liner wear distribution of front wall](image3)

Fig. 6 Simulation of Liner Wear during the skip loading

Fig. 6 shows the wear behavior of the metallic skip during loading iron ore. The abrasion wear is predominantly located in the upper area of the skip container while some other abrasion area extending to the side liner plate. The heaviest wear region appears concentratedly in the impacting section of the skip which the particle flow directly acting on it. Specifically, there is a heavier abrasive wear region on the liner at the level of the lower discharge ports. This indicates that the particles slide at high speed along this bottom liner plate which placed in inclined angle and then accumulate on the arc gate surface.

As simulation shows, the wear mechanisms of metallic skip are mainly two types which are the impact wear and abrasion wear. Impact wear occurs when bulk material freely falling in the skip container exhibits high impacting force on the skip wall. Abrasive wear is caused by the frictional rubbing on the surface of the lining material when the bulk material moving along the skip wall at certain speed.

4. Results and Discussion
4.1 Wear depth of liner during skip loading and dumping
In reality, the total liner surface of skip is divided into different dimension area for easy replacement. In this study, the liner on each wall of skip is divided into 12 blocks which have different dimension size shown in Fig.7(a). The wear depth of each area can be estimated by equation (4) using EDEM model.

Each value shown in Fig.7(b) represents the maximum wear depth of each liner block respectively, of which the skip liner using lining material NM500. It includes two parts which are the wear depth caused by the loading and dumping process respectively. The liner belonging to the back wall of skip will impacted by the loading particle flow directly. This impacting action will continue until the end of skip loading. The liner blocks (marked by No.b5-No.b8), located in the impacting area and bottom liner(No.b12) which belonging to back wall shown in Fig.7(a), have much more serious wear behavior, the wear depth of these liner blocks is much heavier than the other liners. The liner blocks (No.f1~No.f8) located on the front wall of skip have slight wear where the impacting force has no or little effect on this area.

The loading particle flow will scatter in different direction which some particles may be freely falling on the skip bottom, and then accumulate on the surface of arc gate. Two liner plates (No.r7, No.l7), located on the bottom area of right and left wall, will have heavier wear due to these splashing particles. The same situation is also to liner blocks No.f9~f11. In addition, Liner (No.r8-No.r12, No.l8-No.l12), located on the right and left wall, are adjacent to impacting area which belongs to the back wall. These liner blocks have much heavier wear status than that of liner blocks (No.r1-No.r6, No.l1-No.l6) during the skip loading.

Apart from the particle flow impacting on the back wall, some particles may scatter in different direction and freely fall during loading the skip. These freely falling particles impact on the skip bottom liner. During the bulk particle dumping, the arc gate will open under the action force and generates the flow chute to guide the particle dumping efficiently. In generally, the bottom liner No.b12 located on back wall has the severe wear behavior considering the loading and dumping process.

For the bottom dump skip, the particle will flow out of skip container with the aid of gravity during the dumping. The particle will cause sliding wear on the totally liner surface especially those liner neighbor to the back wall.

4.2 Liner service life analysis

The liner located on the impacting area will have severe wear and should be replaced frequently. The simulated result mentioned above is identical to the actual wear status of skip liner. Fig.8 shows the wear depth of liner located on the back wall using different lining materials (NM500,NM400, and 16Mn). The liner on the upper area of back wall has the slight wear behavior, the wear depths of liner using different lining materials has no
difference from each other.

For the liner located on the impacting area and bottom area, the wear of liner using different lining materials will be different apparently. Liner using NM500 material has the minimum wear while the liner wear using NM400 is a little heavier and 16Mn is the heaviest.

The liner block will be replaced if the thickness of liner somewhere is less than 5mm. In order to predict the liner wear evolution, it is assumed: the loading capacity of skip is 50t, the hoisting system operating 12h/per day, the hoisting iron ore weighted about 9000t/day.

The liner thickness is set as 20mm instead of initial value 40mm for liner lightweight. Fig.9 shows the service life of some liner with heavier wear situation based on the wear depth of liner. Considering the liner’s cost and service life, the liner located on the upper area uses 16Mn, the material of liner located in the impacting area should choose the NM500. For the liner block with the heaviest wear behavior, the liner choosing NM500 material can be used for 68 days while NM400 for 23days, and 16Mn only for 8.5days.

The loading and dumping particle flow has different action on the liner which located on the different place. Due to the wear depth of each liner block having different value, the liner thickness and lining material will be chosen suitably based on the liner wear distribution. NM500 lining material generally outperformed the other two materials (NM400 and 16Mn) in terms of abrasive wear resistance. The service life of liner can be guaranteed while lightweight the liner by reducing the thickness of liner with high wear resistance steel.

The liner with the same material is divided into 3 parts shown in the Fig.10. The material of skip liner in high impact and high wear areas marked with symbol 1 can use the high wear resistance material, such as NM500, or cast high manganese. The area marked 2 with relatively heavier wear depth choose the NM400, NM360, or the liner thickness in this area can be reduced. The liner wear in the area with symbol mark 3 is slightly comparing to other area’s, thus the lining material 16Mn or other steel plate is suggested to arranged, or the thickness of liner marked 3 can be reduced furtherly without influencing their service life.
5. Conclusion

The wear performance of liner of metallic skip is studied comprehensively by experimental and simulation method. The wear resistance characteristic of three lining materials has been tested by ball-to-disc wear experiment. The lining material has the great effect on the liner wear rate and wear coefficient.

A DEM model of skip is developed to simulate the wear of lining surface during the iron ore loading and dumping. The heaviest wear area of liner locates on the impacting area where the particle flow directly acting on it. The simulation shows that the liner can be divided into 3 or more different type which the liner with the same type has almost the same wear behavior and similar wear depth.

The lining material with different wear resistance is suggested based on the local wear distribution of liner. The wear depth of liner using material NM500 is much smaller than NM400, 16Mn. The suitable liner thickness is recommended to the different part liner which it can lightweight the skip and extend the life service.

In the further study, the wear distribution of liner will be predicted by meshing the geometry of the liner, and the liner evolution can be investigated relatively accurately for determining the liner thickness.

Acknowledgements

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Reference

Kumar R, Prakash B, Sethuramiah A. A systematic methodology to characterise the running-in and steady state wear


LEVERAGING STATE OF THE ART TECHNOLOGY TO PROMOTE SUSTAINABILITY IN COAL MINING: A CASE STUDY OF COAL INDIA LIMITED

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ABSTRACT

India is well endowed in terms of minerals. The mining sector is a strong contributor to the growth of national economy. There is a strong correlation between growth in mining industry and the manufacturing sector; making it a catalyst for the growth of basic industries, such as power, steel, cement, infrastructure, etc. Mining was important in primeval periods and will remain important in future for the growth of our nation.

Coal has always controlled the economic status of mankind. The coal mining industry plays a major role in the sustainable development of humankind and the global economy. Across the world, it contributes significantly to the government exchequer and thereby to the growth of other industries, infrastructure, generation of employment, quality of governance and quality of life.

Coal India Limited (CIL) is the largest coal producing company in the world. Coal India Limited and its subsidiaries are committed to their role in the growth of the Indian Economy, in order to keep India as one of the most dynamic growing economies in the world.

CIL, since inception, has always acknowledged the urgency and necessity for usage of State-of-the-Art technologies for mining growth. CIL has always encouraged innovation in various interrelated fields to promote sustainability in coal mining. Primarily, stress has been laid on mechanization and automation in mines. This helps to explore and prove the reserves of different minerals with precision. The commitment to mine and utilize the available reserves efficiently and economically with safety has been strengthened with the onset of innovative technologies. International cooperation for utilization of proven and advanced technologies in the fields of use of large equipment, seismic monitoring, satellite imagery, upgraded software and other means is beneficial in bringing excellence to the industry in planning large capacity mines, underground and open cast. These have also been used to fulfill commitment to save and preserve environment as well as to ensure the restoration of environment and ecological balance disturbed due to mining activities.

In different open cast mines, “surface miner” technology and large capacity equipment with advanced technologies have been adopted. Continuous mining technologies are being introduced in underground mines. CIL has adopted several IT initiatives in a big way, viz. e-Procurement with reverse auction, e-Vehicle Tracking System, e-Performance Management System, e-Payment which have promoted efficiency and reduced waste. Coal to methanol Project has been planned. Mega mining projects are being planned with a washery, a reject based power plant and a green curtain along mine periphery.

Central Coalfields Limited (CCL), a subsidiary of CIL produced 48 MT in FY 2012 and 67.05 MT in FY 2017 registering a growth of 40% with increase of 65% in gross turnover with 1% increase in cost of production per Tonne.

KEYWORDS

Coal, Energy, Innovation, Technology, Growth

INTRODUCTION

COAL AND INDIA

Coal has a significant role in providing energy around the world. Coal has always controlled the economic status of mankind. Utilization of coal helped the transformation of civilization. Coal has written the destiny of nations. Possession and use of Coal has remained one of the reasons regarding the rise and fall of empires. The Industrial Revolution took place in England due to innovative use of coal. Coal gave an edge to the nations who started its use early. Coal still has a very significant role to play in today’s world in bringing prosperity and happiness to more than a billion families round the globe. Coal has a key role in the growing economy of India for decades to come. Coal is the preferred primary commercial energy provider because of its abundance, availability and affordability. Being reliable, it will continue to
contribute significantly towards fulfilling the need of power required for the growth of nations. Coal will continue to occupy the center-stage of India's energy scenario, considering the limited reserve potentiality of petroleum & natural gas, eco-conservation restriction on hydel projects and geo-political perception of nuclear power.

In developing countries, energy deficiency is a big problem affecting more than a billion lives. The Economic Commission for Africa of United Nations deliberated upon Sustainable Development, at its fourteenth session in April 2006 and observed that, “Energy is central to sustainable development and poverty reduction efforts.” The current per capita commercial primary energy consumption in India is about 535 kgoe/year, which is well below that of the developed countries. The economic development of any country largely depends upon its industrial progress. A set of nation-building initiatives has been taken by the Government of India to transform India into a global design and manufacturing hub. Because of these initiatives, growth rate of the nation has stabilized. To achieve the desired growth rate, there is a huge demand for power in India. Power is essential and the most important factor for industrial and business set up. Driven by the rising population, expanding economy and a quest for improved quality of life, energy usage in India is expected to rise further. Demand for coal will go up every year with the growth and development of our industries. The share of coal in India’s commercial energy supply was 55% in FY 16 and is expected to remain high at 48-54% in 2040. Coal based power generation capacity has been estimated to jump to more than 330-441 GW by 2040. The known level of proven reserves (138 BT as on 31.03.2016) may only support an annual peak production of 1.2 – 1.3 BT till 2037, with a gradual decrease thereafter.

Renewable energy sources like solar and wind are being explored, encouraged and added as energy upgrades to meet the growing demand in the country. They are viewed as supplemental but they cannot substitute coal as the preferred energy source in India. In India, coal is the largest source of energy, because of its abundance, availability and affordability. Coal is the most reliable primary commercial energy provider and the country is going to rely upon coal in the coming decades. It is Coal India Limited, which virtually fuels and empowers the power sector in the country.

Recently, the International Monetary Fund, in its January update of the World Economic Outlook: Brighter Prospects, Optimistic Markets, Challenges Ahead stated that India is forecast to grow 7.4% in 2018-19 against 6.7% this year, gaining pace to 7.8% in 2019-20. This makes India the world’s fastest growing economy in 2018 and 2019, the top ranking, it briefly lost in 2017 to China.

The estimated geological reserve of coal in India is 315.15 billion tonnes as on 01.04.2017. In India, The Coal resources of India are available in older Gondwana Formations of peninsular India and younger Tertiary formations of north-easterern region. Indian coal offers a unique ecofriendly fuel source to domestic energy market for the next century and beyond. Hard coal deposits spread over 27 major coalfields, which are mainly confined to the eastern and south-central parts of the country. The coalfields in India are located mostly in Jharkhand, Odisha, Madhya Pradesh, Chhattisgarh, Uttar Pradesh, Meghalaya, Telangana, West Bengal, Sikkim, Arunachal Pradesh and Bihar.

The yearly production of coal in India was around 6 million tons a year in the beginning of the 20th century. Now Coal India Limited has produced 554.14 million tonnes in 2016-17. In 1950, our GDP was $30 billion, which has risen to the present level of $2400 billion. During this period, installed capacity for generation of electricity has increased from 1700 MW to about 330,000 MW. Today, because of our collective effort, the whole world readily appreciates us as a superpower in the field of technical and scientific knowledge.

The energy sector has been given top priority by the Government of India as well as the State Governments. Government of India (GoI), under current leadership, is putting out all efforts to provide continued supply of electricity at cheap rates to all. Some of the initiatives taken by GoI are Power for All by 2022, Saubhagya, Ujala and others. With successful implementation of these schemes, India will surely be able to retain the status of the fastest growing economy in the world and we will be able to bring smile to a billion of faces. In 2015-16, because of the initiatives taken by Government of India, 7108
unelectrified villages have been provided electricity. Around 10,000 MW of conventional power capacity has been added over a two-year period from 2014-16.

The overall long term demand of coal is closely linked to the performance of the end use sectors. In India, the end use sectors of coal mainly include electricity, steel and cement. Demand from the unorganized small sector comprising primarily of the brick and ceramic industry is relatively large though infirm. Other industries using coal have only a marginal impact on the long term demand for coal.

In view of increasing demand of coal and reliance on coal for power generation, collective effort of the government, power producers, coal miners and service providers are necessary to ensure modern and sufficient infrastructure. To reduce reliance on imported coal and boost the domestic supply, development and expansion of coal mines in the country is necessary. Further, railways, port authority, and the coal industry need to work in close collaboration to plan development of infrastructural facilities as per requirements.

COAL INDIA LIMITED

Since inception in 1975, Coal India Limited and its subsidiaries are committed to sustainable coal mining to achieve the production targets and play their role in the growth of the Indian Economy, so that India remains the most dynamic growing economy in the world.

VISION OF COAL INDIA LIMITED

Coal India Limited (CIL) is a ‘Maharatna’ company under the Ministry of Coal, Government of India. CIL is the single largest coal producing company in the world and one of the largest corporate employers as well. The vision of CIL is to emerge as a global player in the primary energy sector while staying committed to providing energy security to the country by attaining environmentally & socially sustainable growth through best practices from mine to market. Coal India Limited plans to produce and market the planned quantity of coal and coal products efficiently and economically in an eco-friendly manner with due regard to safety, conservation and quality. The vision also extends to dedicate itself to the service of the countrymen in providing the primary commercial energy in an affordable and environmentally friendly manner. Coal India aims to be not only a valued company, but a company with values through constantly innovating on ease of doing business.

COAL INDIA LIMITED

The manpower of CIL was 3.10 lakh as on 1st April, 2017. CIL has eight fully owned Indian subsidiary companies and a foreign subsidiary company. Indian subsidiary companies are Eastern Coalfields Limited (ECL), Bharat Coking Coal Limited (BCCL), Central Coalfields Limited (CCL), Western Coalfields Limited (WCL), South Eastern Coalfields Limited (SECL), Northern Coalfields Limited (NCL), Mahanadi Coalfields Limited (MCL) and Central Mine Planning & Design Institute Limited (CMPDIL). The foreign subsidiary is in Mozambique namely Coal India Africana Limitada (CIAL). The mines in Assam, i.e. North Eastern Coalfields Limited is managed directly by CIL. Coal India Limited not only fuels the country, but it also helps to lift millions of people out of poverty. Coal India produces 84% of the country’s entire coal output. Around 73% of the entire power generated in the country is coal based.

CIL operates through 82 mining areas spread over eight provincial states of India. CIL has 394 mines as on 1st April, 2017 of which 193 are underground, 177 opencast and 24 mixed mines. CIL operates 15 coal washeries, 12 coking coal washeries and 3 non-coking coal washeries. CIL further manages other establishments like workshops, hospitals and so on. CIL has 27 training Institutes. Indian Institute of Coal Management (IICM), an excellent training center operates under CIL and imparts multidisciplinary management development programs to the executives. The major consumers of coal are power and steel sectors. Other consumers include cement, fertilizer, brick kilns and a host of other industries.

During 2016-17, Coal India produced 554.14 Million Tonnes coal. CIL achieved an off take of 543.32 MT and a gross sales of Rs 1,22,294.46 crores. The net worth of CIL in 2016-17 has been assessed at Rs 24,506.97 crores. CIL is one of the highest contributors to the government exchequer. Coal India paid
corporate taxes of Rs 8,942.7 crores to Government of India in FY 2016-17. Coal India and its subsidiaries had also paid/adjusted Rs 44,068.28 crores as Royalty, Cess, VAT, DMF and NMET and other levies. During 2016-17, Coal India as a whole earned a pre-tax profit of Rs 14,433.71 crores and a profit after tax of Rs 9,265.98 crores. Due to effective dispatch of coal, not a single power–utility was in critical or super-critical condition for want of coal during 2016-17. Further, import of coal has decreased. The shares of CIL are listed in two major stock exchanges of India, viz Bombay Stock Exchange and National Stock Exchange on and from 4th November, 2010.

**STRATEGIC APPROACH FOR GROWTH OF COAL INDIA LIMITED**

Special e-auction schemes have been introduced in 2015-16 for meeting the coal requirement of power plants. Special Forward E- Auction scheme to cater the needs of Power Plants, Exclusive E-auction scheme for non-power sector. A Special Spot e-auction was conducted once in 2016-17 with the objective for liquidating coal stock especially from the high stock mines and to provide scope for procuring coal at a competitive price by the consumers of non-specified end use. A web portal “Coal Allocation Monitoring System” was launched in 2016 with the aim to ease the conduct of business for small and medium sector consumers having annual requirement of less than 10,000 tonnes of coal. The Company’s Gross Sales turnover was Rs 1,22,294.46 crores during 2016-17.

CIL has come a long way in the service of nation because of continued innovation in different fields related to sustainable coal mining. To keep the country on the path of growth in the coming years, CIL has to maintain sustainable growth momentum in its production and off take. CIL has taken up the challenge of producing 1 Billion Tonnes coal in 2019-20. To achieve this ambitious target, the crucial issues involve three critical railway lines, mechanization through latest technology, upgrading skills of employees, speedy acquisition of land, expeditious environmental and forest clearances and fast track state level clearances are crucial for realization of 1 BT coal production. Use of State of the Art Technologies by CIL at a large scale across the company is going to play a crucial role in realization of this ambitious goal. The multi-pronged strategic approach adopted by Coal India is as follows:

(i) **Critical Railway Links** – Collaboration with State Governments & Railways- CIL in a move of “Synergy for Energy” has undertaken three major Railway Infrastructure Projects, viz:
(a) Tori – Shivpur – Kathotia New BG Line – This railway line caters to North Karanpura Area of CCL and it is planned to evacuate about 32 MTY of coal.
(b) Jharsuguda – Barpali- Sardega Rail Link relates to coalfields of MCL and it is planned to evacuate about 70 MTY of coal.
(c) East Rail Corridor and East West Rail Corridor is planned for evacuation of coal of Mand-Raigarh and Korba – Gevra Coalfields of SECL respectively. In all, it is planned to evacuate about 180 MTY of coal.

**Following initiatives are being taken for enhancing off take:**
(a) Regular co-ordination with Railway board to optimize use of logistics resources available in the subsidiary coal companies, analyzing inputs of the subsidiaries to identify alternate source for coal movement wherever and whenever required to achieve overall sectoral targets and mitigating critical fuel requirement of consuming sectors, particularly power stations.
(b) Coordination with Ministry of Coal for various long and short term policy decisions to overcome coal movement constraints for power and non-power consumers and taking operational decisions for moving coal from various sources on contingent

(ii) **Acquisition and possession of land**
In CIL, the major portion of land is acquired under the Coal Bearing Areas (Acquisition & Development) Act, 1957. During 2016-17, 3826.19 Hectares of land had been taken into possession in various subsidiaries of CIL. For sustainable mining combined efforts by CIL and Government are being made and in the changed scenario, acquisition of land has become less tenuous.

(iii) **Web based Online Monitoring System**
Web based online monitoring of coal mining projects has been started and during 2016-17, the monitoring of 69 projects costing more than Rs 150 crores and capacity 3.0 MTY has been completed. Further, CIL
has started monitoring of 67 coal mining projects costing more than Rs 150 Crores with the Project Monitoring software MS Project.

CIL and the subsidiaries upload the critical issues being faced by them on the e-CPMP portal of the Ministry of Coal. This helps MoC to follow up with State Governments and associated ministries for early approval in respect of clearances related to environment and forest.

(iv) **Technology Development**
(a) Augmentation of exploration capacity with more use of hydrostatic drills, geophysical loggers, 2D/3D seismic survey technology and Optimization of number of coring boreholes based on the complexity of geology of the block.
(b) Introduction of high capacity equipment, Operator Independent Truck Dispatch Systems, Vehicle Tracking System using GPS/GPRS, CHP and SILOS for faster loading have been planned to augment coal production from opencast mines. Further, monitoring of production will be done using laser scanners.
(c) Introduction of Continuous Miner Technology on large scale, Longwall Technology at selected places, Man Riding System in major mines and use of Tele-Monitoring techniques continued to receive priority to increase production from underground mines.

(v) **System Improvements**
Introduction of e-procurement of equipment and spares, e-tender of work and services, ERP implementation, establishment of connectivity, revision of guidelines and manuals, use of GPS for monitoring operational efficiency in road transport of coal are being implemented to improve the overall system.

(vi) **Quality and Consumer satisfaction**
(a) 2017-18 has been declared as “Quality Year” to lay special emphasis on Quality Management to ensure consumer satisfaction.
(b) Third Party sampling has been implemented
(c) Laboratories in Areas of subsidiary coal companies are equipped with 121 Bomb calorimeters for accurate analysis of coal samples. 28 labs have got NABL accreditation and for another 27 labs, accreditation process is under way.
(d) 24 Auto Mechanical Samplers (AMS) are working in subsidiary companies. Further AMSs are to be procured.
(e) 157 rail weighbridges have been installed at rail loading points to ensure that coal dispatches are made only after weighment. 569 road weighbridges have been installed for weighment of coal to be dispatched through trucks.
(f) For consumer satisfaction and to resolve consumer complaints, emphasis has been given to quality management and redressal of complaints by consumers. CIL has started online filing of complaints and their redressal. CIL has been able to resolve 99.42% complaints of consumers in 2016-17.
(g) For greater consumer satisfaction and to resolve consumer complaints, online filing for redressal of complaints has been initiated. 99.42% of consumer complaints have been resolved during 2016-17.
(h) CIL has been accredited with IS/ISO 9001:2015 (Quality Management System) and IS/ISO 50001:2011 (Energy Management System) certification on 27th October 2016.

(vii) **Safety**
Safety of miners and mines has always remained a top priority concern for CIL. No compromise is to be made on this front. CIL has taken safety issue in a scientific and holistic manner. In CIL, to ensure safety in all mines and establishments to achieve “Zero Accident” goal, many preventive measures are being pursued. Safety personnel are being given direct exposure to the best practices internationally by sending them abroad.

(viii) **Corporate Social Responsibility**
Apart from improving the quality of lives of people, Coal India’s Corporate Social Responsibility initiatives also take them along towards the company’s goal by partnering with them. While pursuing the
enhancement of coal production, CSR is being undertaken for inclusive growth of villagers and the nearby affected communities. Coal India has spent Rs 489.67 crores in 2016-17 on CSR initiatives.

(ix) **Coal handling**
(a) CIL has enhanced coal handling plant capacity of about 320 MT per annum so as to maximize dispatches of crushed / sized coal to Power Sector. CIL is supplying (-) 100 mm sized coal to all power plants with effect from 01.01.2016, except those at pit head. In addition, mobile crushers have been installed to meet the additional crushing requirement.
(b) 15 Washeries at BCCL, CCL, WCL and NCL have facilities to crush about 36.8 million tonnes.

(x) **Extraction of coal**
(a) CIL produced 554.14 MT raw coal in 2016-17 against 538.75 MT produced in 2015-16. The overall system capacity utilization for the year 2016-17 was 84.51%. It was 99.87% during 2015-16. Coal production has taken a leap of over 100 MTs in a five year span from the production level of 452.21 MT in 2012-13 to the current level.
(b) Use of modern technologies like surface Miners and Continuous Miners generates lesser airborne dust compared to conventional mining. During 2016-17, CIL produced 255.027 MT through surface miners, about 48.89% of its production from opencast mines.
(c) CIL produced 4.689 MT from underground mines through Continuous Miners. The underground production got major boost with the introduction of Longwall Technology in Jhanjhra Combined UG mine (Capacity 3.5 MTY) of ECL in August, 2016.
(d) During 2016-17, 7 Projects, each costing Rs 20 crores and above, with an ultimate capacity of 24.20 MTY and completion cost of Rs 1190.98 crores have been completed. These include three projects of CCL, one of NCL and three of MCL.

![In-pit crushing – conveying technology at Piparwar OCP, Central coalfields Limited](image-url)
Surface Miner at Mahanadi Coalfields Limited

(e) During 2016-17, 4 projects started coal production. These include three projects of WCL and one of ECL.

(f) As on 31.03.2017, CIL had 35 draglines, 658 shovels, 2783 dumpers, 936 dozers and 688 drills.

(f) Surface Miners- Emphasis has been given to maximize production of coal in mines through surface miners, wherever technically and commercially feasible. CIL is producing about 50% coal through surface miners. At present 75 surface miners are working in opencast mines.

(g) In 2019-20, CIL has planned to produce 908.10 MT coal with a CAGR 12.98% with respect to 2014-15. In 2017-18, CIL is to produce 600 MT with a growth of 8.3% over the achievement in 2016-17. In 2018-19, the targeted coal production is 773.70 MT with a growth of about 28.95%.

(h) In next 2/3 years, CIL has planned to procure 6 Draglines costing around Rs 1176 crores, 87 shovels costing around Rs 1929 crores, 515 Dumpers costing around Rs 3305 crores, 124 Dozers costing around Rs 314 Crores, 35 Drills costing around Rs 144 crores.

(xii) **Washeries**

CIL has 15 washeries having capacity to wash 36.8 million tonnes coal per year. 12 washeries are coking coal washeries with capacity to wash 23.3 MT coal per year, and 3 washeries are non-coking coal washeries with capacity to wash 13.5 MT coal per year. 22 new coal washeries and renovation of 05 existing washeries combined capacity of 123.7 MTPA are to be commissioned in near future. 13 new coking coal washeries will have the capacity to wash 41.35 MT coal per year and 9 new non coking coal washeries will have the capacity to wash 75.5 MT coal per year.

(xii) **Payment to Government**

CIL is one of the highest contributors to the government exchequer. CIL paid corporate taxes of Rs 8,942.7 crores to Government of India in FY 2016-17. CIL and its subsidiaries had also paid/ adjusted Rs 44,068.28 crores as Royalty, Cess, VAT, DMF and NMET and other levies.

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<th>Description</th>
<th>2016 – 17 (Rs Crore)</th>
<th>2015-16 (Rs Crore)</th>
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<td>Additional Royalty (MMDR Act)</td>
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<tr>
<td>DMF</td>
<td>3964.47</td>
<td>-</td>
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<tr>
<td>NMET</td>
<td>221.16</td>
<td>-</td>
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<tr>
<td>Cess on coal</td>
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<td>1590.67</td>
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<tr>
<td>State Sales Tax/ VAT</td>
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<td>2444.75</td>
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<tr>
<td>Central Sales Tax</td>
<td>1200.09</td>
<td>1144.79</td>
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### Stowing Excise Duty

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<th>Amount 1</th>
<th>Amount 2</th>
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<td>Stowing Excise Duty</td>
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<tr>
<td>Central Excise Duty</td>
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<tr>
<td>Clean Energy Cess</td>
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<tr>
<td>Entry Tax</td>
<td>283.82</td>
<td>259.37</td>
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<tr>
<td>Others</td>
<td>941.00</td>
<td>848.06</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>44068.28</strong></td>
<td><strong>29084.11</strong></td>
</tr>
</tbody>
</table>

(xiii) **Expenditures**

The overall capital expenditure during 2016-17 was Rs 7700.06 crores as against Rs 6123.03 crores in previous year. CIL has set the target of capital expenditure for the year 2017-18 at Rs 8500 crores.

(xiv) **Future Outlook**

(a) **Kayakalp Model of Governance:** Under guidance of current leadership, the priority of company’s growth has been redefined in view of the prevailing socio-economic milieu of Jharkhand. It is now centered on inclusive growth through a series of initiatives collectively known as Kayakalp Model of Governance. This dynamic model is based on five pillars:

(i) Transparent, Ethical and Philanthropic approach of management, particularly CEO.
(ii) Development of subordinates through intensive training
(iii) Enforcement of discipline through “Lead by example” theory
(iv) Innovation, Automation and State-of-the Art Technology
(v) Democratic planning and Autocratic control.

#### Kayakalp Model of Governance

With the successful implementation of this unique Kayakalp Model of Governance, Central Coalfields Limited (CCL), a subsidiary of CIL produced 48 MT in FY 2012 and 67.05 MT in FY 2017 registering a growth of 40% with increase of 65% in gross turnover with 1% increase in cost of production per Te. For sustainable coal mining, this innovative model of governance is being implemented across the Company.

(b) **Diversification in different fields:** During 2017-18, CIL planned to invest Rs 6500 crores in various projects, viz Super Critical Thermal Power Plant (STPP), Solar Power, Revival of Fertilizer Plants, Coal Gasification, Acquisition of coal blocks in India & Abroad, CBM, etc.
Global Warming: CIL gives cognizance to the spread of global warming. In view of Paris Protocol and subsequent changes across the world, CIL has decided to diversify its operations towards renewable energy sources, like solar power and clean energy sources like CMM, CBM, CTL, UCG, etc. Accordingly, CIL is in process of finalizing “Vision Document 2030” to decide future course of operation for sustainability in coal mining.

Sanction of new projects by CIL: During 2016-17, CIL Board sanctioned 8 coal mining projects for an ultimate capacity of 56.25 MTY and a total capital investment of Rs 8931.05 crores. One revised Project/ RCE was sanctioned by CIL Board for a capacity of 1.5 MTY for sanctioned capital of Rs 60.10 Crores.

Sanction of new projects by subsidiaries: During 2016-17, 11 coal mining projects for an ultimate capacity of 16.74 MTY and capital investment of Rs 3427.26 Crores have been sanctioned by Subsidiary Coal Companies. Two revised Project/ RCE was sanctioned by SECL Board for a capacity of 4.40 MTY for sanctioned capital of Rs 603.22 Crores.

Railway Projects: CIL has undertaken three major railway infrastructure projects, which will help achieving the planned growth in production and evacuation in future. These will be implemented either by railways or by JV companies. The three projects include Tori – Shivpur – Katholia New BG Line, Jharsuguda – Barpali – Sarodega Rail Link, East Rail Corridor and East – West Rail Corridor. 282 MT coal per year will be evacuated from different Areas of CCL, MCL and SECL.

Conservation of energy: Coal production in 2016-17 increased by 2.9% compared to 2015-16, but there was reduction of 1.7% in electricity consumption, from 4971.13 Million units to 4886.83 Million Units. Specific Power Consumption (kWh/T) during 2016-17 was 8.82 kWh/T vis-à-vis 9.23 kWh/T during 2015-16 with a reduction of 4.42%.

In the last three years, there has been a reduction of coal consumption for generation of one unit of electricity by 9%. Consumption of coal for generation of one unit electricity has come down from 0.69 Kg in 2013-14 to 0.63 Kg in 2016-17.

E-initiatives and system improvements: CIL has adopted several IT initiatives in a big way, viz. e-Procurement with reverse auction, e-Vehicle Tracking System, e-Performance Management System, e-Payment which have promoted efficiency and reduced waste. Introduction of e-procurement of equipment and spares, e-tender of work and services, implementation of Coal Net, establishment of connectivity, revision of guidelines and manuals, use of GPS for monitoring operational efficiency in road transport of coal have been planned to improve the overall system.

Green Initiatives: Coal India has put in all-out effort for environmental and eco-system restoration of nature to its original pristine condition. CIL has submitted Green Energy Commitment letter to MNRE for developing 1000 MW Solar Power Projects. In this regard, CIL has signed MoU with Solar Energy Corporation of India (SECI). Coal India has given its consent for floating of NIT for 80 MW Solar PV Project to SECI. In the 1st Phase, land has been identified by SECI for setting up of 25 MW Solar PV Project in the Solar Park of Madhya Pradesh Government in Neemuch Area. The power generated from the Projects will be used for captive consumption.

Plantation: Plantation and Green belt are developed through extensive tree plantation programs every year by the subsidiaries of Coal India. The subsidiaries of CIL have planted around 94.015 million trees covering an area over 37557.458 Ha till March 2017. During 2016-17, 1.66 million trees have been planted covering an area of 661.20 Ha.

Reclamation: CIL has taken up reclamation of mined out areas and the external OB dumps in a large scale. In all new mines reclamation of mined out areas are being done as per Environmental Management Plan and Mine Closure Plan, approved by Ministry of Environment and Forest. Backfilling of overburden in mine voids, topsoil preservation, storing and its use are being done in opencast mines. Concurrent reclamation and rehabilitation of mined out areas are being done in many areas. Coal India is conscious of
its commitment towards the environment and its ecosystem. The Company strives to give back to Nature, to the best extent possible, what has been pried away from her. The Company does this by taking appropriate measures to mitigate the impact of mining and associated activities in accordance with EIA / EMP of each project. Coal India has created a green wealth of about 92.35 million trees covering an area over 36,896.26 Ha till March 2016. Of this, 1.68 million trees have been planted over 719 Hectares (Ha) in FY 2016. Satellite surveillance has been adopted for monitoring reclamation activities of 50 major OCPs, producing 5.0MM$^3$ (Coal + OB) or more every year and for other OCPs once in three years. It is evident from satellite surveillance that reclaimed land area has increased by 9.63 Square Kilometer during FY2016 in 50 major OCPs w.r.t FY 2015.

**Kayakalp Vatika at Piparwar OCP**

**Fire Area in Jharia:** Study of National Remote Sensing Centre (NRSC), ISRO, Hyderabad in 2013 revealed that the fire area in Jharia Coalfields had reduced from 8.9 Sq. Km (as assessed in Master Plan) to 2.18 Sq. Km after undertaking various methods while implementing the Master Plan.

**Measures to mitigate pollution:** CIL is committed to sustainability in coal mining and gives utmost importance to protect environment. CIL takes various measures concurrently with mining operations for maintaining acceptable/ permissible limits of major physical and chemical attributes of environment namely air, water, hydrogeology, ground vibrations, noise, and land pollution. Environmental Management Plans are prepared for the mines of CIL before commencement/ enhancement of production of coal mines. The EMP is prepared in reference to the impact on existing environment and forest due to coal mining projects through Environment Impact Assessment (EIA) study of each project. Plantation in surroundings of active mining areas and along the haul roads are carried out to create green buffers/ green belts in and around the mines.

**Mode of transportation:** Transportation to thermal power stations is being carried out by rail/ series of belt conveyors. Rail heads are constructed and made available nearer to mines to reduce road transportation. Integrated CHP have been constructed for rapid loading of wagons and trucks. Tube conveyor mode of transportation is being introduced in some mines for transportation of coal to thermal power plants.

**Water Management:** For mine water management, many effective measures have been taken by CIL. The domestic effluent from major residential colonies is treated in Domestic Effluent Treatment Plants either by activated sludge method or by extended aerated lagoons. For treatment of mine water, Mine Discharge Treatment Plants (MDTP) are installed. The treated mine water is used for dust suppression, fire-fighting, plantation, washing, etc. After further treatment, it is used as drinking water standard for supply to Company Township and nearby villages. The remaining treated mine water is used mainly for agricultural use. Regular assessment of the impact of mining activities on ground water levels is carried out. Further, efforts are being made to recharge ground water within mine as well as in nearby villages through different means of rainwater harvesting.

**Noise Management:** To control noise pollution in mines and nearby areas, emphasis is being given for introduction of new technologies which extract coal without blasting, like use of surface miners, continuous miners, etc.
(xviii) **International Co-operation**
CIL has gone for international co-operation with a view to:
(a) Incorporate proven and advanced technologies and management skills for exploiting UG and OC mines, coal preparation and related activities.
(b) Exploration and exploitation of methane from coal bed, abandoned mine, ventilation air, shale gas, coal gasification, etc.
(c) Locating overseas countries interested in Joint Venture in the field of coal mining with special thrust on coking coal mining.
(d) CIL aims to acquire suitable technology in different areas of interest through international bidding. The priority areas include acquisition of modern and high productive underground mining technology, introduction of productive opencast mining technology, improvement in working underground in difficult geological conditions, fire control and mine safety, coal preparation, application of 3 D seismic survey for exploration, extraction of coal bed methane, coal gasification application of Geographical Information System, satellite surveillance, subsidence monitoring, environmental control, oversea ventures in coal mining.

Bilateral co-operation is also being encouraged for locating availability of cost effective and latest technologies in the aforesaid areas.

**Indo-US collaboration:** With Indo-US collaboration, work is going on in the field of Development of Coal Preparation Plant Simulator, Beneficiation and Recovery of Fine Coal, Underground Coal Gasification, Shale Gas, Coal Mine Methane, Dynamic Planning of Large Capacity Opencast Mines and Advanced Dry Coal Beneficiation Technology.

**Indo-EU collaboration:** With Indo-EU collaboration, work is going on in the field of Introduction of a new underground mining technology at North Eastern Coalfields in Assam. Further, collaboration is being sought in the field of reclamation practices, land management and utilization of mine voids for storage of mine water which is generally of good quality.

**Indo-Australian collaboration:** With Indo Australian collaboration, work is going on in the field of Coal Bed Methane, Ventilation Air methane, Coal Mine Methane and Underground Coal Gasification. SIMTARS in collaboration with ISM & CIMFR, Dhanbad has been engaged in mining simulation, explosion testing and mining safety training for Indian coal mines through an R&D project funded by CIL.

**Indo-Poland collaboration:** With Indo-Poland collaboration, work is going on in the field of slope stability of overburden dump (using advanced modelling technique), Dry Coal beneficiation, extraction of remnant coal pillars with surface protection, pre- drainage of coal mine methane and commercial recovery of coal bed methane (CBM) and control measures for mine fires of Jharia.

**Indo-Japanese Collaboration:** New areas have been identified to work with Indo – Japan collaboration, such as Dry Coal Beneficiation, Slope Stability Monitoring and Subsidence Measurement & Monitoring using DINSAR Technology in Jharia Coalfield.

**Indo-Russian Collaboration:** During the 21st Meeting of India-Russia Joint Working Group on Energy and Energy Efficiency, India expressed interest in technical co-operation with Russian companies in the field of Underground Coal Gasification (UCG) and resource assessment of Coal Bed methane (CBM) in distressed conditions.

**Indo-Belarus Collaboration:** Collaboration has been sought by a company of Belarus regarding trial run of 350 Tonne dump trucks for North Eastern Coalfields.

(xix) **Acquisition of coking coal assets abroad**
CIL has taken initiatives for acquisition of coking coal assets abroad, with particular focus on Australia. Australia has been chosen as the prime destination for sourcing coking coal to India.

(xx) **Import of coal**
India imported 25% of its coal requirement in FY 16, which dropped to 23% of the requirement in FY 17. Import of coal has come down and in last few years more than Rs 25,900 crore foreign exchange has been saved.
The top echelon of management at CIL has always correlated energy sufficiency in India with the great future of India. The management has a belief that all children, particularly children of poor families, must have access not only to education and health, but to quality education and quality health facilities. Under this guidance, CCL has played a significant role in facilitating these to such meritorious children, who never dreamt of having proper meals two times a day. CCL has successfully implemented CCL ke Lal, CCL ki Ladli (Lal and Ladli in Hindi indicate those who are very close to one’s heart), Sports Academy, Kayakalp Public School, Skill Development Centre and other such schemes.

**CONCLUSION**

Coal Mining is an integral part in the growth of Indian Economy. It is at the base of all the manufacturing industries. Mines must deploy state of the art technology, like surface miners, tube conveyors, silo loading, rail transport for reducing pollution and for increasing production. The railway sidings should be within the leasehold of the mines to facilitate dispatch of coal to the consumers. Coal India Limited and its subsidiaries are going for “Green Energy Hub” comprising mine, washeries, and reject based power plants. Eco-parks have been planned. It is better to transfer “Coal by Wire” than by Rail.

The present work is expected to mark a way toward the efforts put in by CIL and its subsidiaries in leveraging State-of-the Art to promote sustainability in coal mining, which is essential for the growth of Indian Economy. These measures will have long and lasting impacts on making India better and stronger. The Coal Indians believe that coal has a key role in providing energy to each and every home, so that the spread of education and health is done in a better way. The responsibilities towards the nation are bigger. The Coal Indians understand this and are committed to ensure that there is no shortage of coal in the country and to make the country self-reliant in coal.

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INCREASE OF STABILITY OF LONG-TERM ROADWAYS OF DEEP COAL MINES IN CONDITIONS OF FLOOR HEAVING

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ABSTRACT

"DTEK Pavlogradugol" PJS Company is a large modern coal mining association in the Western Donbass. The geological conditions of the company's mines are very difficult. This is due to the great depth of coal mining, the small strength of rocks, fault zones. As a consequence, extensive deformations of the rock contour, floor heaving, and destruction of the support elements are characteristic for long-term roadways. To support the long-term roadways, a large amount of labor-intensive and expensive repair work is required. Prospect of coal mining in Western Donbass associated with increasing depth. Reliable access to new coal reserves may be provided with long-term roadways. To maintain the stability of roadways, additional measures are required. Their choice should be based on the study of deformation processes in rock mass around roadways. The paper presents the results of studying the regularities of the development of geomechanical processes in a rock mass in the mines of the Western Donbass. A complex of research was carried out in the roadways of the Geroyev kosmosa mine "DTEK Pavlogradugol" PJS Company. Characteristic types of metal frame support deformations and manifestations of rock pressure are revealed. It is proposed to design of a combined support, using load-bearing capacity of the rock mass. It has been established that the use of a plugging or a spray shotcrete on a metal mesh-wire lagging in combination with a metal frame support and anchors is an effective measure to reduce metal consumption and increase the stability of long-term roadways. It is shown that it is possible to make plugging at different stages of the construction of the roadway. It is considered an effective technology for increasing the stability of roadways by filling the space after the support with hardening mixtures using a pneumatic method. The results of the introduction of a new combined support during the construction of the long-term crosscut on the mine "Samarskaya" "DTEK Pavlogradugol" PJS Company.

KEY WORDS
long-term roadway, combined support, plugging, sprayed shotcrete, floor heaving

INTRODUCTION

«DTEK Pavlogradugol» PJSC is a large modern association dealing with mining in Western Donbass. Mines of the region are characterized by very complicated operational conditions. It is stipulated by the fact of the availability of weak rocks, considerable decrease of their strength while moistening, thin-layer structure of the mass, and geological disturbance.

Due to the fact, at the depth of 200…300 m there are already intensive displacements of border rock mass, considerable support deformations and damages, rockfalls, and intensive floor heaving (Solodyankyn, Martovytskyy & Smirnov, 2015).

Further coal mining in Western Donbass is associated with the development of deep levels. Long-term roadways can provide safe access to new coal reserves. Additional measures are required to preserve long-term stability of the mine roadways.

Practices of deep mines performance show that following measures aimed at interacting “support-rock mass” system are the most effective in the process of increasing the stability of roadways: either partial or complete filling up of underpropped area; rock mass strengthening with the use of binding substances or rock bolts etc. Currently, a number of solutions are available to provide stability of long-term roadways (Shashenko, Solodyankyn, & Martovytskyy, 2012, Grigoriev, Tereschuk & Tokar, 2015, Logunova, 2015). However, each specific case should involve particular characteristics of geomechanical conditions implying a set of studies.

Objective of the studies, stated in the paper, is to substantiate effective structures of supports and technologies to support long-term roadways, taking into consideration the regularities of deformation processes in the neighbourhood of a roadway, and providing its safe operation as well as reducing in maintenance expenses.

MATERIALS AND RESULTS OF UNDERGROUND INVESTIGATIONS

To determine technological parameters of an approach to improve the stability of long-term roadways under the conditions of mines in Western Donbass, detailed studies of rock mass deformation regularities have been carried out (Vygodin & Yevtushenko, 1989).

Figures 1 and 2 demonstrate typical graphs of rock displacements while constructing main haulage roads in Geroyev Kosmosa mine (“DTEK Pavlogradugol” Company).

Analysis of the carried out studies made it possible to determine features of border rock mass defragmentation.

Consideration of rock mass border displacements helps separate the two typical periods: a period of intensive displacements and a period of steady-state displacements. Interval of intensive period is 20 to 50 days. During the period, displacement velocities of the rock border are characterized by great values as well as spasmodic and irregular changes. In the course of time, displacement velocity reduces nearing to the steady-state velocity (Figure 2). Arching loads are
registered after certain period of the support setting. In terms of argillites and aleurites, the period is 2 to 15 days; in terms of sandstones, it is 25 to 30 days.

Figure 1—Displacements of roof rocks (a), floor rocks (b) within the main conveyor drift of level 370 m: 0, 1, 2, 3 – benchmark depth, m; t – time to tampon underpropped area, days

Figure 2—Graph of displacements (1) and displacement velocities (2) of floor in the process of crosscut #3 (level of 470 m)

It has been registered during observation process that within rock mass in the neighbourhood of long-term roadways not subject to the effect of mining, several zones of destruction are formed. Each of the zones is of identical nature; besides, it is formed with certain time period. Figure 3 demonstrates a process of the destruction zone formation.

Thickness of the seams starting from external one (relative to roadway border) is on average 25, 50 and 75 cm respectively. In this context, displacements of layer one are up to 0.5 m; displacements of layer two are up to 0.2 m; and displacements of layer three are up to 0.05 m. Width of the fissures within roof rock between the layers is equal to 0.1 m, 0.05 m, and 0.01 m correspondingly; they are twice less in the floor.

Floor heaving is being developed from the side of the bedrock. In this context, depth of the active zone is about the half-span of the roadway. Walls of a roadway displace unequally inward and the process is especially visible near the floor. Zone of destructed rocks developing load on the support is gradually formed within the roof. The process is extended in time and space.

During final deformation stage, the depth of broken rock expansion into the rock mass is as follows: 7 meters and more within the roofing; and 3 to 4 meters within the roadway walls. In this context, intrusion depth of floor is 1.5 to 2.0 m.

Under the conditions of mines in Western Donbass, the process takes place at the distance of 30 to 50 meters from a face. It can be described with the help of a function of a mine face effect \( k(L) \) (Figure 4). Its values vary from a certain minimum value right near the rock face \( k(L)_{\text{min}} \approx 0.1…0.2 \) up to \( k(L) = 1 \) at the distance of 30 to 50 meters. The function reduces conventionally gravitational forces (\( \gamma H \)) within a zone of the face effect; their decrease factors into changes in geomechanical state of the rock mass; failure zone is formed (Shashenko, Solodyankyn, & Smirnov, 2015).
Area of plastic deformations (APD) is formed within a zone I. Floor heaving takes place within a zone II. Within the zone III, a process of failure of enclosing rocks with fracturing within walls of the roadway and its roof continues resulting in the increase of natural arch where the support load arises (Shashenko, Solodyankin & Gapieiev, 2009). While moving away from the face, formation of the zones is followed by plastic rock expansion within border area.

**IMPLEMENTATION OF EFFECTIVE TECHNOLOGIES TO PROVIDE STABILITY OF MINE ROADWAY**

Tamponage of underpropped area by means of hardening mixtures turned out to be the most effective technique to improve the stability of long-term roadways under the conditions of Western Donbass. The technique provides a support-surrounding rock mass interaction, several times increasing the bearing capacity of the support.

Concerning the technological factor, it is more convenient to perform tamponage operations either at the boundaries of zones I and II or within zone II (Fig. 4). In this context, operations connected with tamponage are performed longwise the mine roadway. The procedure simplifies arrangement of the activities; moreover, equipment cannot block bottom-hole area.
While implementing the technique in Geroev Kosmosa mine, joints of reinforced concrete plates were isolated mechanically with the help of AS-1 plant (Solodyankyn, Kravchenko, Prokudin, & Vygodin, 2016). That made it possible to accelerate tamponage velocity up to 120 m/day without falling behind rock tunneling.

Roof bolting in a face is provided within an area of tunneling equipment arrangement to improve roof rock stability before carrying out operations connected with tamponage (Figure 5).

Nevertheless, despite the obtained positive results, a fundamental defect of the technology should be mentioned. Underpropped area is filled up with hardening mixtures and rock mass is strengthened beyond the technical complex that is at the distance of 30 to 40 meters from the face (zone II of the mine working, Figure 4). At the stage, considerable amount of fault rocks has already been accumulated imposing increased load on the support. If the roadway crosses zones of geological fault, then the possibility of roof rock fall arises. Besides, processes of wall rocks deformation and floor heaving experience their intensification.

To preserve the strength of border rocks and to decrease a level of rock displacements at the initial stage of deformations as well as to develop “support-rock mass” system quickly, it is expedient to fill up underpropped area with hardening substances right in a face. By means of shotcrete flow or with use of another technique, hardening substance is supplied to underpropped area (Shashenko, Solodyankyn, Pozdnyakov & Pilyugin, 2012).

There are the two techniques to supply substances behind a support: axial technique and radial one. Radial filling is coating of a border line of a roadway by hardening substance. The procedure is performed through a plate with large openings. The use of specific plate makes it possible to operate both in a face and at a certain convenient distance from it.

The technique is more suitable for roadway where coal cutting takes place. General demand for underpropped area substances is their quick hardening which favours quick service of the support.

Axial technique concerning the supply of hardening substances to underpropped area is implemented from the roadway face side after each tunneling cycle. The technique helps combining the tunneling with underpropped area filling up. On the other hand, the use of axial technique for the substances supply involves accurate structuring of the process while carrying out each operation of tunneling cycle. The latter is possible if only reliable and efficient pneumatic filling up facilities are available and provisional as well as delivery process is well functioning.

Moreover, operations carried out according to the technology involve the arrangement of pneumatic filling up facilities in the neighbourhood of a face; the facilities block the borehole zone. If coal cutting is involved, the abovementioned complicates the activities.

In 2015, one of the alternatives was implemented in Samarskaia mine (“DTEK Pavlogradugol” Company) while constructing long-term roadway.

To prepare С10 seam for its mining, haulage crosscut was constructed from the level of 300 m. The crosscut crossed a zone of significant geological fault. The planned operational period of the crosscut is 15 years. The roadway will be operated within a zone of disturbed rocks; that is connected with a risk of the crosscut instability and heavy maintenance expenses.

The crosscut was constructed with the help of coal shearers. Flexible support and roof bolting were applied for ground control (Solodyankin, Hapieiev, Vygodin & Yanko (2017). To strengthen border rock mass, polymeric compositions of “MasterRoc” type were injected with the help of special “IRMA” roof bolts (Figure 6).

Underpropped area of the roadway was filled in with the help of torkret process every other meter from the side of a face by means of torkret machine AS-1P with the use of mixtures of “Techard-T” type. To reduce consumption of torkret-mixture maximally, coal shearer performed rock excavation with minimum overbreaks. After reinforced-concrete plate was placed on a frame, width of underpropped was 0 to 0.10 meters (Figure 7). Average consumption of dry mixture was 1.9 t/m.
Figure 6 – Scheme of construction and roof bolt for a haulage crosscut of $C_{10}$ seam within dangerous area of tectonic fault

Figure 7 – Torkret mixture Techard laid for the support of the roadways: a – in the sides, b – in the roof

SUBSTANTIATION OF THE PARAMETERS OF A NEW STRUCTURE OF A COMBINED SUPPORT

Nowadays, crushing complexes to use the rock while making hardening materials, such as torkret and shotcrete mixtures applied for roadway support technology, are being implemented in Western Donbass mines. Possibility to produce the materials in mines makes it possible to implement AMS-R (arch + mesh + shotcrete + roof bolt) support (Shashenko et al. 2015); in the process of the support construction, such operations as the mounting of reinforced-concrete plates as the lagging and backing of the underpropped area are eliminated.
Support structures include metal arch flexible support made of light-weighted mine profile CBII-17 (19) and metal mesh-wire lagging (Figure 8). Support frames are mounted with 1 m spacing. The underpropped area is filled with the help of shotcrete applied on the rock border through the plates meshes by means of a shotcrete plant. Shotcrete coating is applied when the roadhead has advanced by 30…50 m.

In this case, the hardening mixture penetrates through the fissures into the mass, fixing and strengthening it; actually, rock tamponage takes place. Resin-grouted roof bolts are mounted within the roadhead so that shotcrete operations may be moved away at the sufficient distance from the zone of operating mechanisms to prevent large displacements and deformations of a border mass as well as roof spalling and falling.

In the process of support mounting, it is expedient to use two layers of shotcrete. First shotcrete layer is applied a little behind the roadhead; the layer is in the form of high-elasticity material penetrating through metal mesh into the formed rock fissures. Second layer is a bearing and stiff one; it is applied much later and its layer is much thicker. While applying the second layer, it is desirable to add either reinforcing components (fibre) or high-strength fillers.

Currently, different types of fibre can be used – metallic, polypropylene, fibre-optic, nylon, and basalt. Dispersed reinforcement allows increasing shotcrete bending strength by 2…3 times. It should be also taken into consideration that shotcrete strength at the expense of its solidification is by 30…35% higher than the samples of concrete with the same composition solidified at a screed vibrator.

Shotcrete applied on the roadway surface makes the rock a monolith, strengthens its surface layer, improves the adhesion between separate rock blocks, and prevents its further foliation as well as border mass falls. Moreover, monolith layer of shotcrete prevents air and moisture access into the rock, i.e. stops weathering and moistening of the most deformed zone of the rock.

Mechanization of the process of filling the underpropped area with the shotcrete layer will make it possible to raise the rates of roadway development, improve the quality of operations, support roadway capacity; moreover, it provides long-term stability of permanent mine roadways.

Spatial metal lagging (Figure 9) are going to be used as a lagging that, along with shotcrete, will increase bearing capacity of a lagging by several times; in addition, it will reduce specific amount of metal within the frame metal support. Shotcrete should be applied in the required layer (from 5 to 15 cm) in a single action.
SUBSTANTIATION OF THE PARAMETERS OF AMS+R SUPPORT

The abovementioned model of the development of fissures in the neighbourhood of a single mine roadway (see Fig. 3) is the basis to set and solve the numerical problem of evaluating stress and strain state in the process of deformation development in the context of mine roadway movement. Software product Phase 2 of Canadian company Rocksciense has been used as the calculation instrument.

Methodology of numerical modeling involves several stages; in this context, calculation results at each stage should be adequate to in-situ measurements and correspond to the deformation model proposed earlier.

Figure 10 demonstrates general calculation scheme and the results of modeling the problem of mass deformation around a roadway. Figure 11 shows the stage of modeling the problem with the rock bolt mounting.

Number of roof bolts varied from 3 to 9. Strength of the border rock mass not disturbed with fissures is increased by 60%.

Analysis of the modeling results has shown that in this case size of a zone of inelastic deformations and the value of movements within the roadway border decreases considerably (by 1.5…2 times).

Figure 12 demonstrates the dependences illustrating the effect of the number of rock bolts – \( N \) and the distance from the roadhead – \( l \) upon the value of floor movements – \( U_n \). The dependences show that in terms of 5 mounted roof bolts, tamponage operations can be performed at the distance of 60 m from the roadhead, comparing to the distance of 30 m if no roof bolts are available. Thus, the rate of floor heaving is decreased down to 0.3…0.4 m at the expense of active bearing reaction to the rocks under deformation along the whole support contour (Shashenko et al. 2015). Moreover, it allows extending considerably the operations in terms of driving and plugging of the area behind the support within the whole mine roadway length.
CONCLUSIONS

Long-term stability of permanent roadways in terms of complicated geomechanical conditions of Western Donbass mines can be achieved by means of applying combined structures of supports and technologies which provide common operation of both support and border rock mass preserving its bearing capacity. Tamponage of the underpropped area, shotcrete coating, and roof bolting are the efficient support components in such structures. In this context, the underpropped area should be filled with the minimum lag from the roadhead. A technique of shotcrete-tamponage has demonstrated its high efficiency in the process of mine roadway development within the zone of geological disturbances.

It is expedient to apply roof bolting mounted within the roadhead zone to improve the roof stability prior to performing the operations of filling the underpropped area.

The developed combined AMS-R (arch + mesh + shotcrete + roof bolt) support is of high efficiency; during the construction of the support such operations as the mounting of reinforced-concrete plates as the lagging and backing of the underpropped are eliminated. Application of shotcrete layer on the roadway border along with the metal spatial lagging and not less than 9 roof bolts makes it possible to reduce the weight of frame support metal profile as well as to decrease floor heaving down to 0.3…0.4 m. The considered structures and supports and techniques of their mounting will allow providing long-term stability of roadways with minimum costs for repair activities during their operation.

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DRAG PRESSURE LOSS DUE TO SHAFT BUNTONS: INVESTIGATION FOR IMPROVEMENT OF BROMILOW’S EMPIRICAL MODEL

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ABSTRACT

Major portions of the mine ventilation pressure, up to 50%, is consumed in mine shafts especially in cases of deep mines. The high velocity pressures in the shafts combined with the presence of regularly spaced obstructions in the shafts contributes to this problem. Studies on the shaft pressure losses (Deen, 1991; Kempson, et al., 2013) show that the drag pressure loss caused by shaft buntons contributes to 50% to 75% of the total shaft pressure losses depending on the bunton configuration. The documented work for the estimation of the shaft bunton pressure loss is by Bromilow (1960). His work is compilation of field and lab scale studies from different published sources and formulation of empirical relationship between spacing-width ratio of buntons, (s/w) and interference factor, F. There are no subsequent works in this area for the past half century. With the availability of numerical simulation techniques and advancements in instrumentation, there is a need to revisit the empirical model which has bearing on the significant part of the mine ventilation pressure.

In this study, lab scale experiments of regularly spaced obstructions in a circular shaft are performed at different spacing-width ratios. The scale model dimensions ratios and the obstruction configurations are taken from an actual mine. The spacing width ratios of the obstructions cover the range specified by Bromilow. The buntons themselves are simulated by elements of circular cross-section. The pressure measurements are taken using high precision digital pressure instruments. Concurrently, numerical simulations of experimental parameters are performed using ANSYS FLUENT software to support the experimental results. The interference factor from the experimental results is compared with the Bromilow’s model and it is found that the interference factor values are significantly different. Bromilow’s model is a linear relationship while the empirical model for the interference factor from the experiment results is $F = 0.288 \ln(s/w)-0.3374$.

KEYWORDS

Shaft pressure losses, Shaft resistance, Buntons, Drag pressure loss, CFD in mining

INTRODUCTION

Mine ventilation is an essential part of underground mining addressing the issues of worker comfort and safety, toxic gases, dust, fire and explosions. Mine ventilation is continuous operation contributing to a major part of mine power demand. The pressure generated by the mine fan is distributed throughout the mine and determination of these pressure losses is important for planning a safe and economical ventilation system. Main intakes and returns consume major portion of the fan pressure as they handle the entire mine quantity and have high air velocity.

Typically intake and return shafts together consume more than 50% of the pressure generated by the fan. This has encouraged researchers to study the shaft pressure losses aiming at both determination (Deen, 1991; Purushotham, et al., 2010) and optimization (Kempson et al., 2013). From the studies it is seen that the drag pressure losses due to the shaft buntons amount to 50-75% of the total shaft pressure losses, depending on the bunton configuration.

Shaft buntons are horizontal members dividing the shaft into compartments. They also support and carry the shaft guides, pipes, cables and other shaft fittings. The drag pressure loss due to buntons cannot be directly calculated using the drag coefficients as the buntons are installed along the length of the shaft at regular intervals. Presently, the drag pressure loss is estimated using the Bromilow’s empirical model for interference factor, $F$ (Bromilow, 1960). The empirical model is outcome of the scale model data.
from different published sources available at that time and discrete field data from few mines. The empirical relationship of interference factor with spacing-width ratio is given by:

$$F = 0.0035 \left( \frac{s}{w} \right) + 0.44$$ (1)

Where, \(s\) is the spacing between buntons and \(w\) is the width of the bunton with respect to the projected surface area in the path of air flow. The equation is valid for spacing-width ratio in the range of 10 to 40. Using the interference factor, the combined drag pressure loss for a set of \(n\) buntons in a shaft is

$$P_{d(\text{Combined})} = F \cdot n \cdot P_{d}$$ (2)

Where, \(P_{d}\) is the drag pressure loss due to a single bunton calculated using the drag coefficient value of the respective shape of the bunton. The Bromilow’s model is dated and based on not well documented data and also there is limitation on quality of instrumentation used at that time. There are no subsequent works in this area for the past half century. The difficulty in validation of the model using field measurement is in isolation of the bunton drag pressure loss component from the total shaft pressure loss and there is no scope of changing the spacing width ratio of a working mine.

With the advancements in instrumentation there is a need to revisit the empirical model which has bearing on the significant part of the mine ventilation pressure. In this study, lab scale experiments are performed to investigate the drag pressure loss due to circular shaped buntons in a circular shaft.

**METHODS**

Scale model is used to investigate the drag pressure loss due to buntons in a laboratory setup. To compare the results from the model with the real world application, similitude conditions must be followed (Zohuri, 2015). The scale model is designed to have geometric, kinematic and dynamic similarities with an actual mine shaft. Details of the scale model, instrumentation and measurement strategies are given in this section.

**Scale model**

Shaft and bunton dimensions along with the bunton configuration and spacing distance of an intake shaft of a working coal mine (Chasnalla deep mine, SAIL, India) is used in this study. The scale model is designed to maintain a geometric ratio of 1:17 with the Chasnalla mine intake shaft. To have the dynamic similarity, the dimensionless Reynolds number (Re) of the scale model is matched with the shaft. Flow straighteners are installed to streamline the flow and have a developed velocity profile in the test section between the measurement station 1 and 2 shown in Figure 1.
The mine shaft has “I” section buntons but circular section bunton is considered in the study due to the issues with the fabrication. Friction property of the inner surface of the model shaft is established by experiment. The circular section buntons are placed along the model shaft at the specified spacing width ratios (Figure 2).

The buntons are fixed using a small nail to the side and the bunton and the duct is seamlessly joined using a filler material. This is made to ensure there is no additional drag pressure loss due to the gap between the duct and the buntons. Finally the model shaft is clamped together into circular shape and made leak proof. The assembled model shaft with buntons is shown in Figure 3.
Pressure measurement

The pressure loss in the test section is determined by measuring the total pressure and velocity pressure at the station 1 and station 2. Equal area - precise traversing method is used to measure the average total and velocity pressure at each station. Pressure at each point in the area is measured using a pitot tube and digital manometer. The DP-Calc (differential pressure calculator) manufactured by TSI will be used as the digital manometer for measurement. The instrument has an accuracy of 0.025 Pa and resolution of 0.001 Pa.

Experiments are conducted at three different Reynolds numbers (duct flow) in the range of $10^5$. This provides repetition to the experiment trials. The spacing width ratio of the Chasnala mine shaft is 20. The spacing width ratios of buntons found in mine shafts are in the range of 10 to 40. Spacing width ratio in multiples of 8 from 8 to 48 is examined to cover this range.

RESULTS

The Atkinson’s friction factor for the empty model shaft is experimentally determined to be 0.0018 kg/m$^3$ and this value is used in calculating the frictional loss in the subsequent experiments. This frictional loss is subtracted from the total pressure loss between station 1 and 2 to get the drag pressure loss caused by the buntons in the test section.

The results of the experiment carried out for different spacing width ratios and the corresponding interference factor are shown in Table 1. The Calculated drag pressure loss in the Table 1 is the theoretical drag pressure loss for a single circular bunton multiplied by the number of buntons ($n.P_d$ in Equation 2). The drag coefficient value of a circular section (1.2) is taken from the drag coefficient chart for buntons (Bromilow, 1960). Interference factor from experiment is the ratio of the drag pressure loss determined from experiment to the calculated drag pressure loss for $n$ buntons. The length of the model shaft test section is fixed and the hence the number of buntons drop drastically with increase in spacing width ratios. The Bromilow’s interference factor is calculated by substituting corresponding spacing width ratio in Equation 1.
Table 1 - Analysis of experiment results and comparison of Bromilow’s interference factor

<table>
<thead>
<tr>
<th>Spacing Width ratio (s/w)</th>
<th>No. of buntons, n</th>
<th>Reynolds number for duct flow (10^5)</th>
<th>Drag pressure loss from experiment (Pa)</th>
<th>Calculated drag pressure loss for n buntons (Pa)</th>
<th>Interference factor from experiment</th>
<th>Bromilow’s Interference factor</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>8</td>
<td>330</td>
<td>2.06</td>
<td>154.08</td>
<td>561.78</td>
<td>0.274</td>
<td>0.468</td>
</tr>
<tr>
<td></td>
<td></td>
<td>1.58</td>
<td>95.82</td>
<td>328.76</td>
<td>0.291</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>1.03</td>
<td>39.84</td>
<td>140.60</td>
<td>0.283</td>
<td></td>
</tr>
<tr>
<td>16</td>
<td>166</td>
<td>2.31</td>
<td>152.97</td>
<td>353.22</td>
<td>0.433</td>
<td>0.496</td>
</tr>
<tr>
<td></td>
<td></td>
<td>1.65</td>
<td>88.97</td>
<td>181.40</td>
<td>0.490</td>
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</tr>
<tr>
<td></td>
<td></td>
<td>1.08</td>
<td>36.21</td>
<td>77.34</td>
<td>0.468</td>
<td></td>
</tr>
<tr>
<td>24</td>
<td>110</td>
<td>2.19</td>
<td>123.40</td>
<td>211.51</td>
<td>0.583</td>
<td>0.524</td>
</tr>
<tr>
<td></td>
<td></td>
<td>1.69</td>
<td>70.97</td>
<td>126.21</td>
<td>0.562</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>1.08</td>
<td>29.26</td>
<td>51.06</td>
<td>0.573</td>
<td></td>
</tr>
<tr>
<td>32</td>
<td>82</td>
<td>2.32</td>
<td>118.36</td>
<td>176.30</td>
<td>0.671</td>
<td>0.552</td>
</tr>
<tr>
<td></td>
<td></td>
<td>1.77</td>
<td>68.72</td>
<td>102.97</td>
<td>0.667</td>
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<tr>
<td></td>
<td></td>
<td>1.15</td>
<td>27.85</td>
<td>43.17</td>
<td>0.645</td>
<td></td>
</tr>
<tr>
<td>40</td>
<td>68</td>
<td>2.45</td>
<td>112.38</td>
<td>163.36</td>
<td>0.688</td>
<td>0.580</td>
</tr>
<tr>
<td></td>
<td></td>
<td>1.83</td>
<td>66.28</td>
<td>90.57</td>
<td>0.732</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>1.19</td>
<td>26.97</td>
<td>38.50</td>
<td>0.700</td>
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<tr>
<td>48</td>
<td>54</td>
<td>2.42</td>
<td>97.84</td>
<td>125.93</td>
<td>0.777</td>
<td>0.608</td>
</tr>
<tr>
<td></td>
<td></td>
<td>1.84</td>
<td>56.58</td>
<td>73.03</td>
<td>0.775</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>1.19</td>
<td>23.38</td>
<td>30.55</td>
<td>0.765</td>
<td></td>
</tr>
</tbody>
</table>

The experiment trials at different Re at a specific spacing width ratio results is repeatable. The variation is within 5% and this includes other factors such as minor leakage and measurement errors.

**DISCUSSION**

The Reynolds number of the shaft is of the order of $10^6$ and the model Reynolds number is of the order of $2*10^5$. As these Re are in the highly turbulent region, the variation in the Re is not expected to have a significant effect on the model results.

As the interference factor is dimensionless, the experiment results can be directly compared with the Bromilow model without any scaling. The experiment results and the comparison with Bromilow model is shown in Figure 4. The Bromilow model is for the spacing width ratio in the range 10 to 40 and shown in dotted line.

Increase in interference factor implies increase in drag pressure loss caused by an individual bunton. The graph shows that interference factor increases with the spacing width ratio. But for a fixed length of shaft, as the number of buntons decreases with spacing width ratio, the total drag pressure loss in the shaft steadily decreases with spacing width ratio. Hence a higher spacing width ratio is preferred to have minimum drag pressure loss.

From the Figure 4, we can observe that the interference factor from the experiments is close to the Bromilow’s interference factor for the spacing width ratio of 16 and deviates for other spacing width ratios.
This suggests that the Bromilow’s interference model needs improvement and better model is required for prediction of this interference phenomenon.

Bromilow’s model is a linear relationship while the empirical model for the interference factor from the experiment clearly shows a nonlinear trend. For spacing width ratios other than 16 and 24, the difference between interference factor from experiment and the empirical model is more than 15%.

![Graph comparing interference factors](image)

**CONCLUSION**

The empirical model for the relation between interference factor, $F$ and spacing width ratio, $(s/w)$ found from the experimental results is of the form

$$F = 0.288 \ln(s/w) - 0.3374$$  \hspace{1cm} (3)

The above proposed model is for circular cross-section bunton and for specific bunton configuration. Further experiments with different shapes and configuration of buntons are required for a generalized prediction model for interference factor. Such experiments are in progress along with the comparison of experimental findings with CFD simulations.
REFERENCES


DESIGN OF THE SHAFT LINING FOR DEEP SALT DEPOSITS OF SIGNIFICANT THICKNESS

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Key words: shaft lining in salt, mining shafts, shaft sinking

ABSTRACT

The Polish Copper Belt is situated in the Fore-Sudetic Monocline and the North Sudetic Basin in the Lower Silesia. The characteristic feature of the northern part of the copper ore deposits is the presence of the rock salt layer in the Zechstein Kupferschiefer formation, located under 1000 m below the surface. Thickness and depth of the salt deposit layer is variable, increasing with the ore body’s dip direction. It causes the thickness of the salt deposit exceeds 100 m at locations of new shafts.

The shaft lining design must take into consideration the geomechanical properties of the rock mass. The support challenge encountered when excavating through the salt deposits is that of time dependent (creep) deformation control. The deformation is radial, causing the shaft excavation to close up over time. Under these conditions, shaft lining designers are faced with the challenge to create innovative solutions that will be able to protect the shaft excavation permanently, effectively and safely.

The paper will present the authors’ own experiences with the existing shaft linings in the presence of salt layers, as well as a new design solution for permanent shaft protection in a way that does not require its cyclical reconstruction.

INTRODUCTION

Kombinat Górniczo-Hutniczy Miedzi (KGHM) engages in the extracting of copper ore deposits located in the south-western part of Poland. KGHM operates in the Legnica-Głogów Copper Belt (LGOM) placed on the southern edge of the Fore-Sudetic Monocline. Three basic stratigraphic units have been distinguished here: the metamorphosed Proterozoic-Paleozoic substrate, Permo-Triassic formations, which gradually dips in the north-east direction, and the sub-horizontally thick layer of the Cenozoic sediments. In the southern part of the Fore-Sudetic Monocline, two hydrogeological complexes have been identified:

• Cenozoic complex, including loose Quaternary and Paleogene-Neogene formations, 300-400 m thick;
• Permo-Triassic complex, occurring in solid rocks of Bunter, Zechstein and Rotliegend, with the thickness growing in the NE direction and exceeding 1,000 m in the northern part of the area.

The copper ore deposits dip monoclinally at the depth from several hundred to 1,500 m. They are stratoidal and occur in sedimentary rocks of Zechstein with a diversified thickness reaching up to several meters. In the ore series fault zones are present with displacements up to several dozen meters.

KGHM Polska Miedź S.A. has currently the copper ore extraction concessions for 6 deposits: "Lubin-Malomice", "Polkowice", "Rudna", "Radwanice Wschód", "Sieroszowice" and "Głogów Głęboki-Przemysłowy" (GG-P) (Figure 1). The LGOM copper ore deposits have been developed by 30 shafts so far with the depth from 500 to 1,250 m and the 6 or 7.5 m diameter. Two shafts were liquidated in 2008-2009 so only 28 operate at the present time.

The 31st shaft is currently under construction. It is located in the central part of the "Głogów Głęboki-Przemysłowy" mining area with the target depth of 1340 meters and the diameter of 7.5 meters. The GG-1 shaft sinking is expected to be completed by the end of 2020. It will be the deepest of all shafts explored up to now in the Copper Basin.

The characteristic features of the rock mass surrounding the northern part of the copper ore deposit are the deepest location of copper-bearing rocks from all presently extracted deposits in LGOM and the rock salt layer of considerable thickness at the depth exceeding 1000 m occurring in Zechstein formations. Its thickness and depth increase in the dip direction of the copper ore deposit. For this reason, the thickness of the rock salt layer exceeds 100 m significantly in places where new shafts are designed.
Less than 270 million years ago, the climate in Poland was hot and dry. The shallow Zechstein Sea flowed in from the north-west and stopped on the mountain ranges: the Sudetes and the Świętokrzyskie Mountains. Cyclical fluctuations in sea level caused the development of characteristic rock sequences. Seasonal loss of connection between the Zechstein Sea and the global ocean made the sea turn into the highly steaming lake. This led to the precipitation of large rock salt deposits, as well as calcium and magnesium carbonates (calcite, dolomite) and calcium sulphates - anhydrite and gypsum.

The Zechstein Sea was associated with copper and silver deposition. Those metals have been accumulated as a result of the metalliferous solution flow through the contact rocks between Rotliegend and Zechstein (sandstones, shales and dolomites), which caused their oxidation and zonal distribution.

Within the LGOM area, above the copper ore deposit, the rock salt layer was formed during the Permian period. It builds one of the lithological units of the Zechstein cyclothem called Werra (P21). The deposit is considered as the irregular WNW-ESE trending layer, smoothly dipping to the NE at an angle of 3° ± 8° (locally up to 15°). Its thickness ranges from 0 m from the S and SE side, to 186 m to the N and NW side.
The Oldest Rock Salt has been developed in form of a diversified complex of layers, in which the main component is halite. The base part of the complex is laminated of bright pure salt with laminas of the thickness varying from 3 to 30 cm and darker or black anhydrite with laminas of the thickness varying from 1 to 10 cm. Most often salts in this part of rock complex have a multi-grained medium- and coarsely-crystalline, in some parts porphyritic, structure. Above the laminated salts there are pure and slightly contaminated salts, which are the major part of the salt complex. They have multi-grained medium- and coarsely-crystalline structure, massive texture and milky or transparent color. The crystal sizes usually range from 0.5 to 0.7 cm. The main contamination of these salts is anhydrites in the form of breccia, as well as regularly or irregularly distributed loam dopants.

Rock salt is a medium strength and low elasticity rock, having rheological properties. The impact of time on salt behavior under load becomes evident in the creep phenomenon (with constant load the strains increase) or in the strain relaxation phenomenon (with constant strains the stresses decrease). In addition, share of elastic deformation in the total salt deformation is small. When the yield strength is exceeded, there is a process of strains continuous increment in the rock salt, i.e. the transition from the creep stage to the flow stage of the material. The creeping process of a salt rock mass is influenced by such factors as the stress concentration and value, temperature and humidity.

The rheological properties of the salt massif are usually manifested in the convergence of underground excavations located within it. In order to predict the scale of deformation, one should estimate the parameters of the constitutive equation of the rheological model describing the behavior of the specific salt type. It can be done on the basis of long-term tests or measurements such as:

- laboratory creep tests with constant load,
- in situ convergence measurements in excavations located in the salt rock mass.

INFLUENCE OF THE SALT ROCK MASS BEHAVIOUR ON THE PROCESS OF SHAFT LINING DESIGN

The rock salt layer of considerable thickness is present in the profile of 7 shafts located in the mining areas: (OG) "Sieroszowice" (shafts SG-1, SG-2, SW-1 and SW-4), OG "Rudna" (shafts R- IX and R-XI) and OG "Głogów Głęboki-Przemysłowy" (shaft under construction of GG-1) (Table 1).

In the LGOM shafts, in the section passing through the rock salt, various lining designs have been implemented so far. The shaft lining constructions are dependent on the deposit depth and the thickness of the rock salt layer.

In different shafts the various types of lining were built in the salt section, such as:

- in the SG-1 shaft - masonry-concrete lining and concrete lining with an air gap behind,
- in the SW-1, SG-2 and R-IX shafts - tubing lining with a 0.6 m thick concrete backfilling (Fabich et al., 2016).
- in the R-XI shaft - monolithic concrete lining with a thickness of 0.6 m
- in the shaft SW-4 - bolts-shell lining reinforced with steel profiles V25.

In the R-XI shaft the concrete of C25/30 strength class and W10 waterproof resistance has been used. In addition, there are the styrofoam blocks behind the concrete wall, which have been put into a ring and connected through solvent-free glue. The empty space between the rock mass and the styrofoam ring has been filled with styrofoam plates. Styrofoam blocks have been assembling from bottom to top together with applying a concrete mix with the use of a 1.0 m high rearranged formwork. In order to protect the concrete lining against possible tensile forces, reinforcement has been used in the form of a mesh with dimensions of 2.4 x 1.5 m and 2.1 x 1.5 m, distributed uniformly around the shaft perimeter (Fabich et al. 2016).

In the shaft SW-4, where the depth of the salt layer and its thickness is the largest from whole existing shafts in LGOM, a bolt-shell lining reinforced with V25 steel profiles has been used (Fabich et al., 2009). Three element system including a spraying system consisting of a polyurethane undercoat with a thickness of ~ 2 mm, a tensile mining mesh and a membrane with a thickness of ~ 5 mm has been applied as a preservative shell protecting the salt wall against the atmosphere in the operating shaft. The tensile mining mesh with a load capacity of 50 kN/m has been used to reinforce the system against the salt overhangs arising as a result of plastic deformation of the
The massif. It was stabilized to the shaft wall with Hilti nails. The membrane was pressed to the shaft wall with yielding steel arches (every 0.75 m) and fixed with cuttable resin bolts in which, from the depth of 1099.2 m, the length of the rod was reduced up to 0.5 m due to unfavorable working conditions in the salt massif. The part of the bolt rods projected more than 200 mm beyond the shaft wall was equipped with two washers separated by a rubber hose and a nut. The nuts were screwed on the entire thread height eliminating damage to the washers.

The yielding steel arch lining was applied as the additional support in order to take up deformations resulting from the creeping process of the salt massif. It was constructed of the V25 steel profiles with a spacing of 0.75 m. The individual arches were connected with each other by RSM tubular struts of 750 mm length and fixed to the salt wall with two screw anchors of 1.8 m length. All steel elements of the yielding lining were protected against corrosion by hot-dip galvanizing with a protective layer thickness of 70 μm. In addition, the arches were made of steel of more than three times higher corrosion resistance to saline water compared to standard steel types.

### Tab. 1. The depth of rock salt layer with the type of lining used in the LGOM shafts on the salt interval

<table>
<thead>
<tr>
<th>Mining area</th>
<th>Shaft</th>
<th>Rock salt interval</th>
<th>Rock salt thickness</th>
<th>Shaft lining</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sieroszowice</td>
<td>SG-1</td>
<td>933.5 - 951.5</td>
<td>18</td>
<td>masonry-concrete and concrete lining with air gap</td>
</tr>
<tr>
<td></td>
<td>SG-2</td>
<td>957.5 – 987.1</td>
<td>29.6</td>
<td>tubing lining with concrete backfilling</td>
</tr>
<tr>
<td></td>
<td>SW-1</td>
<td>863.4 – 875.2</td>
<td>11.8</td>
<td>tubing lining with concrete backfilling</td>
</tr>
<tr>
<td></td>
<td>SW-4</td>
<td>1026.3 – 1181.8</td>
<td>155.5</td>
<td>bolts-shell lining reinforced with steel profiles V25</td>
</tr>
<tr>
<td>Rudna</td>
<td>R-IX</td>
<td>965.6 – 986.6</td>
<td>21</td>
<td>tubing lining with concrete backfilling</td>
</tr>
<tr>
<td></td>
<td>R-XI</td>
<td>1125.4 – 1137.0</td>
<td>11.6</td>
<td>concrete lining with styrofoam backfilling</td>
</tr>
<tr>
<td>Głogów Głęboki-Przemysłowy</td>
<td>GG-1</td>
<td>1145.7 – 1215.2</td>
<td>69.5</td>
<td>5 layer lining: 2 tubing lining layers with concrete backfilling and yielding foam backfilling</td>
</tr>
</tbody>
</table>

The technology of shaft sinking and lining application in the salt interval depends on the lining construction. Generally, the salt walls were extracted by blasting, except the SW-4 shaft. The KDS-2 shaft-sinking roadheader was used there due to the large thickness of the salt layer (over 150 m) and the necessity of obtaining a regular circular cross-section of the shaft because of the specific lining construction. This type of roadheader had been previously applied to excavate the frozen Cenozoic layers (mules, sands, gravels, brown coals, silts) in the depth up to about 400 m. It was first time in LGOM when it was used to sink the shaft below 1000 m of its depth.

The lining design process in the SW-4 shaft on the salt section was one of the most difficult ones that occurred in shaft construction history in LGOM. It resulted from the natural, rheological properties of rock salt and the considerable depth of the salt deposit with a very large thickness. The designed shell lining reinforced with steel arches was not fulfilled its function. After a few years it was necessary to reconstruct it due to the increased convergence of the shaft alongside the salt interval.

Based on the experience associated with the operating shafts with the salt walls, especially in the SW-4 shaft with the largest thickness of the rock salt layer in LGOM, it has been considered to design an innovative lining construction that would guarantee safe conditions throughout the lifetime of the shaft.

Since the specific function of the GG-1 shaft (hoisting and ventilation) absolutely excludes any operation stops intended to possible rebuilding, it was necessary to design lining that would be able to fully carry the expected pressure resulting from the salt massif creeping. The lining construction should also provide the safe operation of the shaft until the "Głogów Głęboki-Przemysłowy" deposit will be completely depleted. The anhydrite interlayers, particularly visible in the region of salt-anhydrite breccia in the GG-1 shaft profile, may result in some difficulties in maintaining the stability of the shaft walls. Anhydrite rocks splitting as a result of salt creeping determines the use of the additional local-placed support.

The presence of the salt layer in the shaft profile therefore determines the design process of its lining. It is very important to assess the behavior of the salt massif, especially immediately after shaft sinking. At the same time, the shaft lining construction should be based on the solution that fully transfers the rock mass pressure. Due to the fact that the salt creep process is started immediately after the shaft walls unveiling, it is very important to
determine the proper diameter of the shaft, so that at the time of applying the final lining it has at least a nominal value. The further part of the article presents the details related to the design and construction of the shaft lining on the salt section in the GG-1 shaft, which, according to the authors, would guarantee safety throughout its whole lifecycle.

**METHOD FOR EVALUATION OF THE SHAFT BREACH DIAMETER IN THE SALT MASSIF**

The general model of salt massif behavior in the LGOM area has been determined based on real convergence measurements which were conducting from February 1, 2013 to June 11, 2014 at 8 levels of the salt section in the SW-4 shaft profile. In the purpose of predicting creep rate of the rock salt deposited in the overburden of copper ore deposits on the Fore-Sudetic Monocline, the classic linear viscoelastic Burgers model was fitted to measurements data from SW-4 shaft using statistical analysis software Statistica v.10. This four element statistical model is a combination of two others, the Maxwell model and the Kelvin-Voigt model (connected in series) (FLAC, 2016), in the following form:

\[
\frac{u(t)}{\sigma_p} = a \left[ \frac{t}{2G_M} + \frac{t}{2\eta_M} \left( 1 - \exp \left( -\frac{t}{2G_K} \right) \right) \right]
\]

\(u(t)\) – one-sided horizontal convergence of the side wall in time \(t\), m,
\(\sigma_p\) – primary vertical stresses in salt rock mass, MPa,
\(a\) – radius of shaft breach, m,
\(t\) – time from the salt side wall unveiling on the specific depth of the shaft, day,
\(G_K, G_M\) – shear moduli of Kelvin body and Maxwell body, respectively, MPa,
\(\eta_K, \eta_M\) – viscosity coefficients of Kelvin body and Maxwell body, respectively, MPa/day.

The indicators of the creep rate of rock salt are the Kelvin and Maxwell body shear moduli as well as the viscosity coefficients of those two elements. Their estimation was performed using the Lavenberg-Marquardt’s non-linear least-squares method, which is an extension of the Gauss-Newton algorithm (StatSoft, 2017). Consequently, the 8 sets of creep model parameters were evaluated for different depths on the salt section in the SW-4 shaft, assuming that the initial radius of the shaft breach was 5 m, and its convergence was uniform over the entire circumference.

The obtained statistical models, allowing to determine the creep rate of the salt medium at various depths in the SW-4 shaft, were calibrated in the FLAC 2D software by numerical simulation of the shaft breach convergence increasing in time. Numerical calculations were performed under plane strain conditions. The axisymmetric model of the stratified rock mass consisted of gray and dark gray anhydrite, anhydrite and clay breccia, rock salt and gray and gray-beige anhydrite (Fig. 3) was implemented for analysis in accordance with the actual system of rock layers in the vicinity of the SW-4 shaft. The zero displacement boundary conditions were defined at the lower edge of the model. The upper edge of the model was loaded with vertical pressure reflecting the influence of overburden rocks. Variable horizontal pressure changing with the depth and conditions of the rock mass was applied to the lateral edge. It was assumed that all layers except rock salt were considered as elastic-plastic materials. The salt rock layer was divided into 8 intervals, to which the appropriate parameters of the Burgers model were assigned, estimated using the Lavenberg-Marquardt method, adjusted to the data from subsequent measurement stations in the SW-4 shaft.
Calibration of statistical models describing the behavior of the salt massif was carried out by comparing the values of horizontal displacements measured at individual measurement stations in the SW-4 shaft to the values of horizontal displacements calculated in FLAC 2D in the points of the numerical model corresponding to the location of measurement stations. The values of the calculated horizontal displacements were read for the time points corresponding to the dates of measurements carried out in the SW-4 shaft. Timepoint 0 means the moment of salt wall unveiling at the level of the measurement station in the shaft, i.e. the moment at which the creep process begins. Figure 4 presents the example of comparison of real and calculated one-sided horizontal displacements of the shaft breach obtained at measurement station No.1.
Fig. 4. Comparison of measured and calculated in the FLAC 2D shaft wall displacements in the SW-4 shaft

Eventually, only a slight modification of the viscosity coefficients and shear moduli values estimated using the Lavenberg-Marquardt method have allowed to obtain a good agreement between shaft wall displacements calculated in the FLAC 2D with measured at all 8 measurement stations in the SW-4 shaft.

Tab. 2. Parameters of the Burgers model after calibration separately for 8 measurement stations on the salt wall in the SW-4 shaft

<table>
<thead>
<tr>
<th>Benchmark N°</th>
<th>Benchmark depth</th>
<th>Shear modulus of Kelvin’s body</th>
<th>Viscosity coefficient of Kelvin’s body</th>
<th>Shear modulus of Maxwell’s body</th>
<th>Viscosity coefficient of Maxwell’s body</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>H [m]</td>
<td>$G_K$ [MPa]</td>
<td>$\eta_K$ [MPa*doba]</td>
<td>$G_M$ [MPa]</td>
<td>$\eta_M$ [MPa*doba]</td>
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<tr>
<td>1</td>
<td>1060,70</td>
<td>353</td>
<td>21 991</td>
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<td>144 935</td>
</tr>
<tr>
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<td>353</td>
<td>5 651</td>
<td>1 690</td>
<td>125 706</td>
</tr>
<tr>
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<td>353</td>
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<td>1 690</td>
<td>104 167</td>
</tr>
<tr>
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<td>353</td>
<td>12 037</td>
<td>1 282</td>
<td>121 528</td>
</tr>
<tr>
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<td>1129,60</td>
<td>353</td>
<td>5 709</td>
<td>4 805</td>
<td>115 741</td>
</tr>
<tr>
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<td>353</td>
<td>4 977</td>
<td>1 690</td>
<td>138 889</td>
</tr>
<tr>
<td>7</td>
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<td>353</td>
<td>9 259</td>
<td>1 690</td>
<td>150 463</td>
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<tr>
<td>8</td>
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<td>380</td>
<td>5 709</td>
<td>1 690</td>
<td>344 907</td>
</tr>
</tbody>
</table>

In order to predict the creep rate and the side wall displacements magnitude in the GG-1 shaft to identify the appropriate diameter of the shaft breach, ensuring safe work conditions, it was assumed that the rheological properties of the salt massif in the GG-1 shaft profile is the same as determined at corresponding depth intervals in the SW-4 shaft profile. This simplification can be justified by the short distance between two shafts of around 5 km.

The predicted creep rate of salt walls in the GG-1 shaft at a distance of about 30 m from the roof and bottom of the salt layer after approx. 80 days from the unveiling of the side walls (after stabilizing the creep process) have reached value of about 0.6 mm/day (determined for the designed diameter of the 11.54 m shaft breach).
Based on the calculated results, it is anticipated that at a depth of 1,788.6 m in the GG-1 shaft from the moment of unveiling the side wall to the moment of installing the final lining, the maximum one-side displacement is about 35 cm. Therefore, it was proposed that for safety the designed shaft breach diameter throughout the salt interval would be increased to by 40 cm.

**INNOVATIVE CONSTRUCTION OF THE SHAFT LINING WITHIN THE SALT INTERVAL**

Geological and engineering conditions related to the salt layer occurrence in the area of the GG-1 shaft under construction have imposed the lining design of special construction (Fabich et al., 2017).

Due to the presence of anhydrite interlayers in the salt deposit, the decision was made to secure the salt side walls of the shaft breach with the rock bolts and mesh. The final lining throughout the salt section is designed to be made of five layers. Starting from the shaft breach side, there are a yielding backfilling layer, an external concrete backfilling, an external tubing ring, an internal concrete backfilling and an inner tubing ring, respectively.

The yielding backfilling layer is made of a two-component foam obtained by mixing an aqueous solution of a phenol-formaldehyde resin and an aqueous solution of organic and inorganic acids with the total thickness of about 100 mm. Its function is to initial unload the lining, so that rock salt creeping does not lead to concrete structural damage during the initial phase of its bonding. The second function is to equalize the salt rock pressure on the lining to a circular-symmetrical level of the pressure. The yielding layer should be applied in one cycle.

The main supporting elements of the lining are two layers of tubing rings made of spheroidal cast iron. The external ring is designed of the 9.3 m inner diameter, and the inner one - of 7.5 m diameter. Each of the rings was backfilled with the layer of concrete type C50 / 60 and thickness of 0.42 m for the outer ring and 0.5 m for the inner ring. Both tubing columns will be placed on the base foot through assembly flanges. The final lining will be built from the bottom upwards, after applying the yielding foam layer on the shaft breach face.

Due to technological reasons, in the final lining the outer tubing ring will be installed in advance of the inner tubing ring by 0.6 m.

In order to preserve the nominal thickness of the yielding layer and both concrete backfilling layers, the minimum diameter of the shaft breach should be 11.14 m. However, taking into account the fact of time lag between unveiling the salt walls and installing the final lining, the breach must be made of a diameter ensuring the nominal thicknesses of the lining layers. Based on the numerical simulation, it was determined that the shaft breach should be of 11.94 m diameter.
SUMMARY

In the northern part of the copper ore deposit on the Fore-Sudetic Monocline, the GG-1 shaft is under construction. It will be the deepest shaft in the LGOM Copper Basin so far. A characteristic feature of the rock mass surrounding the GG-1 shaft is the deepest depth of copper-bearing rocks among the currently being operated deposits as well as the rock salt layer with considerable thickness at the depth of more than 1000 m in the Zechstein formations. Rheological properties of salt massif are crucial factors deciding about the shaft stability and life-span of its lining over the salt section. The salt creep causes horizontal convergence of the shaft excavation. Following the shaft breach unveiling, the elastic creep process gradually goes into the steady-state flow. After reaching the dilatation threshold, the third phase of creeping takes place, i.e. rock damage.

The presence of rock salt layer in a shaft profile determines necessity of assessing the salt massif behavior following the shaft breach unveiling, until it will be secured with the final lining, as well as estimating the rock mass pressure. It is crucial to determine the proper diameter of the shaft breach, so that at the time of the final lining installation it has at least a nominal value.

During the lining design process for the GG-1 shaft, the experience gained so far with the lining construction installed through the salt section in the SW-4 shaft, distant from the GG-1 shaft by 5 km, was used. The parameters of the Burgers model were obtained using the Lavenberg-Marquardt method and numerical simulation, which allowed to determine the rate of salt flow in the "Kaźmierzów" deposit at various depths in the shaft. Because the function of the GG-1 shaft excludes turning it off for a period of possible reconstruction of the salt section, which took place in the SW-4 shaft, a new design solution for the housing was proposed. Designed construction of a four-layer casing with the approval is based on a force solution that fully transfers the pressure associated with the rheological flow of the salt rock mass. In the opinion of the authors, it will permanently protect the shaft excavation in a way that does not require its cyclical reconstruction and guarantees safe conditions for its functioning until the depletion of the deposit in the area of "Głogów Głęboki-Przemysłowy".

REFERENCES


INNOVATIVE DEVICE FOR NON-DESTRUCTIVE-TESTING OF CONCRETE

Slawomir Switon, Slawomir Fabich, Joanna Switon, Aldona Waligora
KGHM Cuprum Research and Development Centre Ltd.
Key words: concrete testing, non-destructive methods

ABSTRACT

Ultrasonic testing is one of the NDT methods commonly used in industrial practice. It allows to identify cracks, defects, pores and other discontinuities within the elements. Ultrasonic testing involves applying one transmitting and one receiving head to the surface of the concrete. The transmitter generates short pulses of ultrasonic waves, and the receiver head is responsible for reading them. Then the time required to cross the road between the two heads is analyzed. In accordance with the requirements of EN 12504-4 standard and taking into account the object conditions resulting from one-sided access to the tested structure, the measuring device shall be capable of measuring the ultrasonic wave transition times on two known points. Currently there is no NDT devices that would allow to measure ultrasonic wave on two-points concurrently (one generator, and two receivers). Taking into account the specifics of the examined objects (mining shafts) and the qualitative parameters of the tested concrete (often very corroded concrete with low strength parameters) a measuring set was made, which is characterized by a special grip construction for three ultrasonic heads, high power of the ultrasonic beam introduced into the studied structure, an oscilloscope for simultaneous differential reading. Designed device is patented by Patent Office of the Republic of Poland. This paper describes properties of the device designed and built in KGHM Cuprum Ltd.

INTRODUCTION

Ultrasonic testing belongs to the group of non-destructive tests most commonly used in industrial practice. They allow for identification of cracks, delaminations, gaps, pores and other discontinuities inside the elements. These tests are also used to estimate changes in the microstructure of materials resulting from their long-term use. The technology of materials testing with the use of ultrasonic tests, widely used in the industry, is the A-type scanning technique, which is characterized by the simplicity of measurements in unilaterally available materials. This method is based on the relationship between the velocity of elastic waves with high frequency (50 ÷ 100 kHz) propagating in a solid medium, and the physical and mechanical properties of this medium. After introducing the ultrasonic pulse to the tested medium, its passage time is measured on a specified path. Then the waveform propagation speed is calculated. Using the laboratory-developed relationships between the velocity of the ultrasonic wave and the physico-mechanical properties of the medium in which the wave propagates, the compressive strength of concrete is calculated. Ultrasonic testing is based on one transmitting and one receiving heads. Those heads are attached to the concrete surface. The transmitting head generates short pulses of the ultrasonic wave, and the receiving head is responsible for reading them. Then, the time required to cross the path between both heads is analyzed. In accordance with the requirements of polish standard PN-EN 12504-4 "Concrete testing - Part 4: Determination of ultrasonic wave velocity" and taking into account the object conditions resulting from unilateral access to the tested structure, the measuring device should enable measurement of ultrasonic wave transition times at two known positions. These times form the basis for determining the speed of the ultrasonic wave, and then the strength parameters of the concrete. This method in appropriate standards is called an indirect method of measuring the velocity of the ultrasonic wave in concrete. At present, none of the world producers of non-destructive testing equipment has a device that meets the requirements of the above standard. There are measuring devices available on the market equipped with a single transmitting and receiving head (eg TICO set, UK1401 Surfer tester). In regards to the current standard, measurement of such a set in the conditions of unilateral access to the tested object is possible after fulfilling the criterion of putting a measurement grid on the surface of the tested object, determining the exact location of the heads during measurements. In the conditions of measurements carried out in the mining shaft, due to the shape and geometry of the location of the tested surface and accessibility to it, the application of measuring nets is very onerous, and in many cases simply impossible. In addition, a measurement method based on grid points would significantly extend the research cycle, which would be unacceptable to the client due to the limited accessibility of the test site. The aim was therefore to develop a test methodology and to create a device for the needs of performing non-destructive testing of the concrete casing of shafts in the field of ultrasonic wave velocity measurements.

THEORETICAL FOUNDATIONS OF ULTRASONIC TESTS
Ultrasounds are the mechanical vibrations of the medium particles (elastic waves) at a frequency higher than the upper limit of audibility of the human ear. This limit is estimated at 16,000 or 20,000 Hz. Between the wave parameters, which are: wavelength $\lambda$ [m], its spreading speed $c$ [m/s], vibration period $T$ [s], or vibration frequency $f$ [Hz], the following relationship takes place:

$$\lambda = cT = \frac{c}{f}$$

(1)

Ultrasonic waves differ in the direction of vibrations of the center's particles in relation to the direction of the wave propagation. We distinguish the following types:

- Longitudinal waves
- Transverse waves
- Surface waves (Rayleigh)
- Sub-surface waves
- Plate waves (Lamba)
- Love waves

From the point of view of non-destructive testing of a concrete center one-sidedly accessible by the most important types of waves:

- Longitudinal waves - the particles of the medium vibrate in a straight line in the direction of wave propagation. The compaction and dilution of the medium alternate. These waves propagate in every material medium (L, cL);
- Transverse waves - so-called shear waves cause tangential stresses. The center particles vibrate in a plane perpendicular to the direction of the wave propagation. The propagation of these waves is not accompanied by changes in the density of the medium. These waves propagate only in permanent centers (T, cT);
- Surface waves (Rayleigh) - propagate on the surface of a free solid body, penetrating to a depth of about one wavelength. The particle movement takes place over a slender ellipse (R, cR);

The device constructed within the framework of this project is to be used to study the propagation process of longitudinal waves with one-sided access to the tested material.

THE CURRENT STATE OF THE ART AND MEASUREMENT CAPABILITIES

Apparatus used for ultrasonic measurements in unilaterally available materials consists of two main elements: a concrete and ultrasound transducers. The ultrasonic ultrasound machine is used to assess the quality and uniformity of materials using ultrasound. The principle of operation consists in activating the transmitting head transducer (piezoelectric element) and “listening in” of the signal echo through the receiving head, which is used to identify the velocity of the ultrasound wave in the material being examined. This in turn can be used to determine its strength and homogeneity. Characteristics of concrete scopes:

- Measurement of the transition time of the ultrasonic wave and its speed, distance between the transducers and the depth of the gap perpendicular to the surface.
- Three transmission methods: direct, semi-direct and indirect (surface).

Ultrasound heads are used to transmit and receive ultrasonic waves, the division of which may be made depending on the type of transducer used, the type of transmitted and received ultrasonic waves, the method of excitation of ultrasonic waves or performance of heads. The heads introducing waves perpendicular to the tested surface are called normal heads, and the heads that introduce waves at a given angle to the surface are called angle heads. [4]. Ultrasound heads are used to detect discontinuities (defects) inside materials of a flat, spatial or located at a certain angle to the surface being tested, that is, from which
ultrasonic waves are introduced. They are also used to measure thickness, wave speed or damping. The scope of the use of probes to detect defects in the material under study is limited by the following factors:

- On one side (from the direction of the head application) - dead zone,
- On the other side - so-called the limit distance, which is related to the supply of reinforcement and the ability to detect small defects lying under the bottom of the material under test.

“Old type” of equipment that has been used in Rock Testing Laboratory was build out of following components:

- Ultrasonic wave generator with simple and emergency structure;
- Measuring rod with two ultrasonic heads (with 100 kHz frequency) and a secondary amplifier;

Unfortunately, this set enabled only one base, and cooperated with heads that are not currently produced, which in turn prevented its servicing in the aspect of head replacement.

**TECHNICAL DESCRIPTION OF THE MANUFACTURED APPARATUS**

At present, none of the companies producing equipment for non-destructive ultrasonic testing produces a device that would enable measurement of ultrasonic wave transition time from the transmitting head (N) to the receiving head (O1) and then to the receiving head (O2) with one application of the head assembly ultrasound. Taking into account the specifics of the examined objects (mining shafts) and the quality parameters of the tested concrete (often a severely corroded concrete with low strength parameters) a measurement set was made, which is characterized by:

- a special handle construction for three ultrasonic heads, allowing access to the housing often from a distance of about 1.5 m;
- high power of the ultrasonic beam introduced into the tested structure;
- oscilloscope enabling simultaneous reading of differential time (four channels)
- an appropriate level of signal amplification processed in the receiving heads, at a low level of personal noise

![Figure 1. Schematic diagram of the device made](image)

The ultrasonic wave measurement set made in the unilaterally available material consists of the transceiver part, processing the signal and the part responsible for displaying the measurement results. The schematic diagram of the discussed set is presented in Fig. 1. The transceiver part of the device consists of a set of three measuring heads (two receiving and one transmitting) located on the appropriate boom (holder). Each of the heads is separated by 10 cm from each other, the first of which performs the transmitting function.
and the next two of them receive the transmitting function. Such a system allows for the configuration of the work of the set on two measuring bases - the first of 10 cm and the second 20 cm. A preamplifier was connected to each of the receiving heads, allowing the amplification of the useful wave, thus reducing the amount of noise. The signal processing part consists of a two-channel amplifier of the receiving wave with a gain of 100x and a generator and amplifier of the transmission signal, which due to its power is powered by dry batteries with a voltage of min. 12 V. For this reason, the amplifier was placed in a separate casing (in order to compensate for interference originating from waves induced on the wires). Registering device the measurement results are supplied with a battery-powered four-channel digital oscilloscope, enabling the readout of measurement results from both heads simultaneously on the LCD screen and their recording in the internal memory. Due to the difficult working conditions of the set, all of its components have been made in a high class of dust and water resistance (IP 65). The technical description of individual components of the constructed device is presented below.

TESTING TO CHECK THE OPERATION OF THE EQUIPMENT

After completing the set-up, attempts were made to correct its operation. Testing took place at the Materials Testing Laboratory at KGHM Cuprum. The picture below (Fig. 2) shows the set after connecting and starting.

![Working set during laboratory tests](image)

Fig. 2. Working set during laboratory tests

The first element that was checked was the own time of heads, i.e. the time necessary to overcome the route from the piezoelectric element (which is the main element generating and receiving the wave) to the face of the head. This time is different for different types of heads and depends on their design. The measurement of this time was made by applying two heads with measuring sides to each other (Figure 3). Next, the time necessary to stabilize the wave front was read on the oscilloscope screen (Fig.4). On this basis, it was found that this time in the case of heads used in the constructed set is 16.8 μs.
At the time when the own time of the heads was known, the correct calibration of the device and its readings were checked. For this purpose, a calibration rod from Proceq cat. No. 32540174 was used. According to manufacturer's time of wave transition by this pattern is 26.4 ms. The measurement was made by placing the heads inside the discussed rod (Fig. 5), and then the measurement result was read on the screen of the digital oscilloscope.
The measurement was made by moving the cursor in the oscilloscope to the top of the first positive pulse of the received wave (Figure 6). The noise visible on the oscillogram before the mentioned impulse is caused by the high sensitivity of the amplification system of the set, which in turn is dictated by the purpose of this device, i.e. ultrasonic wave measurement in a concrete center over a distance of 20 cm (further base). If the amplifier was set to a lower value, the noise would also be smaller, but then there would be a serious problem with reading the useful signal. The second cursor in the present case was set on a synchronizing pulse, i.e. a wave-inducing impulse. As can be seen from the discussed waveform, the wave transition time was 43.2 ms, which after deducting the time of its own heads of 16.8 μs gave the result of 26.4 μs, which is exactly as much as in g. manufacturer this time should be. This testifies to the correct configuration of the constructed set, and provides the basis for measurements on concrete samples and the use of the device in the field.

The last of the laboratory tests was the measurement using all three heads, i.e. the measurement on two bases, i.e. the one that was assumed from the very beginning of the creation of the discussed set. For this purpose, a special concrete block was created, additionally dampened on the test side with the use of mineral wool (in order to eliminate the impact of acoustic waves in the air). This measuring block with heads placed in it is shown in the following photograph (Fig. 7).
The result of the study is shown in the oscillogram below (Figure 8). The blue color is marked by the waveform of the ultrasonic wave coming from the head located on the first measuring base (the distance from the transmitting head is 10 cm), while the burgundy color is marked by the wave coming from the head located 20 cm from the transmitting head (second measuring base). The measurement was made by directing the first cursor in the oscilloscope to the top of the first positive pulse of the received wave from the first measurement base, while the second cursor determines the same pulse, but in the wave coming from the head placed on the second measuring base. This approach to the measurement allows to eliminate errors related to the need to take into account the time of the own head, and the inaccuracy of the cursor to the appropriate fragment of the wave (the cursor should always be guided to the wave). The final measurement is the difference of two times (coming from two measuring bases) and is displayed automatically on the oscilloscope screen. In the case of the study, it amounted to 46.4 μs.

Fig. 7. Measurement heads during measurement on a concrete block

Fig. 8. Result of measurement on a concrete block
CONCLUSIONS

The main goal of internal project that was run within KGHM Cuprum Research and Development Center in 2016 was to develop a methodology for non-destructive testing of concrete enclosures in LGOM shafts using the one-sided non destructive method, corresponding to the requirements of PN-EN 12504-4 "Concrete testing - Part 4: Determination of ultrasonic wave velocity". The main result of this project was the development and construction of a device enabling measurement in accordance with the subject standard, i.e. simultaneously on two measuring bases. As part of this work, a measurement set was designed consisting of an ultrasonic wave generator, a receiving wave amplifier, three ultrasonic heads and a digital oscilloscope. All solutions used during the design are characterized by high innovativeness, related to the security of the implemented solutions. It should be noted that the voltage generated at the output of the ultrasound wave generator exceeds 1000 V. In connection with the above, the main goal was, therefore, to choose solutions so that there is no threat to health or life of operating crew, when the equipment's functional requirements are met. The constructed measurement set is characterized by innovativeness resulting from the fact that at the moment none of the companies producing equipment for ultrasonic testing has a device enabling fast and precise simultaneous measurement of ultrasonic wave transition time on two measurement bases. In order to verify the correctness of the applied solutions, a number of tests were carried out, which unequivocally determined that the constructed measurement set enables intuitive measurements to be obtained while at the same time obtaining high readability of waveforms, thanks to which the differential waveform time is easily identifiable. Equipment that was developed by KGHM Cuprum employess is protected by Polish Government Patent Office (P.417413).

LITERATURE

TECHNIC AND RESULTS OF THE MEASUREMENT OF GAS CONTENT AND PERMEABILITY OF COAL SEAMS IN UNDERGROUND MINES

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ABSTRACT

The application of the hydrodynamic impact on the coal seam from horizontal boreholes drilled from the mine entries is considered. A scheme for connecting the main and auxiliary equipment to the mouth of the borehole and a sequence of fracturing operations are given. To monitor the fluctuation of the filtration properties of the wellbore zone before and after the hydrofracturing to intensify methane release, the high-precision geophysical equipment was used. The results of a core sampler application developed by the authors for the collecting of coal samples and measurement of the gas content of coal seams are discussed. The observation of the controlled change in the permeability of a coal seam with regard to its gas content are presented. The developed technic of the coal seams properties measurement have been used on specific coal mines in Kuzbass in order to increase efficiency of coal seams degasification and mine safety conditions.

KEYWORDS

Coal seam, Gas content, Methane, Mining, Desorption, Filtration properties, Injection fall-off test, Hydraulic Fracturing, Degasification, Wellbore zone, Permeability

INTRODUCTION

In the practice of coal deposits development by the underground method, when coal seams have high gas content, a degasification technique is used to reduce the methane release into mine entries and to improve the mining work safety. The volume and rate of methane release from chipped coal depend on the filtration properties of coal seams and on their methane content. To increase the efficiency of degasification and to improve coal seams permeability at coal mines of the Kuznetsk coal basin (Kuzbass) - the main coal-mining region of Russia, the hydraulic fracturing of coal seams from horizontal boreholes drilled from mine entries is used to open existing fissures and to form new technogenic ones, with a view of enhancing filtration properties of coal seams planned for development, as well as to increase their methane production and to reduce residual gas content (Shi & Durucan, 2005; Scott et al., 1996). To determine the reservoir properties of coal seams, the authors developed and applied approaches and special devices for selection of coal samples and undertaking of injection fall-off tests (Tailakov, Utkaev, Makeev & Smyslov, 2003; Tailakov, Utkaev, Zastrelov & Sokolov, 2015).

METHODOLOGY AND EQUIPMENT

Determination of residual gas content of coal seams

For the selection of coal samples directly from mine entries and subsequent determination of gas content in coal seams, several variants of open-type core samplers, differing in shape and type of drill bits have been developed. Structurally, core samplers consist of two sections that are separated after the core is removed from the borehole to ensure the subsequent placement of a coal sample into a sealed canister and the first measurements of methane desorbed volume directly in mine entries under existing mining conditions. In this case, the high-alloyed steel shell coaxial arrangement with a core receiving part is provided to ensure the washing liquid feeding into the space between the shell and the core-receiving part. This allows to exclude a coal sample heating and to reduce gas losses during its transportation from the face to the mouth of a horizontal borehole. The efficiency of designed core samplers’ structures is assessed by a system of indicators.

\[
\begin{align*}
B_k &= \frac{h_k}{H_c} ; \\
K_0 &= \frac{d_k}{D_c} ; \\
k_n &= \frac{d_k}{h} .
\end{align*}
\]

Here \(B_k\) – core removal; \(K_0\) – coefficient of core formation; \(k_n\) – coefficient of core acceptance; \(h_k\) – distance from the core acceptance point to the borehole face; \(H_c\) – borehole length; \(d_k\) – diameter of the drilled core; \(D_c\) – borehole diameter; \(h\) – core acceptance height.

Figure 1 and Table 1 show the results of three types core samplers testing, with regard of time characteristics of drilled coal samples, a coefficient of coal strength, and the averaged coefficients of core formation \(k_0\), core acceptance \(k_n\), and core removal \(B_k\). Comparative analysis has shown that the advantages of the type III core sampler
are the design simplicity and the largest coefficient of core sampling (up to 0.7). Wherein, the drilling time of the sample made 20-27 min with a core length of 300 mm and its diameter of 62 mm.

\[
y = 4.4876x + 16.993 \\
R^2 = 0.98264
\]

\[
y = 4.2756x + 9.5504 \\
R^2 = 0.96701
\]

\[
y = 1.8021x + 9.3869 \\
R^2 = 0.7659
\]

Figure 1 - Dependence of a coal sample drilling time \( (T_d) \) on a coefficient of coal strength \( (f) \), for the coal sample obtained in mining conditions from LFG grade coal using core samplers of types I-III

Table 1 - Results of core samplers testing when drilling samples on coal with \( f = 0.7-2.2 \) a coefficient of strength according to Protodeakonov scale

<table>
<thead>
<tr>
<th>Type of a coal sampler</th>
<th>Coal grade</th>
<th>Core sample drilling time, ( T_c, \text{min} )</th>
<th>( k_o )</th>
<th>( k_n )</th>
<th>( B_k, % )</th>
<th>( B_{k_k}, % )</th>
</tr>
</thead>
<tbody>
<tr>
<td>I</td>
<td>CWB</td>
<td>20-30</td>
<td>0.50</td>
<td>1.52</td>
<td>67</td>
<td>70</td>
</tr>
<tr>
<td></td>
<td>LFG</td>
<td>12-20</td>
<td>0.56</td>
<td>1.45</td>
<td>72</td>
<td>70</td>
</tr>
<tr>
<td></td>
<td>G</td>
<td>10-15</td>
<td>0.7</td>
<td>1.48</td>
<td>70</td>
<td></td>
</tr>
<tr>
<td>II</td>
<td>KWB</td>
<td>18 - 20</td>
<td>0.35</td>
<td>1.46</td>
<td>70</td>
<td></td>
</tr>
<tr>
<td></td>
<td>LFG</td>
<td>26 - 27</td>
<td>0.62</td>
<td>1.58</td>
<td>70</td>
<td>70.6</td>
</tr>
<tr>
<td></td>
<td>G</td>
<td>13 - 14</td>
<td>0.6</td>
<td>1.54</td>
<td>70</td>
<td></td>
</tr>
<tr>
<td>III</td>
<td>CWB</td>
<td>17 - 19</td>
<td>0.45</td>
<td>2.0</td>
<td>75</td>
<td></td>
</tr>
<tr>
<td></td>
<td>LFG</td>
<td>25 - 27</td>
<td>0.65</td>
<td>2.08</td>
<td>80</td>
<td>78.3</td>
</tr>
<tr>
<td></td>
<td>G</td>
<td>10 - 13</td>
<td>0.7</td>
<td>2.14</td>
<td>80</td>
<td></td>
</tr>
</tbody>
</table>

CWB - Coke weakly baking
LFG - Long-flame gassy
G - Gassy

The selection of coal cores is made from boreholes drilled through the coal seam from the sides of mine entries beyond the bearing pressure zone. The distance between the boreholes aimed for core extraction is not more than 300 m (for gently sloping and inclined coal seams), depending on the geological conditions of the coal seam bedding. To increase the reliability of the obtained results three coal samples are taken in each borehole prior and after degasification and measures undertaken to stimulate coal seams gas production.

**Determination of coal seams permeability**

To determine the filtration properties of coal seams in mine conditions by means of undertaking measurements in horizontal boreholes aimed for in-seam degasification, a special chamber was designed and manufactured in a form of a chiseled on one side reinforced rubberized hydraulic hose with a sealed threaded connection on the other side. A pre-programmed electronic self-contained manometer is placed in this chamber and is connected to a borehole water supply system for recording pressure changes caused by hydrodynamic impacts.

The tightness of a wellhead is ensured by setting the casing string down into a predetermined depth and filling the borehole clearance with a sealing material. For casing a column of metal pipes of a 70 mm diameter is used. Upon the completion of work on sealing the wellhead and polymerization of sealing components, the borehole is drilled out and through its uncased part a high pressure pump feeds working fluid into the formation which is filtered through the
system of pores and fissures in a coal seam (Figure 2). The length of the borehole uncased part should be sufficient to maintain the fracturing regime, characterized by a multiple excess of the fluid injection rate above the coal seam natural acceptance capacity (Puchcov, Slastunov, Karkashadze & Kolikov, 2008). After carrying out hydrodynamic impacts on the coal seam the sensor is extracted for processing and interpretation of data, on the basis of which the characteristics of the performed hydraulic fracturing and its effect on the filtration properties of the coal-rock massif are determined.

Figure 2 - Three-dimensional view of coal seam permeability:
1 - coal seam; 2 - casing string; 3 - borehole uncased part

MEASUREMENTS

With the application of developed approach and rigging, the measurements of residual gas content of coal seams in mine entries of Kuzbass coal mines were made. Figure 3 shows graphs of methane desorption from coal samples taken before and after the coal seams degasification.

To assess the effectiveness of coal seams degasification using the developed approach based on direct determination of gas content of the coal, the estimate of degasification efficiency was conducted for four coal mines.

\[
K_d = \frac{\chi - \chi'}{\chi}, \tag{2}
\]

where \( \chi \) – in-situ gas content of a coal seam, (m\(^3\)/t) ; \( \chi' \) – gas content after coal seam degasification (m\(^3\)/t)

The results of degasification efficiency estimate for four considered mines are given in Table 2.
To verify the obtained values of gas content of coal cores selected in mine workings, the relationship between the gas content and the physical and technical characteristics of coal seams has been studied.

Measurements of filtration properties of coal seams were carried out in underground conditions in 132 mm diameter boreholes drilled into a coal seam from a mine entry into 145 m depth and cased with a metal pipe with an internal diameter of 61 mm to a depth of 35 m. The investigations were carried out in three stages: hydraulic fracturing of the coal seam; control of the hydraulic fracturing; evaluation of filtration properties after hydraulic fracturing (“Coalbed Methane Reservoir Engineering”, 1996). To carry out the injection fall-off test, a pre-programmed deep-seated, self-contained electronic, tense type gauge was installed into the path of fluid delivery to the borehole, allowing to measure pressures up to 34.45 MPa with an error of 0.05%. The liquid supply was done by a pump station intended for working fluid injecting into hydraulic systems of coal shearsers and mechanized supports used in mines of any category according to gas and dust content indicators. To prevent hydraulic fracturing, the injection of the fluid was carried out up to 8.5 MPa, pressure which lasted for 70 seconds (Figure 5). The injection period was additionally continuously monitored by the pump manometer reading. At the end of the liquid injection period, the horizontal borehole was overlapped, and the pressure drop was recorded. Then, the electronic manometer was extracted from the borehole and delivered to the surface for the reading of the array of recorded data and their subsequent analysis and interpretation with the aim of coal seam water permeability, capacitive ratio, and skin factor determination.

### Table 2 – Estimate of coal seams degasification efficiency on the basis of direct method

<table>
<thead>
<tr>
<th>Mine</th>
<th>Coefficient of degasification efficiency (KэД)</th>
<th>In-situ gas content, (m³/t dry ash free mass)</th>
<th>Residual gas content, (m³/t dry ash free mass)</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>0.9-0.85</td>
<td>9-14</td>
<td>1.2-2.1</td>
</tr>
<tr>
<td>B</td>
<td>0.01-0.1</td>
<td>9-11</td>
<td>8.7-8.9</td>
</tr>
<tr>
<td>C</td>
<td>0.9-0.6</td>
<td>17-20</td>
<td>2.0-7.6</td>
</tr>
<tr>
<td>D</td>
<td>0.9-8.4</td>
<td>18-20</td>
<td>1.0-3.2</td>
</tr>
</tbody>
</table>

Figure 3 - Desorption of methane from coal samples of LFG grade, selected in mine entries of Kuzbass mines
Figure 6 shows data obtained during the working fluid injecting into boreholes possessing different geological conditions with hydraulic fracturing application (Figure 6, a) and without hydraulic fracturing use (Figure 6, b).

**DISCUSSION**

Based on the performed measurements, it was established that degasification efficiency coefficient for considered mines varies from 10 to 60% (Table 2) depending on the filtration properties of coal seams, as well as under the influence of mining operations, during which a coal rock massif is relieved from the rock pressure. This leads to the increased amount of desorbed methane and stimulates its migration into mine entries or into degasification boreholes, and the unevenness of gas content distribution within a single extraction pillar is caused by the variability of petrographic properties of coal, namely, by the anisotropy of coal matter components, which is characterized by the yield of volatile substances $\text{V}^{\text{daf}}=37-43\%$ and the degree of vitrinite reflection $R_0=0.7-0.9$ and also by the presence of local disturbances in a coal seam structure (Seidle, Jeansonne & Erickson, 1992; Atilla Ozturk, Nasuf & Bilgin, 2004). When constructing isogases, their shape and placement within the coal seam is compared with the type of isolines corresponding to the volatile matter output. Their proximity characterizes the reliability of gas content determination, otherwise additional measurements of gas content of the coal are carried out (Schraad & Trintafyllidis, 1997). Based on the analysis and interpretation of recorded pressure fluctuation in a horizontal borehole during injection fall-off tests, the permeability of the near-wellbore formation zone of a coal seam has been defined, which made $k_x = 19.17$ mD, $k_y = 11.48$ mD, $k_z = 23.51$ mD. Wherein the permeability of the near-wellbore formation zone was reduced, as evidenced by the positive value of the skin factor $S = 6.93$.

After the pressure drop in the borehole, the hydrodynamic impact on the coal seam was performed, followed by a second study to determine the filtration properties. It was found that the permeability in the horizontal direction increased ($k_x = 73.91$ mD, $k_y = 30.20$ mD), and in the vertical direction it decreased ($k_z = 4.16$ mD). In this case, the skin factor $S$ decreased to -0.92. Thus, the measures undertaken for a coal seam stimulation affected the increase in filtration characteristics in the bottom hole zone of the wellbore and boosted the gas production.
Figure 4 - Pressure (P) and temperature (t) fluctuation under the hydrodynamic impact on a coal seam:
   a) - with hydraulic fracturing; b) - without hydraulic fracturing
CONCLUSION

The studies performed on Kuzbass mines confirmed the feasibility and expediency for the use of direct measurements to assess the gas content of coal seams sections planned for development and on those undergoing mining, as well as to control the efficiency of their degasification, using the elaborated approach and equipment for coal sampling and conducting of desorption tests. The application of modern electronic pressure gauges to undertake the proposed and tested in mining conditions coal seams permeability measuring technic allows to provide rational modes of hydraulic impacts on the coal-rock massif with the purpose of increased gas recovery.

REFERENCES

ECONOMIC EVALUATION OF COAL RESOURCE DEVELOPMENTS USING REAL OPTIONS ANALYSIS

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ABSTRACT

Electricity supply in Botswana and the Southern Africa region face an escalating deficit. However, there are untapped coal resources in Botswana that require a comprehensive study and could meet future electricity demands. Augmenting the coal resources could develop a strategized framework that attains electricity security and other coal beneficiation strategies within the country and region. The economy of Botswana can also benefit from increased revenue from the sale of coal. The aim of this study is to appraise a coal project in Botswana awaiting mine development using Real Options analysis. These development projects undergo investment that require management decision to invest either now or in future. Each stage of development entail optimal timing from management to exercise the investment under appraisal in order to proceed to the next level of development. The options embedded within these energy projects are esteemed as having a single option. Finally, an evaluation investment model is constructed using the Real Options analysis on projects with a stochastic behavior under market price and geological uncertainties. Consequently, the economic value of the project options with uncertainty is calculated, to determine the fate of the underlying project.

KEYWORDS
Real Options, Management Flexibility, Uncertainty, Coal Development

INTRODUCTION

When valuing energy resources based on constant predictions of future variables and no considerations of management flexibility, is the expected value of the underlying asset being underestimated? The answer is yes, due to the incapability of traditional valuation methods to capture the value of projects in the current and future deviations of the economic environment. This has given ground for the increase of the Real Options analysis in the mining related activities to utilize its capability to adjust to future risk and provide better decision options to management especially when multiple assets are involved. In Botswana and the Southern African region as a whole, there is an increased electricity supply deficit. However, there are untapped coal resources in Botswana whose options require extensive investment valuation to recognize how best they could meet the electricity demands and possibly for other coal beneficiation strategies.

Real Options introduces a new insight with respect to the role and impact of uncertainty on investment opportunity value that runs counter to conventional thinking (Trigeorgis, 2002). It is a financial engineering application method to investment of real assets, that define an ‘option’ as a derivative transaction in which the option holder of an underlying asset has the right but not the obligation to trade the asset at a fixed price at a certain point of time in future (Adachi, 2008). There are two types of options namely, American and European options. American options are options allowed for exercise before maturity date of the option whilst European options are options only exercised at the expiration date of the option. The dynamic series of uncertainty on the underlying project require investment decisions that are able to adjust to the economic deviations through options assessment by monitoring discount rate and sensitivity analysis associated with the liberalized markets. At this point, Real Options evaluation towards the underlying assets is formulated to cope with uncertainty through the consideration of option value.

Ideally, resource development projects are executed as a single investment option dependent on the upper limit of the development period of the mining right i.e. exploration, pre-feasibility, mine development, operations and mine closure. As such, the underlying assets are evaluated as American options. The risk associated with coal investments are associated with not only the economic environment such as coal prices but also to geological and technical uncertainties such as resources/reserves, calorific value or ash content and operational capacity. It is to note that the complexity of evaluation becomes ambiguous as many variables are assessed, therefore a constraint should be placed during model design.
Therefore, the purpose of this study is to investigate Real Option valuation towards a coal project in Botswana with comparison to the conventional traditional methodologies such as Discounted Cash Flow - Net Present Value (DCF-NPV). The revelations of the methodologies is whilst DCF is incapacitated to capture the real value of the project, the Real Options method achieves this by defining the managerial flexibility to cater for the economic, geological and technical uncertainties rooted in these projects. Through the Real Options Approach, these projects undergo investments that require management judgement on optimal investment timing in order to proceed to the next level of development. Consequently, the economic value of the project options with uncertainty is calculated, to determine the fate of the underlying projects.

**REAL OPTIONS THEORY AND EVALUATION**

The conventional use of the traditional Discounted Cash Flow alone has deemed inappropriate in valuing strategic mineral/energy projects, especially in the turbulent economic market environment. Therefore, a growing focus on an expanded approach of the Net Present Value (NPV) known as the Real Options theory has taken lead in optimizing the value of projects by emphasizing on options derived from management flexibility (Mun, 2006 & Trigeorgis, 1994).

Option pricing models consider numerical analysis of the Binomial Tree for the derivations of option value. The continuous time is a stochastic process where \( V \) is the underlying asset, \( dV \) is the change in value of the underlying asset, \( \mu \) is the expected growth rate of return, \( \sigma \) is the volatility and \( dz = \sigma \sqrt{dt} \) is the increment with time \( t \) of the Geometric Brownian Motion can be shown as,

\[
dV = \mu V dt + \sigma V dz
\]  

The partial differentiation of the Black-Scholes (European option) is represented by,

\[
\frac{df}{dt} + \frac{df}{dV} rV + \frac{1}{2} \frac{d^2f}{dV^2} \sigma^2 V^2 - r = 0
\]

The solution for this differential equation of the Black Scholes formula yields (Kilic, 2005),

\[
C = V_0 N(d_1) - Ke^{-rt} N(d_2)
\]  

\[
d_1 = \frac{\ln \left( \frac{V_0}{K} \right) + (r + \frac{\sigma^2}{2})t}{\sigma \sqrt{t}}
\]

\[
d_2 = \frac{\ln \left( \frac{V_0}{K} \right) + (r - \frac{\sigma^2}{2})t}{\sigma \sqrt{t}} = d_1 - \sigma \sqrt{t}
\]

where equation 2 where \( C \) is the call option, is the value of the call option, \( V_0 \) is the present value of the project (monetary units), the strike price \( K \), risk free interest rate \( r \) (%), time to maturity \( t \) (years), volatility of the project (%), and \( N(d) \) is the cumulative normal distribution function.

In this work, Black Scholes is used only for comparison analysis due to its limitation exercise period only at maturity. The Binomial Tree Model (Cox, et al., 1979) is a discrete model with the level of uncertainty increasing over time by a factor \( \sigma \sqrt{dt} \). This model valuates American options that can be exercised at any discrete period with change in time \( dt \). The volatility of the underlying asset define the random coefficient probability that can take up the ascending \( (u) \) or descending \( (d) \) volatility rate. These coefficients are defined as,
\[ u = e^{\sigma \sqrt{t}} \]
\[ d = 1/u = e^{-\sigma \sqrt{t}} \]
\[ p = \frac{e^{rdt} - d}{u - d} \]  \hspace{1cm} (3)
\[ C = e^{(-rf)dt}[pu + (1 - p)d] \]  \hspace{1cm} (4)

where equation 3 is the risk neutral probability assuming a constant volatility (Kilic, 2005). Furthermore, the risk may not increase over time as the uncertainty does. Equation 4 expresses the future payoff value from the (call) option in a risk neutral world (Mun, 2006).

**CASE STUDY**

The Sese coal asset in Botswana lies within the eastern belt of the country (Figure 1), north of the country’s sole coal mine, Morupule mine. It hosts more than 2.6 billion tonnes of thermal coal that is amenable to open cut mining. The resource is a shallow lying bituminous coal comprising of two main seams averaging 18.2m in thickness. The raw coal on an air-dried basis contains, on average: 15.57 MJ/kg calorific value, 1.89% sulphur content, and 36.81% ash content. Its commencement would involve a three-staged operations involving (African Energy, 2012):

- Stage 1: 1-2 Mtpa of coal production for the first 3-4 years
- Stage 2: A ramped up coal production up to 4-5 Mtpa with feed to independent power producers
- Stage 3 and 4: increased production over 20-30 Mtpa via expanded exports.
The coal market price is the only variable assessed in the evaluation. Cost information for the underlying asset was derived from pre-feasibility studies. The asset is evaluated through the binomial lattice modelling as a call American option, which can delay investment timing towards development of the asset. Investment expenditure is a once off payment at initial stage and immediately operations commence to generate revenue. The market-risk adjusted discount rate for the asset is assumed at 10%, at an estimated average of coal and coal-related industries (Damodaran, 2018).

Traditional Discounted Cash Flow evaluated the Net Present Value as US$1,940.79 at a 10% discount rate. Using the binomial lattice, the implied volatility is 40%. The operations have a mine life of 25 years. The option can be exercised every year at dt=1 up to the expiration date. The parameters of the valuation are shown in Table 1.
Table 1- Option Valuation Parameters

<table>
<thead>
<tr>
<th>Real Option -Call Option</th>
<th>Variable</th>
<th>Units</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Present Value of underlying asset</td>
<td>$V_0$</td>
<td>MUS$</td>
<td>3404</td>
</tr>
<tr>
<td>Expenditure to develop asset/Investment Cost</td>
<td>$K$</td>
<td>MUS$</td>
<td>1463</td>
</tr>
<tr>
<td>Time to Maturity</td>
<td>$T$</td>
<td>years</td>
<td>25</td>
</tr>
<tr>
<td>Risk Free Interest Rate</td>
<td>$r_f$ (%</td>
<td>0.0288</td>
<td></td>
</tr>
<tr>
<td>Volatility of Cash Flow Returns of underlying asset</td>
<td>$\sigma$ (%)</td>
<td>0.40</td>
<td></td>
</tr>
<tr>
<td>Dividend Rate/Convenience Yield</td>
<td>$c$ (%)</td>
<td>0.01</td>
<td></td>
</tr>
<tr>
<td>Length of Period</td>
<td>$dt$</td>
<td>1</td>
<td></td>
</tr>
<tr>
<td>Increasing Multiplicative Factor</td>
<td>$u$</td>
<td>1.49</td>
<td></td>
</tr>
<tr>
<td>Decreasing Multiplicative Factor</td>
<td>$d$</td>
<td>0.67</td>
<td></td>
</tr>
<tr>
<td>Risk-neutral probability</td>
<td>$p$</td>
<td>0.42</td>
<td></td>
</tr>
</tbody>
</table>

Figure 2 illustrates the first lattice in the binomial approach. In a real world, the lattice is created based on the evolution of the underlying asset’s sum of the present values of the future cash flows. The deterministic value of the asset today such as the present value of the underlying asset (MUS$ 3404), goes into the left most lattice node. In the first-time step, this value can increase by a multiplicative factor, $u$, which is based on the volatility, $\sigma$, and length of time step, $dt$, or it can decrease by the inverse of that factor, $d$. The subsequent time steps further increase or decrease to expand the lattice until the maturity date of the option.

Figure 2- Lattice Evolution of the Sese coal deposit
Figure 3 shows the calculation of an American option valuation lattice with the option to wait. The valuation is calculated in 2 phases: phase 1 with terminal nodes and phase 2 with intermediate nodes through a backward induction (calculation) process. For example, terminal node A has a value of MU$75,591,980.76, which is calculated through maximization of the waiting value and the expiration of the option if it is worthless to wait. The value of the node is calculated as MU$75,591,980.76 - MU$1463, where MU$1463 is the investment cost. At intermediate node B, the option value is calculated using equation 4.

Figure 3- Option Value evolution of the Sese coal deposit

Figure 4 shows the payoff profile on an option to wait before mine development for the Sese coal project. The static straight line indicates the project’s option value. The revenues generated from the project assumes no volatility in the cash flows. The project is executed only when the NPV is positive. However, the curved line shows the static payoff line of strategic option values that considered the volatility of market risks and cash flows. Hence, the uncertainty arising from these cash flows may be higher than expected with time before the expiration time of the project.
The project value on 2017 is MUS$3,404 with an option value of MUS$2,530 as shown in Figure 4. The value of waiting for investment is MUS$1,463, which represents the difference of the option value and NPV MUS$1,941, is MUS$589. At MUS$5,280, as the NPV approaches US$0, the value of waiting would be worthless to the investment. However, if project value exceeds the critical value, management’s decision should optimally be to invest immediately. The project value through Black Scholes calculations is MUS$2,768. Like the DCF analysis, it underestimates the true value of the project. Therefore, it shows that through real options, the project is worth more than it suggests.

CONCLUSIONS

The findings of this study validate the crucial importance of uncertainty and management flexibility in mine project valuation through Real Options analysis. Both these factors when realized, positively affect the project value of the underlying asset. As compared to the DCF analysis, the Binomial Tree modeling through Real Options Analysis provides more accurate and calculations and true value of a project. It is therefore crucial for the integration of Real Options in DCF analysis to provide the best implementation practices towards planning, management and decision analysis.

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MONGOLIAN MINERALS SECTOR'S SITUATION AND TRENDS

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ABSTRACT

The mining and minerals sector plays a significant role in the economy and social development of Mongolia. Mongolia's dominant export commodities are mining products and have a major impact on the development of the world's mineral sector. In recent years, the exploration of gold, coal, copper, iron ore, and fluorspar is becoming increasingly active in Mongolia's mineral sector.

The article presents the introduction to strategically important deposits of Mongolia, the mapping of geological exploration, the current state of the mining sector, number of export, exploration and exploitation licenses, oil exploration, export and import amount, human resources and further action to be taken in the industry.

KEYWORD

minerals, exploration, economy, geology, oil

INTRODUCTION

Currently, Mongolia has 6000 deposits of over 80 types of minerals. Northern part of Khentii and Khangai mountain chain is rich in gold deposits, the basin of Orkhon and Selenge river in molybdenum, Eastern Mongolia in fluorspar, and Khuvsgul province in fluorite. Mongolia has abundant coal deposits such as Tavantolgoi, Khar tarvagatai, Achit Nuur, Baganuur, Uvdugkhudag and etc. The deposits located in Nalaikh, Shariin gol, Adunchuluun, Baganuur and Shivee Ovoo have played an important role in providing fuel and energy to the settlements.

There are important deposits of iron ore in Tumurtolgoi, Bayangaol and Tamir Gol. The most of the copper ore deposits are concentrated in the Orkhon, Selenge, Gobi regions, southern Kherlen, Bayankhongor and Khankhukhii areas. The largest is Erdenet copper-molybdenum deposits in Orkhon and Selenge regions. There are several zinc ore deposits such as Tumurtein Ovoo, Salkhit, Tugalgatain nuruu and Modon Ovoo. Gold deposits are also discovered. There are large fluorspar deposits in the Eastern art of Mongolia such as Berkhiin uurkhai, Bor-Undur.

Overview

Mongolian mineral sector can be shortly defined as 3C: copper, coal and China. That means that minerals represent 90% of total exports and 70-75% of which represent copper and coal D.Damba. (2016).

Furthermore, on February 06 2007 a resolution was approved to consider some deposits as strategically important. These are coal deposits of Baganuur, Shivee-Ovoo, Tavantolgoi, Nariin sukhait, uranium deposits of Mardai, Dornod, Gurvanbulag, iron ore deposits of Tumurtei, copper-molybdenum deposits of Oyu tolgoi, Tsagaan Suvraga, Erdenet, phosphorite deposits of Burenkhaan, zinc deposits of Tumurtein Ovoo, silver deposits of Asgat, and gold deposits of Boroo. There are other 39 candidate-deposits to be strategically important. (https://www.forum.mn/p_pdf.php?obj_id=1871)

Geological sector

Nowadays in Mongolia, the geological mapping with scale of 1:200 000 has reached 99%, and the scale of 1:50 000 - 33,7%, and it’s expected to reach 40% in 2020 G.Tamir. (2017).

Figure 1. Geological mapping
Mongolia is undertaking a geological research works with state budget funds to address regional mineralization issues, assess prospects for mineral resources, and use as bases for conservation, and subsoil use decision-making process. (https://mrpam.gov.mn/public/pages/83/monthlyreport2017%20final.pdf)

**Mining sector**

In total, 3369 exploration and mining licenses were issued in 2017, 1745 of which for exploration and 1624 for mining. The percentage of licensed area to total territory is 6.8%, specifically 5.8% is exploration, and 1%-mining license.

Mining sector occupies 21.7% in GDP and the total industrial output of mining sector is 72.5% (2017).

![Figure 2. Distribution map of mining licenses](image)

Mongolian government has launched “Project to provide financial support to Gold sector” jointly with the Bank of Mongolia.

In 2017, 24 new mines and processing plants were opened and received approval from State Commission. Notably, 19 open pit mines, 4 processing plants, 1 semi-coking briquette plant. With the exploitation of these mines and plants, more than 2700 new jobs have been created and project investment reached over 500 billion tugrugs. In 2018, we expect to have more than 290 enterprises with employees of over 30,000. (N.Amartuul. MMHI /Ministry of Mining and Heavy industries/Occupational safety and health issues in the mining sector).

**Artisanal mining**

- In 2017, there were 7085 artisanal miners, 871 unregistered partnerships, 44 NGOs of artisanal miners.
- With the approval of the “Procedure for artisanal mining”, 68 areas of 12 aimag and 28 soums are available for this type of mining.

**Oil sector**

**Exploitation and extraction**

- There are 31 exploration areas with oil perspectives. As of today in 21 areas, 16 contractors have Production Sharing Contracts and approved by the Government.
- In 2017, 7.6 million barrels, i.e 939.3 thousand tons of crude oil was exported to China and contributed 285.6 billion togrogs to the state budget. (Mongolia's geology, mining, oil, heavy industry sector, MRPAM's Operational Report of 2016, Assumptions of 2017-2020, and outcome)
Import of oil products
• Mongolia is 100% dependent from imported oil. In 2017, we have imported 1.49 million tons of oil and oil products.
• From the total amount of imported oil, 89.3% is imported from Russia, 3.3% from China, 2.7% from South Korea and the rest from other countries.

Exploration of oil
• A Production sharing agreement was carried out in the area of 272890 km2 of oil exploration in during 1993-2017
  ✓ 1:200000 scaling gravity survey
  ✓ Magnetic exploration at 77630 km2
  ✓ Passive seismic to 210 point
  ✓ 33494 length ways km
  ✓ 2D vibration research
  ✓ In 6276 square meter area 3D vibration research
  ✓ gradiometer research
  ✓ research were conducted respectively and 1537 holes of exploration were drilled.
• There has been invested USD3.45 Billion into the oil exploration and exploitation operations. As of 2016, 14 holes were drilled in the Galba XI field of exploration and 11 oil wells were drilled and 3 oil flow were identified. The area has 3 oil perspectives and 1517m3 crude oil were extracted during the testing period. (D.Damba. MMHI. Policy and legal reform of the mineral sector)
• 300 * 104 tons of oil resource were estimated and evaluated at 3.6 km2 area in 2015. Exploration in Hooknur XVIII area explored 3 holes and produced 35.2 m3 of oil.

In total 38 licenses of radioactive minerals are valid as of 31 of December 2016, 29 of which for exploration in 955.7 thousand hectares and 8 for exploitation in 81.9 thousand hectares respectively.

Human resources, training
The School of Geology and Mining at the Mongolian University of Science and Technology has prepared over 2600 engineers and technicians since the beginning of the development of high-tech talent in Mining Industry Lh.Orkhon (2016).
1. In addition, there are other state and private universities that train students in mining field, these are School of Technology, filial of Mongolian University of Science and Technology in Erdenet, Mining School of Mongolian National University, Seruuleg University, Gazarchin University, German-Mongolian Institute for Resources and Technology, Mongolian Mining Polytechnic College and Mining Institute. (Lh.Orkhon. MUST. School of Geology and Mining. Technical and human resource planning in the mining sector)

Economic situation
Mongolia is one of the countries where economy is dependent from its mining sector. Statistics say that Mongolia's economy has grown since 2006 when mineral prices rose and when the copper price reached $10,000/ton and coal to $300/ton, Mongolia’s economy grew up to 17.3% and was considered as fastest growing country in the world in 2011. But, when the commodity prices fell, economic growth dropped sharply to 2.3% in 2015 and reached 0.3% at the end of 2017. (G.Tamir. MMHI. Current situation of Mongolia's mining sector, legal environment and further action)

Table 1. Manufacture of main mining products
<table>
<thead>
<tr>
<th>№</th>
<th>Type</th>
<th>Unit</th>
<th>2015</th>
<th>2016</th>
<th>2017</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Copper concentrate</td>
<td>Thousand tons</td>
<td>1,334.7</td>
<td>1,445.1</td>
<td>1,317.1</td>
</tr>
<tr>
<td>2</td>
<td>Coal export</td>
<td>Thousand tons</td>
<td>14,426.3</td>
<td>25,712.6</td>
<td>33,400.1</td>
</tr>
<tr>
<td>3</td>
<td>Iron ore, concentrate</td>
<td>Thousand tons</td>
<td>4,273.6</td>
<td>4,936.2</td>
<td>3,675.0</td>
</tr>
<tr>
<td>4</td>
<td>Gold</td>
<td>Kg</td>
<td>14,556.2</td>
<td>18,435.7</td>
<td>19,846.8</td>
</tr>
<tr>
<td>5</td>
<td>Zinc concentrate (50%)</td>
<td>Thousand tons</td>
<td>89.6</td>
<td>100.2</td>
<td>82.7</td>
</tr>
<tr>
<td>6</td>
<td>Fluor flotation concentrate</td>
<td>Thousand tons</td>
<td>47.3</td>
<td>34.1</td>
<td>55.2</td>
</tr>
<tr>
<td>7</td>
<td>Fluor flotation concentrate</td>
<td>Thousand tons</td>
<td>183.6</td>
<td>167.7</td>
<td>108.9</td>
</tr>
<tr>
<td>8</td>
<td>Molybdenum concentrate</td>
<td>Thousand tons</td>
<td>5.2</td>
<td>5.2</td>
<td>5.6</td>
</tr>
</tbody>
</table>
Table 2. Main resources exported in 2017

<table>
<thead>
<tr>
<th>№</th>
<th>Resource</th>
<th>Unit</th>
<th>2017</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Iron</td>
<td>Thousand tons</td>
<td>6257.8</td>
</tr>
<tr>
<td>2</td>
<td>Copper</td>
<td>Thousand tons</td>
<td>1472.2</td>
</tr>
<tr>
<td>3</td>
<td>Zinc</td>
<td>Thousand tons</td>
<td>118.2</td>
</tr>
<tr>
<td>4</td>
<td>Gold</td>
<td>Kg</td>
<td>14554.5</td>
</tr>
<tr>
<td>5</td>
<td>Coal</td>
<td>Thousand tons</td>
<td>33400</td>
</tr>
</tbody>
</table>

Suggestions

**In terms of Increase of economic benefits of the sector:**

- Activate the inclusion of strategically important and resource-focused deposits into economic circulation,
- Increase the gold exploration
- Increase the utilization and efficiency of mining concentrators and deposits,
- Rise the exploration and export of oil by enhancing the traditional and non-traditional methods,
- Activate the Brown coal project
- Solve the logistics related issues

**In terms of reform of the system:**

- Establish National Geology Department,
- Create minerals’ exchange,
- Reform the structure and organization of Professional Council of Minerals,
- Improve the interrelations of Research Institutes,
- Integrate laboratories into one policy and organization,
- Strengthen foreign relationship.

**REFERENCE**

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MODERN INFORMATION TECHNOLOGIES FOR OPTIMIZATION OF PROCESSES OF DESIGN AND PLANNING OF UNDERGROUND MINING WORKS

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ABSTRACT

We have analyzed the criteria for the selection and use of modern information systems and technologies for optimizing the processing of mining and geological information, modeling, design and planning of underground mines and quarries.

The efficient production planning system is the basis of effective work of the mining enterprise. Traditional planning methods are one of the most time-consuming processes of engineering support and maintenance of mining operations, often accomplished over time and can not ensure the efficiency and effectiveness of the enterprise.

The production planning of mines and mines is based on the work with spatial mining-geological and technological information, the implementation of various geometric constructions and mathematical calculations. The most effective means for performing tasks of this kind are geo-information and mining-geological systems.

The use of such systems in the mining enterprise allows to fully automate the processes of engineering support of mining operations. The use of a single information space and multi-user mode makes it possible to simplify and speed up the processing of information, improve the accuracy of calculations, analyze several different scenarios for the development of mining operations, perform scenario analysis and select the optimal one based on various criteria and constraints, improve the safety of mining operations and others.

The use of software solutions for planning and design provides the miners with tools for transforming geological models of coal seams or ore bodies into spatial location projects of mine workings, optimal schedules for their execution, and also allows finding a rational way to develop the deposit or its site. Modern geo-information systems can be used for known schemes for the development of stratified and steeply dipping bodies by traditional or mechanized methods (chamber, column, chamber-column, floor, sub-floor, block, layers, with or without tabs).

The Figure 1 shows an example of the structural scheme of an integrated mining management system (hereinafter ISM GR) on the basis of a single software (in this case, K-MINE example), which is a closed cycle of automatic processing of mining and geological data and optimization tasks for engineering support for mining operations. The structure of the system includes a central database and a set of specialized software modules.

![Figure 1 - ISU GR structure for underground mines and shafts](image)

The basis for the work of program modules of planning and design are digital models of the deposit and objects of mining technology. This ISU GR contains tools that allow performing modeling of any genetic type deposits...
for different types of minerals (coal seams: horizontal, inclined or steeply dipping, ore bodies and veins: monoclinic, folded structures, disjunctive structures, vein and lenticular structures, etc.).

Modern technologies of three-dimensional modeling provide a full range of works on the creation and maintenance of models of deposits and objects of mining technology. All information (text, table, graphic in various formats) should be structured and systematized in a single information array. On its basis, it is possible to form three-dimensional models of underground excavations of any technical purpose (horizontal, inclined, vertical). There should also be a toolkit that allows you to exchange data with other GIS, GIS and CAD.

The result of the simulation is a three-dimensional geological model of the deposit, combined with a three-dimensional model of the mine field, structures and surface. The resulting integrated model is the basis for the operation of all technological modules: planning, designing, calculating the stressed state of the rock massif and geo-dynamic phenomena, calculating mine ventilation, accounting tasks of the mine and many others.

The mine or quarry, from the point of view of management, is a complex mathematical object, with a dynamic structure, the state of which is constantly changing: work is carried out to drive the workings and redeem the reserves, and the geometric and qualitative indices of the mineral are refined as they are worked out. Monitoring of rock pressure, air pollution and other safety indicators for mining operations are conducted regularly. Therefore, for the models created and used at the enterprise to be up-to-date and could be used for the tasks of production planning and design, they must be promptly refined and updated. To this end, technological complexes of mine surveying and geological support of mining operations should be implemented within the ISU GR.

Current mining and geological models are also the basis for the design and planning of mining operations.

In the software complex for the design of underground mining operations, tasks that automate most of the processes associated with the implementation and optimization of projects for the development of the field as a whole or its sites should be implemented. In general, the complex should be focused on the execution of design works of mining and capital workings, analysis of preparatory-cutting and cleaning works, design of drilling and blasting operations, calculation of mine ventilation and aeration. When developing projects, you can use typical scenarios for various development systems: solid, column, chamber or combined.

The decisive factors that influence the development of the production program for a given period are the condition of the front of the cleaning faces and the provision of the enterprise with prepared and ready-to-extract reserves of minerals.

In order to create programs and plans for mining operations, we use previously prepared design solutions for the development of the field site such as the panel, the block, the lava and the actual geological model of the deposit or site. Based on the standard performance of the equipment (excavating, tunneling, transport), the program is divided into intervals, indicating the amount and timing of performance of certain types of work (basic and auxiliary). At the same time, the ISU GR should address optimization tasks related to determining the order of carrying out preparatory and cleaning workings, minimizing the amount of drilling operations during the extraction cut, reducing the load on the units, maintaining stable work fronts for the fastest development of production capacity.

An important advantage of using ISU GR for mines and mines is the ability to control all information processes and flows, as well as analysis of project and planned solutions. The functionality of the design and planning complexes should include an analysis block that enables mining engineers to monitor the development of mining operations for compliance with projects and development plans.

The ISU GR analysis module should allow to compare mining projects on the basis of normative indicators of technological processes, to estimate the terms, order and parameters of actually performed work, compare them with plans and projects, and also control the timing and order of execution of work.

In order to ensure the safety of mining operations, the ISU GR is integrated with the dispatching systems to monitor the location of equipment and personnel in real time from different manufacturers.

Thus, ISU GR should be a universal integrated system with a single software interface, a single information base and a single mathematical tool for optimizing calculations. The application of such an ISMS GR in mines and mines allows to cover all stages of work with mining and geological information; solves the problems of mining in an integrated manner; optimizes planning and design processes from the point of providing the enterprise with resources for the whole period of its operation; rational development of mineral resources, through increased extraction of minerals; improves the accuracy of stock assessment, calculation of losses and clogging of minerals; increases the safety of mining, reduces the cost of production, which ultimately allows the company to work more efficiently.
DEVELOPMENT OF EFFECTIVE AND SAFE METHODS OF UNDERGROUND MINING OF INCLINED ORE DEPOSITS USING ARTIFICIAL LANDSLIDE WATER-ROCK STREAMS TO DELIVER BROKEN ORE

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SUMMARY

Methods using artificial landslide water-rock streams are offered for development of inclined (15-35°) ore deposits. The idea of this technology is that a powerful water stream is directed at bulk broken ore lying on the inclined soil of the chamber. The interaction of the water stream with the bulk ore results in an underground landslide water-rock stream which moves down by gravity to a ore accepting working. After outflow of water to a water collector, the dewatered ore mass is loaded into haul trucks and delivered to the main ore chute or shaft.

KEY WORDS

Mining method, inclined deposit, shaft landslide water-rock stream, landslide delivery, efficiency.

INTRODUCTION

Some ore deposits, for example, the Zhezkazgan deposit in the Republic of Kazakhstan are represented by a variety of separate sheetlike inclined deposits located at different depths.

The main problem which does not allow to efficiently develop such inclined ore deposits is that in order to ensure access of drill rigs and loaders to sub-levels, and relocate broken ore from sub-levels to the haulage level using large-size haul trucks it is required to develop cascades of spiral entries (ramps) with big (≥ 25 m²) section and big length (600 – 1,000 m) at flanks of blocks (panels).

In order to reduce the volume of pre-production mining works it is suggested to deliver broken ore using artificial landslide water-rock streams.

The idea to use artificial landslide streams to deliver broken ore in inclined chambers during underground mining of ore deposits was used in the Mining Institute named after D.A.Kunayev for the first time in 1977 [1,2].

As underground mining workings are the place of formation of artificial landslides in mining operations, it is recommended in the work [3] to call such landslides as shaft landslides formed under these conditions; as a landslide stream is used as a method of delivery of ore in the course of mining operations, it was suggested to call such method of delivery as landslide delivery. Pilot testing of this technology at mines of Achpolymetall showed high efficiency of delivery of ore using shaft landslide streams.

METHODS OF DEVELOPMENT OF INCLINED ORE DEPOSITS USING LANDSLIDE DELIVERY OF ORE

Method of development of inclined deposits with moderate thickness

The method of development with arrangement of chambers upwards and landslide delivery of ore with free-flowing water-rock streams was suggested to develop inclined ore deposits with moderate thickness (Fig.1).

The essence of this method is as follows. The developed panel (block) upwards the ore deposit is divided into extraction chambers, and pillars are left between chambers. A haulage gate (1) is made in the lower part of the panel. An air gate (2) is made in the upper part of the panel. Inclined drilling rooms (3) are made between the haulage gate (1) and the air gate (2) along the axis of extraction chambers upwards the ore deposit. Ore collection workings (4) are made from the haulage gate (1) in the gate with extraction chambers. A dismountable water collection tank (5) is erected in the air gate (2) by sealing the area of this gate between inclined drilling workings (3), where folding doors which are opened outside are installed. A water collector (6) is erected in the flank of the panel from the haulage gate (1) by making a blind inclined working. A special inclined ventilation and manway working (7) is made in the same flank near the water collector (6) in order to ensure the closed water supply circuit.

A distinctive feature of the proposed method is that development of chambers must be performed in parallel one by one with simultaneous preparation of pillars of hexagonal shape with elongated sides upwards the ore deposit. The ore is broken in a chamber by blasting a fan-shaped drilling pattern drilled from inclined drilling workings (3) at an angle of 45° relative to the axis of these workings. At that, in order to make inter-chamber pillars in places of cutting of a chain pillar, fans of drill holes are drilled to the middle of this chain pillar at an angle of 45° relative to its axis which allows to break ore in slots and discard to the developed chamber area.
Dismountable water-guiding partitions (8) are installed after formation of bulk ore in the chamber soil. Water is fed to the developed chamber area onto the bulk broken ore through the sealed area of the air gate (2) and the inclined drilling working (3). An unbound water-rock stream is formed as a result of interaction of the water stream with bulk ore which under the action of gravitational force along the inclined chamber soil is moved downwards to the ore collection working (4) where the ore mass brought with the water-rock stream stops. Further, the “used” water arrives in the water collector (6) after passing drainage units (9) already clear of ore fines. As the water outflows from the ore mass brought with the stream it is delivered through the haulage gate (1) to the main ore chute. Then, the ore and wash-out of the ore bulk in the chamber is repeated using the above-described scheme. At that, water is fed to the collection tank (5) from the water collector (6) after ore breaking in the chamber using a pump installed in the pump chamber (10) along pipelines laid in the ventilation and manway working (7) and the air gate (2).

Trimming of pillars directed with their long side at the rise will allow to ensure big stability of the pillar-roof system in inclined deposits, as elongated inter-chamber pillars resist the shear more reliably in the area of contact with the soil and the roof, aspecially when these inclined contacts are represented by surfaces with insignificant adhesion with the body of pillars. Besides, delivery of ore by shaft landslide streams excludes the presence of people in the open developed area of chambers.

Taking into account safe conditions of mining operations and high efficiency of the method with arrangement of chambers upwards and delivery of the broken ore through shaft water-rock streams it may become a decent alternative to methods of development using large-size self-propelled drilling and LHD equipment used for extraction of chamber reserves.
The method of development of thin inclined deposits

We suggest to use the method of landslide delivery of ore in combination with scraper delivery and water jet wash-out of ore to develop low-grade and ROM ore in thin inclined ore deposits.

This method is as follows (Fig. 2).

![Diagram of method of development of thin inclined deposits using the landslide delivery in combination with scraper delivery and water jet wash-out of ore](image)

The developed thin inclined deposit is divided into blocks consisting of two adjacent panels (areas) which in their turn are divided into extraction chambers along strike of the ore body along the whole width of adjacent panels. The haulage gate (1) goes in the lower part of the block. The air gate (2) is found in the upper part of the block. The haulage raise working (3) is made between the air gate (2) and the haulage gate (1) on the boundary of adjacent panels with a deepening in the underlying rock of half of the height of its section, and then it is expanded to 5-6 m. At that, passages (4) at the level of soil of the deposit in the form of rectangular rock benches are made in side walls of the haulage raise working (3). Scraper hoists (5) are installed in the passages (4) on sledges with a horizontal platform relative to the incline of the passages (4), and the movement is performed using a special mounting hoist. The main water collector (6) is arranged at the entry to the haulage raise working (3) by means of heading from the air gate at the level of its soil, the chamber towards the haulage inclined working which are met on the boundary of the panel pillar and adjacent pillars, at that a concrete partition is made in the side wall of the air gate which divides it from the chamber, and a shutter door is installed in the place where the chamber meets with the inclined haulage working at the level of its soil.

A pump chamber (7) and a water collector (8) are installed in the lower part of adjacent panels in the flank of the block in side walls of the haulage gate (1).

Cleanup works are commenced after completion of the indicated preparatory works.

Cleanup works are performed simultaneously in both adjacent panels whereas one panel is cleaned up a little bit in advance.

Ore is broken by descending layers with a thickness of 1.5-2.0 m. Drilling of drill holes with small diameter is performed with light truck- or sledge-mounted mobile equipment. Scraping of ore to the haulage raise working (3) is performed after breaking a layer of ore in the chamber. At that, scraper hoists (5) are installed at the entry to the face
line, and scraping of the broken ore is performed after breaking a layer of ore. Further, as the sufficient volume of bulk broken ore is accumulated in the soil of the haulage raise working (3), the collection tank (6) is filled with water and it is released to this haulage raise working. A shaft landslide water-rock stream is formed as a result of interaction of water streams with bulk ore, and it moves by gravity along the inclined soil of the haulage raise working (3) downwards to the haulage gate (1) where after outflow of the "used" water the broken ore is transported to the main ore chute with the help of self-propelled LHD equipment.

Interchamber pillars (ICP) are formed to support the roof as ore reserves are extracted from adjacent panels. The remaining broken ore is washed out between ICP using low-pressure water jets (9) connected with the pump using a flexible water pipeline. Water jets (9) are installed in the upper-lying chamber. Scraping of the broken ore to the haulage raise working (3) is performed after wash-out of the ore to the lower chamber and outflow of the "used" water, and from there the ore is delivered to the haulage gate (1) using the above-mentioned scheme and the shaft landslide water-rock stream, and then using small-size self-propelled LHD equipment to the main ore chute.

Thus, all reserves are mined by alternating the extraction of the main champer reserves using the scraper equipment and landslide delivery of ore with water jet cleaning of the chamber soil.

**EXPERIMENTAL STUDIES OF ARTIFICIAL LANDSLIDE WATER-ROCK STREAMS**

Due to the commencement of development and establishment of a new highly efficient technology of development of inclined ore deposits with the help of landslide delivery of ore there are not so many works on calculation of the optimal water consumption for preparation and movement of shaft landslide water-rock streams possessing the maximum haulage capacity under different conditions of application.

The work [4] contains a semi-empirical formula for determination of the specific water consumption depending on the working incline angle (landslide stream bed) and the swell factor of the broken rock mass.

\[ \frac{m_w}{m_n} = \frac{\mu \cos \alpha - \sin \alpha}{\sin \alpha + \frac{\mu \cos \alpha}{K_p - 1}} (0.02 \alpha + 0.72) \]  

(1)

where, \( \alpha \) – elevation angle (of the stream bed),  
\( \mu \) – dynamic friction factor,  
\( K_p \) – Swell factor for broken ore,  
\( K_p = (0.02 \alpha + 0.72) \) – water consumption reserve factor depending on the shaft landslide stream bed incline angle.

But to increase the reliability of calculations it is required to perform a research to determine the dependence of specific water consumptions on the length of landslide delivery of the rock mass, as this parameter, as well as the stream bed incline angle, significantly affect the efficiency of haulage of the rock mass using the artificial shaft landslide water-rock stream.

**Modelling method**

Depending on the consumption of water participating in formation of a landslide stream, the amount of solid material and its grain size distribution, it is possible to form different types and kinds of landslide streams.

In our case, the solid part of the landslide stream formed in underground conditions is represented by broken rock. On the basis of experimental studies for determination of the grain size distribution of the broken ore [5], the formed landslide stream can be classified as unbound type of landslide streams in terms of the character of dispersed medium, and water-rock - in terms of its type. As for its physical nature, such stream represents a complicated model of a suspended matter-carrying water stream. Consequently, the main regularities connected with haulage of heavy inclusions by water streams are principally applicable to the description of unbound water-rock streams close to them in terms of the character of turbulence of movement [6]. As there is no general theory describing the movement of two-phase streams of different structural types yet, the modelling is performed at this stage of study of landslide streams on the basis of general criteria of similarity adopted in the hydraulics with some restrains and adjustments determined both by the modelled medium and modelling conditions.

The geometric similarity of the natural and model streams is ensured with the equality of lower Froude numbers for them [5], i.e. at \( Fr_{M} = Fr_{H} \) or

\[ \frac{v^2}{L_M} g = \frac{v^2}{L_H} g \]

where \( g \) – gravity acceleration, \( m/s^2 \);  
\( v \) – stream velocity, \( m/s \);  
\( L \) – linear size in meters.
It is noted in the work [5], that the transition zone between the laminar and turbulent circumfluence of particles with a water stream is a zone with particle sizes from 0.05÷1.0 to 2÷2.5 mm. The particles which are finer than the lower size are circumflown in a laminar way, and particles which are bigger than the upper size are circumflown in a turbulent way, being in the automodel area allowing direct geometric similarity in accordance with the adopted scale.

In our case we deal with the broken ore mass with a small amount of fine particles. But the amount of sand particles with a size of 1 to 3 mm of crushed stone particles from 3 to 20 mm equals to 11 % [4]. At the modelling scale of 1:50, based on the geometric similarity of linear sizes of the stream, we need to replace all sand particles with dusty particles with a size of 0.06-0.4 mm. This way we will distort the physical medium of the stream, as water and dusty particles represent a suspension of the semi-colloidal type, the viscosity of which will be several times higher than the water viscosity. That is why, without having an opportunity to take into account the corresponding criteria of similarity, transfer of the solid part from the zone of turbulent circumfluence to the zone of laminar circumfluence with geometric reduction of particles in accordance with the linear scale of the modelling, a part of sandy and crushed stone particles is replaced with coarser crushed stone particles with a size of 20 to 50 mm.

Table 1 below contains the grain size distribution of the modelled dose of ore, where sandy and crushed stone particles with natural sizes of 1 to 20 mm are replace with coarser crushed stone particles with sizes exceeding 20 mm. These particles in the modelled dose of ore are included in the fine particles with a size of 1 mm.

Table 1 - Grain size distribution of the actual broken ore and the dose of ore in the model

<table>
<thead>
<tr>
<th>Pos. No.</th>
<th>Actual linear size of lumps, mm</th>
<th>Linear size of lumps in the model, mm</th>
<th>Lumpiness in the total mass</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>up to 3</td>
<td>up to 0.06</td>
<td>Actual size</td>
</tr>
<tr>
<td>1</td>
<td></td>
<td></td>
<td>kg</td>
</tr>
<tr>
<td>2</td>
<td>from 3 to 5</td>
<td>from 0.06 to 0.1</td>
<td>17,572</td>
</tr>
<tr>
<td>3</td>
<td>from 5 to 10</td>
<td>from 0.1 to 0.2</td>
<td>32,880</td>
</tr>
<tr>
<td>4</td>
<td>from 10 to 25</td>
<td>from 0.2 to 0.4</td>
<td>54,720</td>
</tr>
<tr>
<td>5</td>
<td>from 25 to 50</td>
<td>from 0.4 to 1.0</td>
<td>92,640</td>
</tr>
<tr>
<td>6</td>
<td>from 50 to 100</td>
<td>from 1.0 to 2.0</td>
<td>131,200</td>
</tr>
<tr>
<td>7</td>
<td>from 100 to 200</td>
<td>from 2.0 to 4.0</td>
<td>182,560</td>
</tr>
<tr>
<td>8</td>
<td>from 200 to 350</td>
<td>from 4.0 to 7.0</td>
<td>800,000</td>
</tr>
<tr>
<td>9</td>
<td>from 300 to 500</td>
<td>from 7.0 to 10.0</td>
<td>800,000</td>
</tr>
<tr>
<td>10</td>
<td>from 500 and above</td>
<td>from 10.0 and above</td>
<td>800,000</td>
</tr>
</tbody>
</table>

Artificial shaft landslides used in the mining industry for haulage of solid material (ore) have certain inherent characteristics different from natural landslide streams. They include:

1. Formation of a landslide stream takes place by flowing of water to the bulk solid material (ore), as a rule, having a certain shape, volume and grain size distribution.
2. The flowing water stream has certain (set) hydrodynamic characteristics.
3. Movement of the formed landslide stream takes place within a small (very limited) period of time.

Movement of the landslide takes place along the hard stream bed, at that the landslide itself is accumulated on a relatively small area of the hard stream bed.

Based on the distinctive characteristics of landslide streams used for haulage of broken ore under shaft conditions, an experimental laboratory unit must meet all main peculiarities of a shaft landslide according to Clauses 1, 2, 3.

Fig. 3 shows the experimental laboratory unit for study of shaft landslides. It consists of the following:

1. The main flume.
2. The additional flume.

Working surfaces of flumes are covered with a layer of concrete (Grade 500) mixed with ore sand. The water-cement ratio of the mixture is 0.3÷0.35.

The width of the stream bed (flume) in the experimental unit is equal to 0.2 m, the length of the main stream bed (flume) is equal to 12 m, the length of the additional stream bed (flume) is equal to 2 m. The additional flume may be connected with the main flume practically in any point.

The main flume (1) and the additional flume (2) were joined with each other at a ramp made of wooden boards. The structure of the main (1) and the additional (2) flumes is made of steel sheets with a thickness of 0.8 mm. A mesh was fixed inside the flumes, and then a layer of sand and cement mixture with a thickness of 5-7 mm was put on it.
At that, the internal width of the main flume (1) was adopted to be equal to the external width of the additional flume (2).

The main difference of the experimental unit is that it gives an opportunity to fully identify the conditions of the stoppage of the modelled landslide streams. The identity of conditions of stoppage of landslides allows to obtain a number of characteristics of formation and movement of shaft landslides studied on these models. Observance of the identity of conditions of stoppage of the studied landslide streams is implemented in the additional section of the landslide stream bed. Such opportunity to study the landslide stream using the additional stream is connected with the fact that the additional stream bed does not differ, as a rule, from the main stream where a landslide is formed and moved, by shape and quality.

The difference is that the additional stream is established at an angle which is smaller than the incline angle of the main stream bed connected with it, ensuring this way full stoppage of the landslide stream within the additional stream bed. Its length and incline angle depend on technical conditions of the experiment. The presence of an additional stream bed established at a selected incline angle allows to create identical conditions for stoppage of the landslide stream with different characteristics.

**Establishment of the dependence of specific water consumption on the length of landslide haulage of the rock mass**

Tests were performed in the landslide flume with a big length (Fig.3). At that, the maximum length of the main flume was 12 m, and the additional flume was 2 m. Measurement of the haulage capacity of the modelled landslide water-rock stream was performed at different sections of the main flume. Lengths of these sections were 2, 4, 8 and 12 m.

After stacking a certain dose of ore into the upper part of the main flume, a minimum amount of water was fed to the bulk ore in the flume. The water quantity fed to the bulk ore was gradually increased in each subsequent test to ensure full wash-out of the ore from the surface of the flume and relocation to the additional section of the stream bed. After relocation of the landslide mass to the additional flume, they recorded the time of stoppage (discharge) of the landslide stream and determined the way made by the landslide in the additional section of the stream bed (flume) characterising its haulage capacity.

It was done the following way. Location of the center of the stopped mass was determined approximately by eye. Then, starting from the tail of the stopped landslide mass, the rock was removed in small quantities into a tank to weigh the removed rock each time and until the tank has the half of the initially stacked dose of the modelled rock less the mass left in the armouring near the walls of the main flume. The way made by the landslide in the additional section of the stream bed (flume) at a set average incline angle of the main flume was the baseline. During further tests, the main flume had different lengths of the landslide delivery, and other specific consumptions were recored at which the haulage capacity of modelled landslide streams was approximately the same as the baseline at the average incline angle of the main flume.

The modelling results are given in Table 2.

Recalculation of the specific water consumption for actual (commercial production conditions) were performed taking into account the modelling scale by means of the following ration:
\[ \hat{E}_i = \hat{E}_i - \hat{E}_i^0 \left( \frac{N-1}{N} \right) \]

where, \( N \) - modelling scale, \( N = 50 \);
\( \hat{E}_i \) - actual relative water consumption factor (under commercial production conditions);
\( \hat{E}_i^0 \) - relative water consumption factor for the model;
\( \hat{E}_i^0 \) - factor characterizing the moisture content of the ore used in the models.

The factor value was determined experimentally. The moisture content of the ore for the described modelling conditions at \( N = 50 \) was \( \hat{E}_i^0 = 0.082-0.084 \).

Table 2 - Results of the laboratory experiment for determination of the haulage capacity of the modelled landslide stream depending on the stream bed length (incline angle of the main flume - 24°, length of the additional flume – 2 m, flume width – 17 cm (1 caliber), bulk solid mass – 6.4 kg, incline angle of the additional flume – 9.5°).

<table>
<thead>
<tr>
<th>Pos. No.</th>
<th>Main flume length, ( l_{\text{main}} ) (m)</th>
<th>Water mass fed to the bulk ore in the model, (l)</th>
<th>Rock mass in the armouring, (kg)</th>
<th>The way made by the landslide in the additional flume (in calibers)</th>
<th>Specific water consumption in the model, ( Q_{\text{Mr}} ) (l/kg)</th>
<th>Actual length of the landslide delivery of ore mass, ( L ) (m)</th>
<th>Actual specific water consumption, ( Q_{\text{H}} ) (m³/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.</td>
<td>2</td>
<td>1.54</td>
<td>0.21</td>
<td>-</td>
<td>0.49</td>
<td>100</td>
<td>0.41</td>
</tr>
<tr>
<td>2.</td>
<td>2</td>
<td>1.75</td>
<td>0.15</td>
<td>0.9</td>
<td>-</td>
<td></td>
<td></td>
</tr>
<tr>
<td>3.</td>
<td>2</td>
<td>2.0</td>
<td>-</td>
<td>2.14</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>4.</td>
<td>2</td>
<td>2.62</td>
<td>-</td>
<td>5.95</td>
<td>0.49</td>
<td>100</td>
<td>0.41</td>
</tr>
<tr>
<td>5.</td>
<td>4</td>
<td>2.0</td>
<td>0.34</td>
<td>-0.47</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>6.</td>
<td>4</td>
<td>2.5</td>
<td>0.27</td>
<td>2.77</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>7.</td>
<td>4</td>
<td>2.75</td>
<td>0.23</td>
<td>4.02</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>8.</td>
<td>4</td>
<td>3.0</td>
<td>0.2</td>
<td>6.05</td>
<td>0.55</td>
<td>200</td>
<td>0.47</td>
</tr>
<tr>
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<td>0.75</td>
<td>-0.52</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>10.</td>
<td>8</td>
<td>3.5</td>
<td>0.56</td>
<td>0.64</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>11.</td>
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<td>4.0</td>
<td>0.52</td>
<td>2.73</td>
<td></td>
<td></td>
<td></td>
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<tr>
<td>12.</td>
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<td>4.5</td>
<td>0.51</td>
<td>5.33</td>
<td>0.78</td>
<td>400</td>
<td>0.70</td>
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<tr>
<td>13.</td>
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<td></td>
<td></td>
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<tr>
<td>14.</td>
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<td>1.17</td>
<td>0.88</td>
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<td></td>
<td></td>
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<tr>
<td>15.</td>
<td>12</td>
<td>5.0</td>
<td>1.09</td>
<td>1.76</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>16.</td>
<td>12</td>
<td>5.5</td>
<td>0.98</td>
<td>5.41</td>
<td>0.93</td>
<td>600</td>
<td>0.85</td>
</tr>
</tbody>
</table>

Fig. 4 shows a dependency graph of specific water consumptions and the length of landslide delivery of rock mass.
The graph (Fig. 4) shows that the specific water consumption $Q_H$ is 0.85 m$^3$/t at the length of landslide delivery of the rock mass of 600 m under natural (commercial production) conditions. At the same time, the specific water consumption is 0.41 m$^3$/t at the minimum length of the landslide delivery of rock mass to a distance of 100 m. I.e. if the delivery length is increased by 6 times, the specific water consumption must be increased by 2.07 times. As the dependence of specific water consumptions, $Q_H$ on the landslide delivery length $L$ is expressed by the linear function, then, the water consumption reserve factor depending on the landslide delivery length will be $K_L = 0.00178L + 0.998$

Thus, to specify the calculations for determination of specific water consumptions depending on the length of landslide delivery under certain production conditions it is required to use the indicated reserve factor in order to ensure the optimal haulage capacity of the shaft landslide stream.

The final calculation formula for determination of the optimal specific water consumption will be as follows:

$$m_{\delta} = \frac{\mu \cos \alpha - \sin \alpha \cdot \mu \cos \alpha - \sin \alpha}{\sin \alpha + \mu \cos \alpha - \sin \alpha \cdot K_p - 1} \cdot K_{\alpha} \cdot K_L$$

where, $m_{\delta}$ - mass of water supplied to the bulk ore,
$m_{\delta\alpha}$ - bulk ore mass,
$\alpha$ - stream bed (chamber) incline angle,
$\mu = 0.82÷0.84$ – dynamic friction factor for granites, diorites, sandstones and other hard rocks,
$K_p$ - swell factor of the ore (rock)
$K_{\alpha} = (0.02\alpha + 0.72)$ – water consumption reserve factor depending on the shaft landslide stream bed incline angle.

For example, if the chamber (stream bed) incline is $\alpha = 30^\circ$, the landslide delivery length is $L = 120$ m, the specific water consumption will be 0.268 m$^3$/t.

**CONCLUSION**

1. To develop inclined ore deposits with the medium thickness (5-15 m) it is suggested to use the method of development with arrangement of chambers upwards, creation of inter-chamber pillars and landslide delivery of broken ore allowing to avoid many sub-level workings along strike of the deposit and cascades of spiral ramps with a big section (22-25 m) to have access to sub-levels for big-sized self-propelled LHD equipment which cannot compete with landslide delivery in terms of haulage of broken ore within one block. Thus, for example, only 40-50 seconds will be required to relocate 100 t of broken ore using a shaft landslide along the inclined soil of the chamber with a length of 100 m.

2. To develop thin (1-5 m) inclined ore deposits it is suggested to use the method of combined delivery of broken ore and descending extraction layer by layer, including separation of the block into two wings with arrangement of chambers along strike, arrangement of an inclined main roadway in the middle of the block with closing of the face line in it which equals to the block width, scraping of ore from flanks of the block to the inclined main roadway and delivery of broken ore through it using an artificial landslide stream. Application of this method will allow to improve the intensity of cleanup works by 2 times and more, at that only one raise working must be made between levels for the whole block with a width of 80-100 m.

**REFERENCES**