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Stability Assessment of Mining Excavations: the Impact of Large Depths

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Abstract: Back in the early 1980s, coal deposits occurring at depths of ~700 m below surface were already regarded as large-depth deposits. Meanwhile, today the borderline depth of large-depth mining has extended to >1,000 m. Design, excavation and maintenance of mining roadways at the depth of >1,000 m have, therefore, become crucial issues in a practical perspective in recent years. Hence, it is now extremely important to intensify research studies on the influence of large depths on the behaviour of rock mass and deformation of support in underground excavations. The paper presents the results of the study carried out in five mining excavations at depths ranging from 950 to 1,290 m, where monitoring stations with measurement equipment were built. The analysis of data from laboratory and coal mine tests, as well as in situ monitoring, helped to formulate a set of criteria for stability assessment of underground excavations situated at large depths. The proposed methodology of load and deformation prediction in support systems of the excavations unaffected by exploitation is based on the criteria referring to the depth of excavation and the quality of rock mass. The depth parameter is determined by checking whether the analysed excavation lies below the critical depth, whereas the rock mass quality is determined on the basis of the roof lithology index (W_L) and the crack intensity factor (n)

Keywords: great depth; coal mine roadway stability; underground in situ monitoring.

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1 Introduction

A continuous increase in the depth of hard coal exploitation forces coal mines to excavate mining workings at depths larger than ever. Back in the 1980s, a mining activity below the depth of 700 m was already regarded as a large-depth exploitation, whereas nowadays, this boundary has been stretched to >1,000 m below the surface [1]. This tendency is clear, as currently several mines already conduct exploitation at such depths. Since executing and maintaining underground excavations in such conditions is a practical issue with crucial importance, it is necessary to carry out regular studies at large depths in order to estimate the behaviour of a rock mass and its interaction with support systems [2].

Proper maintenance of underground excavations is inextricably connected with stability control in the assumed time of their functioning [1, 3–5]. The problem has become one of the fundamental issues of geomechanics as well as the subject matter of serious research studies carried out in recent years [6, 7]. The stability control of mining excavations located at large depths tends to be a particularly difficult task due to the concentration of natural hazards and the high values of the primary stresses affecting the loading and deformation of the support of underground excavations [8–13]. The design of the support in the excavation located at large depths should meet the requirements of the excavation functionality, economic standards and the characteristics of the rock mass in terms of susceptibility to significant loads [14–16].

Research studies conducted in mines provide a lot of valuable information on the phenomena occurring in the rock mass at great depths. However, these studies are not always conducted in a comprehensive manner, because the various factors involved are reduced to serious simplifications in the modelling of the rock mass. Often, the measurements tend to focus only on one dimension or aspect, leaving others, which are equally or even more important, aside [17, 18].

The present paper is based on both laboratory research and in situ tests carried out in several mining excavations at depths of 950–1,290 m. The laboratory research focussed on the determination of the geomechanical properties of strata deposited around the monitored excavations, whereas the coal mine tests comprised the following aspects: loading of yielding arch support, loading of roof bolting systems, change of the state of stress in rock mass, roof strata separation, rock fissuring and changes of excavation dimensions. The paper also presents the results of a wide-ranging scope of measurements. A thorough analysis of the obtained results helped to formulate a set of criteria for stability assessment of underground excavations situated at large depths.

2 Research scope

Previous excavation surveys at large depths have not been conducted in a comprehensive manner; therefore, our main goal was to develop a research project covering both the behaviour of the rock mass as well as the load and deformation of the support.

In order to determine the influence of the large depths of mining excavations on their stability, the research study included excavations in varied geological and mining conditions. The research sites were situated in five underground excavations at depths ranging from 950 to 1,290 m. Each mining excavation was equipped with a research station, monitoring the behaviour of rock mass and the stability of the excavation support. The following measurements were carried out in the monitoring stations:

- loading of yielding arch support – using two types of hydraulic dynamometers, namely a footing dynamometer with a shape of a cylinder and a serial roof dynamometer;
- loading of rock bolts – using instrumented rock bolts with strain gauges;
- fissuring of surrounding rocks;
- changes of excavation dimensions – on the basis of height and width measurements between the constant points or on the arches of yielding support;
- dislocation and separation of roof strata – using the extensometer, at more than a dozen measurement points,
- change of stress in rock mass – using the probe equipped with three steel strings fixed in the cross-section at the angle of 60° towards each other; as a result of the cylinder deformation in the plane perpendicular to the axis of the probe and the evoked changes of resonance frequency of the strings, it was possible to measure the changes of maximal and minimal stresses in the rock mass.

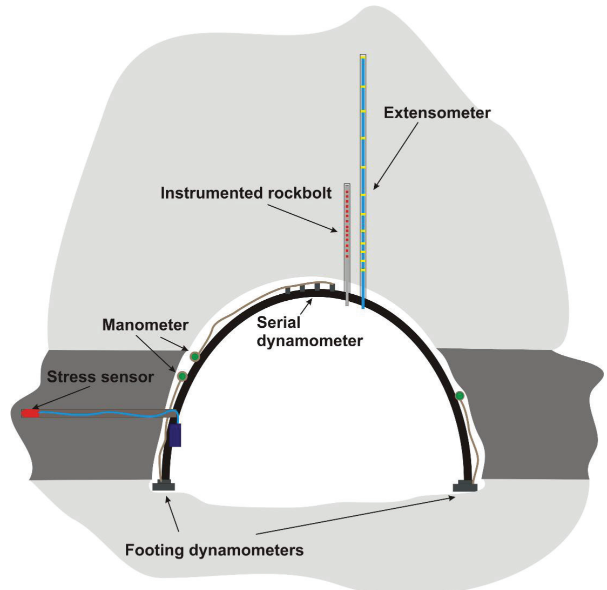


Figure 1: Overall scheme of the measurement station.

An overall scheme of the research station is presented in Figure 1. Due to the varied conditions occurring in mining excavations and the type of measurements, the presented scheme was subject to slight modifications. Each measurement station was constructed in the face of the drilled excavation, and readings from the measuring devices were obtained at about 4-week intervals, after the completion of the drilling also.

In order to determine the geomechanical properties of rock strata, core boreholes were drilled in the areas of all measurement stations, and samples were extracted. In addition, penetrometer tests were carried out in the boreholes in order to determine the strength parameters of the analysed strata. The boreholes were also used for the tests of the properties (orientation, spacing and length) of joints in the fracture zone using an endoscope.

3 Method and results

The footing dynamometers were installed below the yielding arch support footings, and their task was to measure the loading of side arches, whereas the roof dynamometer was installed in the roof arch in the highest point of the excavation in order to monitor the loading from the roof strata. Figure 2 presents the distribution of forces acting on the frames of the yielding arch support in the raise gallery Z-VII.

The largest loading registered from the moment of dynamometer installation took place during the first

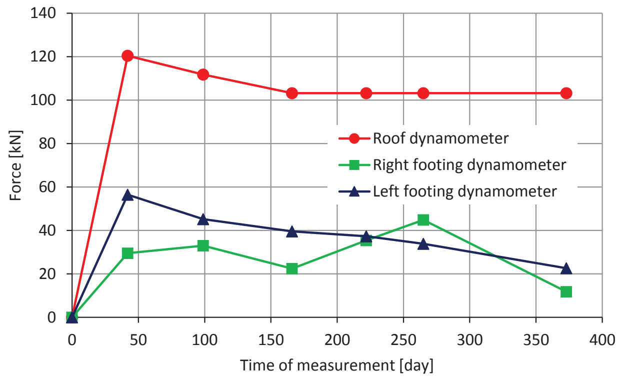


Figure 2: Distribution of forces in dynamometers – raise gallery Z-VII.

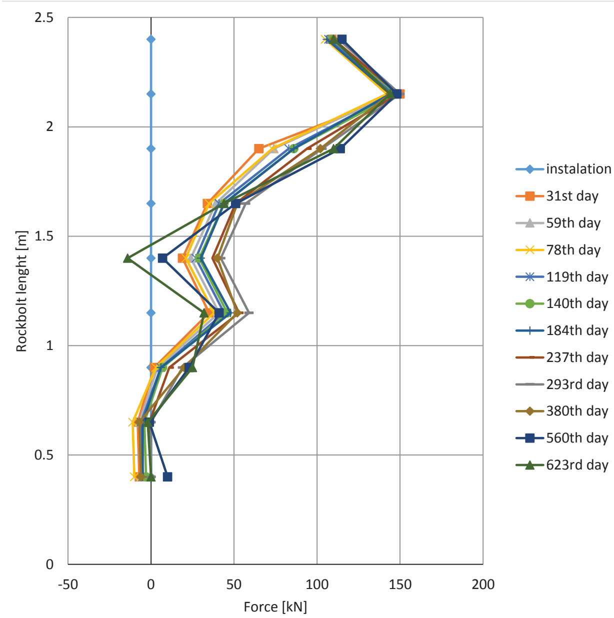


Figure 3: Force values in the instrumented rock bolt – airway cross-cut W-1.

control reading for the roof dynamometer and the left footing dynamometer. The former transferred a force exceeding 120 kN, whereas the latter registered the force of 56 kN. In the case of the roof dynamometer, the loading of the roof arch was 103.2 kN and remained unchanged from the 166th day, whereas in the case of the right footing dynamometer, the force acting on the side arch first increased to the value of 45 kN (265th day after its installation) and then decreased to the value of 11 kN. It can be noticed that the highest load values were noted on the roof dynamometer, and much smaller values were noted on the right and left footing dynamometers. On this basis, it is concluded that the resultant load from the rock mass to the yielding arch support had a vertical direction.

The instrumented rock bolt, with strain gauges fixed at the spacing of 0.25 m, had a total length of 2.5 m and was installed in the roof of every mining excavation. Measurements of the resistance value of each strain gauge were performed using a specialised meter. The first measurement was made before installing the instrumented rock bolt as a base measurement against which resistance changes were calculated during subsequent control measurements. The numerical values recorded by the meter were converted using the Exbolt program into axial forces. The changes of forces in the instrumented rock bolt installed in the airway cross-cut W-1 are presented in Figure 3.

The loading tests of the roof bolting support in the excavation situated at a depth of ~950 m show that the value of axial forces changed along with the rock bolt length. The maximum increase of force occurred at the height of 2.15 m after the first control measurement, when the largest recorded tensile force was 150 kN. Insignificant values of compressive forces at the level of −8 kN were recorded at the initial part of the rock bolt. Every reading that followed later indicated a slight increase of axial forces in particular points of the rock bolt. Only during the last measurement, taken on the 623rd day after the installation, there was a slight decrease in the values of force in the rock bolt. The graph of axial forces acting on the instrumented rock bolt represents the fracture load pattern of the rock bolt. The initial length of the bolt can become subject to a small value of compressive forces; however, along its length, these forces act as tensile forces.

The tests carried out with an endoscope included the recording of discontinuities in the form of cracks (fractures) and fissures (open cracks) occurring in the rock mass in the vicinity of mining excavations. An endoscopic camera, used in the tests, was introduced into the boreholes with diameters of 95 mm, which were also used in the penetrometer tests. Figure 4 presents the scheme of the observed fracture and separation in the roof of C-1 cross-cut.

The endoscope tests showed that a large fracture zone, reaching as much as 6.3 m, appeared in the roof of the analysed excavation. A particularly intensive fracture network, comprising fractures and fissures occurring at distances ranging from several to more than a dozen centimetres, reached the height of nearly 4.5 m. On the whole, as much as 36 fractures and fissures, of a maximum opening reaching ~65 mm, were recorded in this section. In addition, rock rubble was also observed in the initial 20 cm of the borehole.

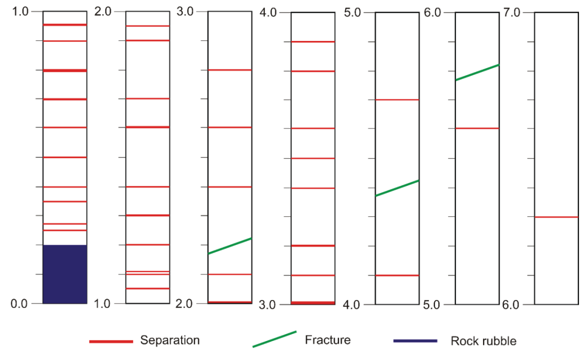
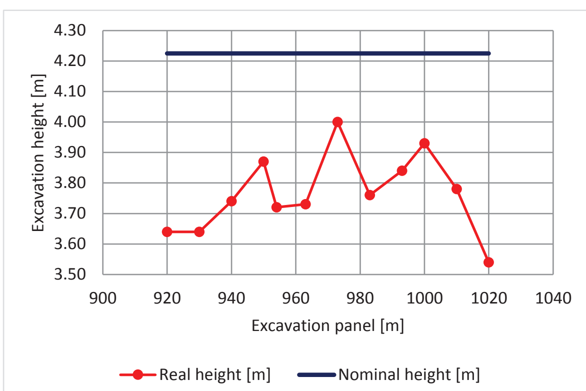


Figure 4: Results of endoscope test in the roof of C-1 cross-cut.

a)



b)

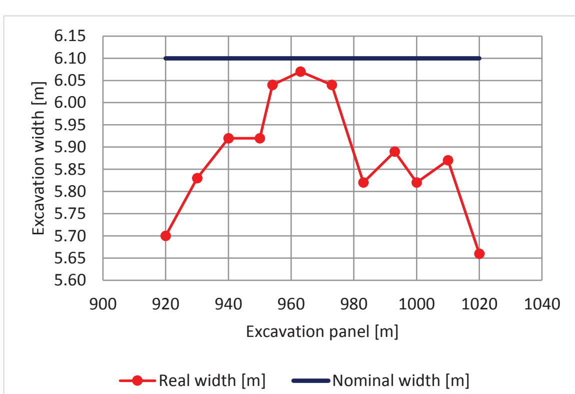


Figure 5: Change of height (A) and width (B) of yielding support arches in relation to nominal dimensions in the panel of airway cross-cut W-1.

The deformations of the support of airway cross-cut W-1, situated at the depth of 950 m, occurred in both vertical and horizontal directions, despite the fact that the yielding arch support was additionally reinforced with the joists anchored in the roof. The average change of height of the arches of the yielding support in the 100-m section of the excavation in the vicinity of the measurement station was ~0.46 m, whereas the average change of width was 0.20 m. The changes of dimensions of the support in the airway cross-cut W-1 in relation to the nominal dimensions of the support frames, measured on the 623rd day after measurement station installation, are presented in Figure 5.

The measurements carried out using an extensometric probe helped to determine the values of dislocation of particular anchors in relation to the highest one in the roof. Positive values of anchor displacement indicate their movement towards the centre of the excavation, whereas negative values indicate the movement in an opposite direction. Changes of distances between particular measurement anchors were identified as separation, which – assuming positive values – mean the stretching of a gap opening, whereas negative values indicate the compaction of strata. The measurements carried out using the extensometric probe aimed at assessing rock mass behaviour in the direct neighbourhood of excavations situated at large depths.

Results of the measurements taken in the S bypass of the IIz shaft indicate that the displacement of the measuring anchors was negligible and reached the value of only 1.5 mm at the maximum (Fig. 6). It may be observed that practically all measurement points in the roof strata dislocate with regular intensiveness. The only exception in this respect is the anchor installed at the depth of 2.48 m, which shows that in this very place, the rocks are subject to slight separation (Fig. 7). It may be, therefore, argued that in this case, such a behaviour stems from the occurrence of a fairly homogeneous and non-stratified roof made of conglomerate and sandstone with varied granulation.

In addition, a regular uninterrupted monitoring of stress change in the rock mass was made as part of the coal mine research. Figure 8 presents the results from the Eastern guideline, including two curves representing the changes of the maximal (p) and minimal (q) stresses, as well as the change of angle (θ) indicating the tilt of the maximal stress (p) from the vertical (negative value means anticlockwise deviation from the vertical). What is characteristic for both curves is the fact that they fail to have a regular course, though there is a sort of continuous oscillation around the values in a given period of time. Observably, the maximum stress (p) increased gradually in the initial period, reaching the highest value of 0.57 MPa, and then it stabilised to reach the lowest value of the change at the level of ~0.3 MPa after 270 days. In the case of the change of the minimum stress (q), after reaching its extreme value at the level of

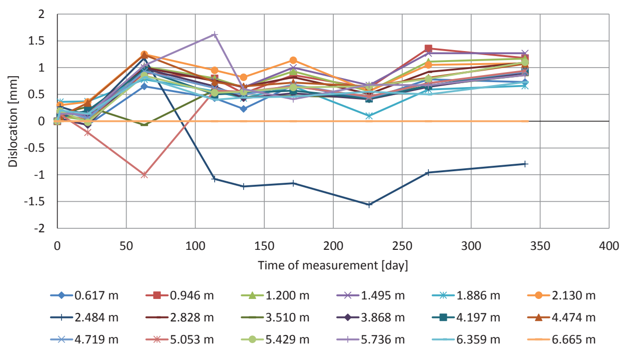


Figure 6: Dislocation of particular anchors in the roof of S bypass of IIz shaft.

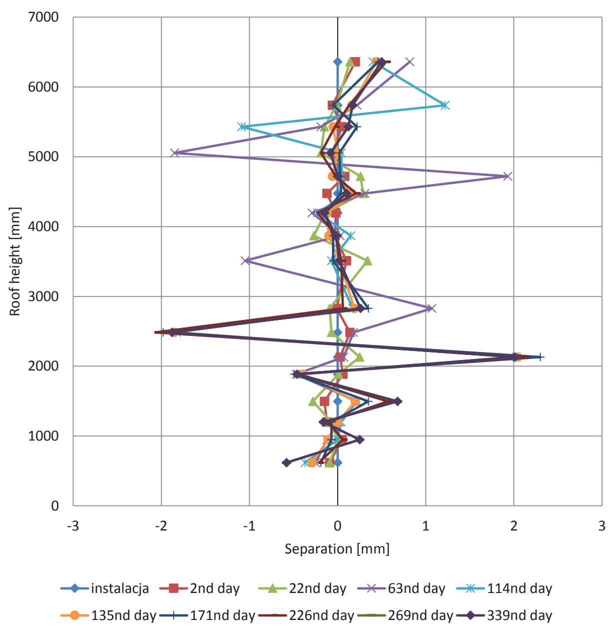


Figure 7: Roof separation of S bypass of IIz shaft.

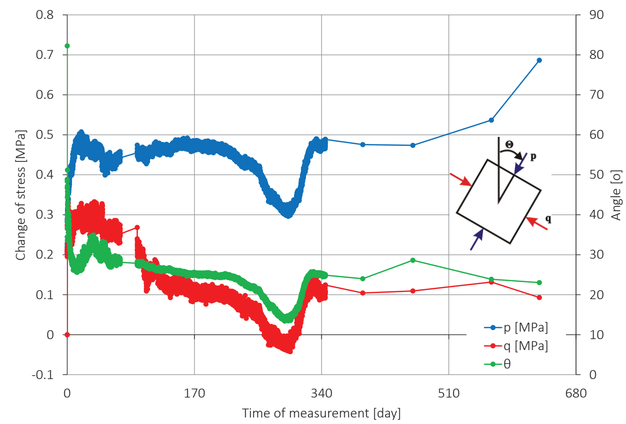


Figure 8: Change of values and orientation of principal stresses in Eastern guideline.

~0.35 MPa, it decreased gradually to the negative values of approximately −0.05 MPa. In the case of the maximum stress (p), an increase in the value to >0.8 MPa appeared during the last control test. The minimum stress (q) increased to ~0.1 MPa and oscillated around this value during the following period of monitoring. Basically, the tilt angle (θ) of the maximum stress (p) from the vertical decreased during the entire monitoring period. At the very moment when the lowest values of the minimum stress (q) and the maximum stress (p) were recorded, the angle also reached the minimal value of ~15°. Hence, it may be argued that in this period, the maximal main stresses were only insignificantly deviated from the vertical.

Table 1 compares and contrasts the results of the research study, presenting the following values: maximal force registered in one of the three dynamometers in the entire measurement period, maximal tensile force acting on the instrumented rock bolt, maximal dislocation of the extensometric anchors, the largest value of stresses, as well as changes in the height and width of the monitored excavation. In addition, the table presents the values of the uniaxial compression strength for the roof strata (determined using the borehole penetrometer and the laboratory samples), the range of the fracture zone and the volume of fractures in the excavation roof (determined using endoscopic research).

4 Criteria for stability assessment of underground excavations

The analysis of all the results of laboratory and coal mine tests proved that the depth and quality of rock mass are the key factors affecting the loading and deformation of the support of excavations not influenced by mining activities. Nevertheless, these factors do not equally affect the changes occurring in the excavation and its surroundings. Hence, three criteria are proposed for the prediction of loading and deformation of the support scheme of excavations situated at large depths (cf. Table 2).

In the first criterion, it should be checked whether the depth of the excavation site exceeds the critical depth. For the determination of the critical depth, the application of uniaxial compression strength of roof strata, obtained from the test using the borehole penetrometer in coal mine conditions, is recommended. It was assumed that the stress concentration coefficient is $K = 2$, since it conforms

Table 1: Research results in monitored excavations.

Parameters		Cross-cut C-1	Eastern guideline	Airway cross-cut W-1	Raise gallery Z-VII	Bypass S of IIz shaft
Depth, m		1,290	1,050	950	950	1,075
R_c *roof*, MPa	Penetrometer	38.1	40.5	27.8	23.3	51.0
	Laboratory	67.9	70.2	52.9	56.2	50.7
RQD, %		6.6	43.7	14.6	13.7	85.4
Number of strata in roof		13	12	9	3	6
Number of fractures in roof		36	27	7	10	2
Range of crack zone, m		6.5	8	0.7	4.7	1
Maximal force on dynamometers, kN		187	222	299	120	0
Maximal tensile force in instrumented rock bolt, kN		277	281	150	255	229
Maximal dislocation of extensometer anchors, mm		–	60	72	–	1.6
Change of principal stresses, MPa		–	0.8	0.61	1.44	0.44
Change in height and width of yielding support arches in relation to nominal dimensions, m	h	–	0.01	0.46	0.27	–
	w	0.84	0.13	0.22	0.06	–

to the assumption that the rock mass preserves its elastic properties if the primary pressure does not exceed 50% of the uniaxial compression strength of the rock mass.

The second criterion is determined using the roof lithology index W_L as a sum of the thicknesses of particular strata, multiplied by a proper reduction coefficient (depending upon the lithology), for the entire length of the analysed core extracted from the excavation roof. The weighted arithmetic mean obtained in this way, where strata thickness is the weight, indicates the dominating rock type in the analysed roof. The values of the reduction coefficient r for particular rock types result from their strength proportion and strata jointing, directly affecting the roof strength.

The third criterion is determined using the fracture intensiveness coefficient n, which determines the density of the weakened cohesion volume in the rock mass. The range and intensiveness of the fracture zone are determined using endoscope and penetrometer tests, whereas the analysis of roof bedding is made during the assessment of the borehole core, obtained from drilling the test boreholes.

Depending on the values of the obtained results, four variants of loading and deformation of a given type of support scheme in the excavation situated at large depths are possible. Table 3 presents the possible prediction variants in relation to the obtained values.

Variant I means that the mining excavation is situated in the rocks of high strength and the roof does not have a densely developed fracture network. The optimal support scheme for such conditions is an individual yielding arch support without any additional reinforcement.

Table 2: Parameters used in the proposed methodology [19],

Parameters	Comments
$H_{kr} = \frac{R_{c\,pen}}{2 * \gamma}$ $\frac{H}{H_{kr}} \leq 1$ $W_L = \frac{\sum h_i * r_i}{\sum h_i}$ $n = \frac{l_s}{l_w}$ $W_L - n > 0$	H – depth of excavation site, m; H_{kr} – critical depth, m; $R_{c\,pen}$ – penetrometric compressive strength, MPa; γ – specific weight of overlying strata. 0.027, MN/m³; n – fracture intensiveness coefficient (–); l_s – number of fractures in roof (–); l_w – number of strata in roof (–); W_L – roof lithology index(–); h_i – thickness of the ith rock layer, m; r_i – reduction coefficient for the ith layer, related to lithology: sandstone – 1.0; siltstone – 0.77; claystone – 0.66; coal – 0.72.

In Variant II, the excavation is driven at a large depth, exceeding the critical value but still situated in strong and unfractured rock mass. The most suitable support scheme for such conditions is a yielding arch support with anchor rods or cable bolts.

Variant III means that the excavation was driven in the rock mass with low strength and intensive network of fracture, at the depth not exceeding the critical depth. In such conditions, the most efficient and effective solution is the application of a support scheme consisting of (i) arches of a yielding support (made of steel with upgraded

Table 3: Variants of possible support loading and deformation of mining excavations [19].

Parameters	Variant I	II	III	IV
$\frac{H}{H_{kr}}$	<1	≥1	<1	≥1
$W_L - n$	>0	>0	≤0	≤0
Final result	Possible insignificant loading and deformation of yielding arch support	Possible significant loading of roof bolting support	Possible significant loading and deformation of yielding arch support	Possible significant loading and deformation of yielding arch support and loading of roof bolting support

strength parameters) and (ii) anchors or anchored joists fixed between the arches.

Variant IV assumes a significant loading and deformation of the yielding arch support, as well as a large loading of roof bolting. Such a situation is possible if the excavation is situated below the critical depth in the rock mass consisting of thin layers with low strength parameters and intensive network of fracture. Stability maintenance of such a mining excavation may be fairly difficult. However, in order to preserve its dimensions, it is recommended to apply a yielding arch support made of steel with upgraded strength parameters, along with anchor rods or cable bolts with upgraded load-bearing capacity.

5 Conclusions

The following conclusions may be drawn from the presented study:

Depth is not a factor obviously affecting the loading of a yielding arch support. The highest value of the force registered by the dynamometer was observed not in the deepest excavation but in the airway cross-cut W-1 at the depth of 950 m (299 kN). However, in the case of the loading of the roof-bolting support, it was observed that the increase in axial force, acting on the anchor rod, becomes larger along with its length, i.e. along with the increase of the distance from the excavation contour. In the same excavation (airway cross-cut W-1), the highest tensile force in the instrumented rock bolt was recorded, amounting to 281 kN.

Endoscopic tests and measurements using the extensometric probe were complementary to each other. The results obtained from both types of tests prove that a dense network of fractures may develop in the roof with a significant bedding ratio. The contact surface between two strata creates a plane with a weakened cohesion, which in the case of thin layers may add to the intensification of the fracture network in the excavation roof. A large number of thin roof strata with low strength parameters may cause additional loading of the support, related to the detachment of particular strata from the virgin rock. This is particularly evident in the eastern guideline where 12 strata and 27 fractures were found in the roof, and the maximum load on the dynamometers was 222 kN.

Deformations of the yielding arch support depend upon the depth of excavation site and the load-bearing capacity of the support scheme. In fact, the application of support with additional reinforcements may reduce the excavation convergence, but the phenomenon may not be completely eliminated. In the C-1 cross-cut, situated at the depth of 1,290 m, where an upgraded load-bearing support scheme with additional reinforcements was used, the change in width amounted to 0.84 m. Change in stress was periodical, which was observed in all monitored excavations.

The proposed method for predicting the loading and deformation of excavation support utilises the relationships pertaining to the depth of excavation in relation to the critical depth and rock mass quality, determined from the difference between the roof lithology index W_L and the fracture intensiveness coefficient n. On the basis of the values of particular criteria, four variants were distinguished, along with presentation of the proposals for the range of measures necessary to be taken in order to reinforce the support schemes of mining excavations.

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