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An analysis of rock mass characteristics which influence the choice of support

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Abstract. There are currently three common methods for selecting excavation supports in Polish hard coal mines. While many factors are considered when choosing appropriate support, these do not include layering or cracking in the excavation ceiling. Although global classifications of rock mass are rarely used in hard coal mines, they are utilised much more frequently during the construction of underground structures such as tunnels. Mining classifications of rock mass have been developed (e.g., in Germany) and they rely on a number of factors but are often related to local mining and geological conditions.
This paper discusses the selected findings of a study carried out on seven excavation sites with diverse mining and geological characteristics. Based on the collected data, two indicators were developed to describe rock mass quality. The first indicator is referred to as the roof lithology index WL and describes the quality of the excavation roof in terms of its layering and lithology. The second indicator is the crack intensity factor n and represents the amount of cracks in an excavation's roof. The correctness of the developed indicators was supported by reliable data from the excavation in which the designed support did not fulfill its task but was changed at a later stage, after calculating the proposed indicators.

Keywords: support; stability; excavation; rock lithology; crack

1. Introduction

Usually, as the depth of mining operations increases, so does the likelihood of natural risks relating to every underground extraction. In addition, there is more stress, which causes deformations in the excavation supports (Hoek *et al.* 1995). Consequently, tunnelling becomes more and more difficult and the applied support structures do not always serve their purpose, causing excavations to lose their stability and usefulness (Chudek and Duży 2005).

The choice of support and the sustained stability of excavations rely on factors that are both geological and related to the current and previous mining operations. In excavations, which are often located more than 1,000 m below the ground, the importance of the parameters influencing the stability of such excavations may differ significantly from those which determine the functionality of excavations located closer to the surface (Zhang *et al.* 2013). It is commonly agreed that depth is a parameter that negatively affects the sustained stability of excavations due to changing rock properties (Singh *et al.* 1992). In many cases, it might be much easier to maintain excavation dimensions in hard rock at greater depths than in soft rock at shallower depths (Lubosik *et al.* 2017, Majcherczyk and Bednarek 2017, Verman *et al.* 1997). The rock mass behaviour around excavations at significant depths is, therefore, a complex issue.

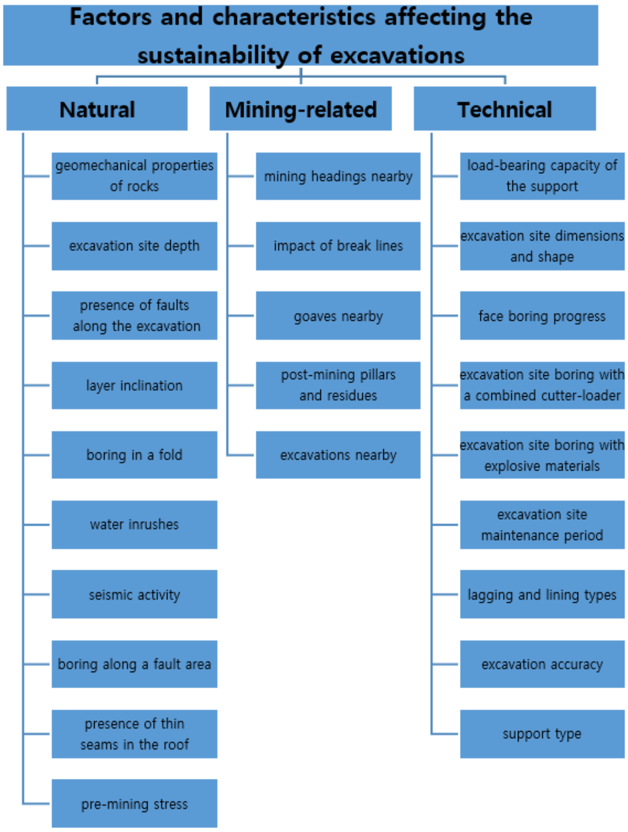

Fig. 1 Factors and characteristics influencing the choice of support and sustained excavation stability

The choice of support and sustained excavation stability are influenced by three types of factors: natural, mining-related, and technical (Fig. 1). In addition, each of these

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Table 1 Geomechanical properties used as the basis of excavation support design principles

Factor	Chudek *et al.* (2000)	Drzęźla *et al.* (2000)	Rułka *et al.* (2001)
Geomechanical characteristics of rocks	1. Uniaxial compressive strength 2. Compactness index 3. Slakeability (reacting with water) 4. Rock massif divisibility 5. Young's modulus	1. Uniaxial compressive strength 2. Compactness index 3. Slakeability 4. Core sample divisibility 5.Young's modulus	1. Uniaxial compressive strength 2. Compactness index 3. Slakeability 4. Core sample divisibility

types include several parameters which, to a smaller or larger degree, affect excavation stability and sustainability.

Previous analyses (Majcherczyk *et al.* 2012) have shown that the most important group of factors influencing sustained excavation stability are the natural factors (43%), followed by mining-related (32%) and then technical factors (25%).

In the group of natural factors, the important parameters that determine the choice of support and sustained excavation stability are driving along a fault area, excavation site depth, pre-mining stress, and the geomechanical properties of the surrounding rocks. With regards to mining-related factors, the most important is the proximity of other mining headings nearby, whilst among the technical factors, it is the load-bearing capacity of the support.

For the purposes of selecting excavation supports, Polish hard coal mining relies on the design principles developed by teams led by Professor Chudek *et al.* (2000), Professor Drzęźla *et al.* (2000), and Professor Rułka *et al.* (2001). The choice of support is based on selected geomechanical properties of the rocks, as listed in Table 1.

The above-mentioned principles take into consideration rock strength characteristics and, in particular, the uniaxial compressive strength of roof rocks, but do not account for roof layering or crack rate. While rock massif divisibility and core sample divisibility are taken into consideration when choosing the type of support, they fail to fully account for the discontinuity of rocks in the rock mass surrounding the excavation site. Interesting research results were presented by Khorzoughi *et al.* (2018) where rock strength and fragmentation can also be determined during borehole drilling by looking at the drilling parameters, such as penetration rate, torque, etc.

The literature offers many rock mass classifications, from the first system developed by Terzaghi in 1946 to the *RMR* (Rock Mass Rating) (Bieniawski 1973), the *Q* index (Rock Mass Quality) (Barton 1974), and the GSI (Geological Strength Index) (Hoek and Brown 1998). However, none of these are commonly used in hard coal mining. Rock mass assessment methods which take into account local conditions could well be used in coal mines (Babets *et al.* 2017).

The RMR, Q and GSI classifications are the basic classifications of rock mass used in the design of tunnel supports (Rehman *et al.* 2018). The Surface Rock Classification (SRC) is also used which, like most of this type of classification, consists of the pointwise attribution of values to individual properties of the rock mass (Gonzalez de Vallejo 2003). The listed classifications have been used to design the support in tunnels with great successes for many years and have recently been supported by numerical modeling (Aksoy *et al.* 2016). Sometimes there is a case, however, to apply rock mass classification to an unconsolidated sedimentary rock (Ko and Jeong 2017). Therefore, many factors sometimes lead to some modifications of the known classifications by taking into account local conditions (Daraei and Zare 2019, Mohammadi and Hossaini 2017, Panthee 2016, Sun *et al.* 2014). In recent times, even the PSU-RQ (an android application) for smartphones has been created in which you can assess the rock mass in a stepwise manner and choose the right support, saving all of the data using the RMR and Q (Pantaweesak *et al.* 2019).

Table 2 DSK – German rock mass classification (Witthaus and Polysos 2007)

Rating index	Class	Rock type
0 - 80	Ia	Stable rock: Local displacement, closed joints and bedding elements (separation planes)
80 - 131	Ib	
131 - 196	IIa	Caving rock: Local displacement and sporadic caving areas up to decimeter size in the roof and the upper sides, particular separation planes
196 - 264	IIb	
264 - 304	IIIa	Friable rock: Increased separation results in displacements and caving up to meter size, separation planes pronounced and partially opened
304 - 347	IIIb	
347 - 434	IVa	Very friable rock: High density of jointing and intensive transaction results in regular displacement caving up to 1 m sliding gravity wedges
434 - 521	IVb	
521 - 621	Va	Squeezing rock: Local gouge zones and squeezing areas, opened separation plane, high density of separation and intensive transaction, loosening of strata, and high mobility of gravity wedges

For the specific purpose of support selection, miners in Germany have developed Deutsche Steinkohle Klasifikatio (DSK), which is a rock mass classification (Opolony and Witthaus 2003). This classification not only relies on uniaxial compressive strength as a standard parameter but also takes into consideration the nature and magnitude of discontinuities. Each evaluated characteristic has its own weight and, based on its total score, a rock mass is categorised in terms of quality, belonging to one of five classes (Witthaus and Polysos 2007) (Table 2). The most favourable parameter values are given the lowest scores. For example, a roof rock strength above 80 MPa is attributed a weight of 4.6. Conversely, the least favourable parameter values receive high scores so, for instance, if cracks run along the excavation axis, the score is 50.

American miners rely on their own roof rock classification, known as Coal Mine Roof Rating (CMRR) (Mark and Molinda 2003). CMRR is based on both rock strength parameters and the extent of layer separation. For this purpose, CMRR uses its Discontinuity Spacing Rating (DSR), calculated on the basis of Rock Quality Designation (RQD) or fracture spacing. This classification has also been widely used in mines across the Republic of South Africa, Canada and Australia (Calleja 2006).

An interesting solution for the design of excavation support was presented by Małkowski *et al.* (2016), where two indicators were presented: Roadway Design Efficiency index (*RDE*), which indicates the extent of problems that can be associated with roadway design, and Roadway Maintenance Functionality index (*RFM*), which indicates the difficulties that may occur during roadway use. Both indicators are based on geological, mining and technological data while rating points and weightings for the factors are obtained from the Analytic Hierarchy Process (*AHP*) method of analysis.

This article examines some factors and characteristics influencing the choice of support and the sustained stability of excavations. Also, based on a study of the rock mass in the immediate vicinity of excavations located at various depths, the rock mass characteristics that change and significantly affect the choice of support are identified.

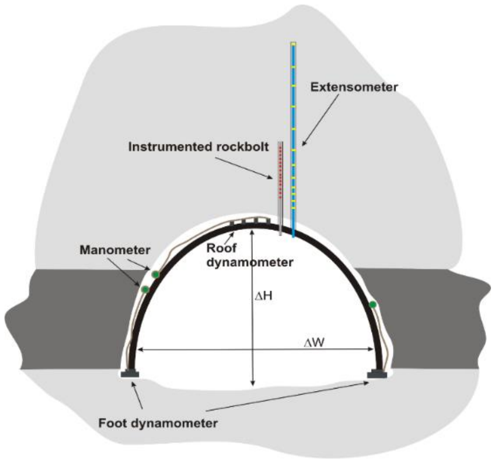

Fig. 2 Measuring station

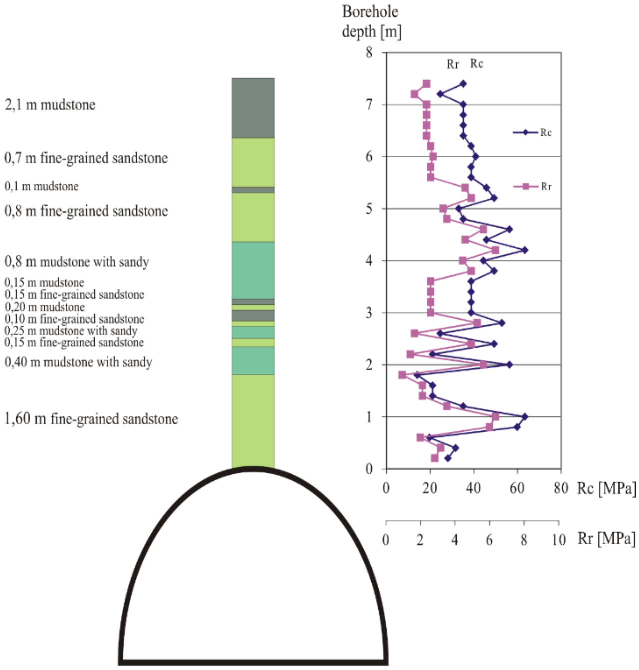

Fig. 3 Penetrometer test results for one of the analysed excavation sites in view of the roof rock lithology in excavation no. 1

2. Study findings

In order to assess the rock mass characteristics influencing the choice of support in excavations, regions with various geological and mining conditions were explored. In the analysed excavation sites, special testing stations were established to record rock mass behaviour, and support loading and deformations (Fig. 2). 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 the shape of a cylinder and a serial roof dynamometer;
- loading of rock bolts – using instrumented rock bolts with strain gauges;
- strength and fissuring of surrounding rocks in boreholes within 5 m of the measuring instruments;
- 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.

Fig. 3 shows the results of a penetrometer test, carried out to determine the strength properties of roof rocks, including their uniaxial compressive strength, *Rc*, and tensile strength, *Rr*, for each rock layer.

Another very important test, performed after penetrometer testing, is the observation of cracks and layer separations using an endoscope camera. Without doubt, both rock strength properties and cracks are factors that need to be considered when selecting support. Fig. 4 shows the results of strength testing in excavation no. 4, using instrumented bolts over a period of 267 days, and the discontinuities identified during an endoscope inspection during the construction of a measuring station. As can be seen, above the observed layer separations, instrumented bolts were not subject to significant loads. In an area below the discontinuities, the axial forces exerted on the bolts were twice as high as in the area above. In other excavations, the situation was similar: the highest values of axial forces practically occurred in places of cracking and de-lamination. This indicates that rock layer separation is correlated with the loads borne by the supports installed in the excavation site. The analysis of the described process was possible based on the results of research on the number of cracks and de-laminations in the roof of the excavations.

Table 3 presents technical data for the excavations and *in situ* research results. Moreover, data was provided for each of the analysed cases, regarding excavation depth and endoscope test results, i.e., the number of discontinuities identified in the borehole using an endoscope camera. The data showed that in an excavation site with a large number of layers identified in the roof, crack and layer separation intensity is also high. This could be due to the fact that the

Table 3 Technical data of excavations and in situ research results

Excavation		1	2	3	4	5	6	7
Depth [m]		1290	1050	950	950	950	1080	838
Size of support		ŁPZ11 /V32	ŁP11/V32	ŁPCBor 12/V36	ŁP12/V32	ŁP12/V32	ŁP12/V32	ŁPCBor 12/V36
Type of steel		S480W	S480W	S550W	S480W	S550W	S480W	S550W
Spacing of frames [m]		0,5	0,5	0,8	0,5	0,5	0,6	0,6
Lining		manual	mechanical	manual	manual	manual	manual	manual
Additional reinforcement		Two anchored joists both in roof and walls	none	none	Two joists anchored in roof	Two joists anchored in roof	none	none
Number of rock layers in the roof		13	12	3	9	3	6	6
Number of cracks in the roof		36	27	0	7	10	2	5
RQD [%]		6,6	43,7	60,0	14,6	13,7	85,4	34,0
R_c [MPa]	Sandstone	43.11	-	57.66	-	43.36	51.61	39.34
	Sandy mudstone	-	41.08	-	37.72	-	-	-
	Mudstone (or mudstone with sandy)	33.73	34.97	43.89	29.55	18.86	-	41.13
	Coal	-	-	-	14.87	-	-	-
	Weighted average	38.1	40.52	55.89	27.87	23.31	51.61	41.26
R_r [MPa]	Sandstone	4.31	-	3.84	-	2.89	3.44	2.62
	Sandy mudstone	-	2.78	-	3.02	-	-	-
	Mudstone (or mudstone with sandy)	2.25	2.33	3.25	2.19	1.39	-	3.05
	Coal		-	-	0.53	-	-	-
	Weighted average	3.21	2.7	3.76	1.98	1.66	3.44	3.11
Maximal force on dynamometers [kN]		187	222	13	299	120	0	146
Maximal tensile force in instrumented rock bolt [kN]		277	281	153	150	255	229	265
Maximal dislocation of extensometer anchors [mm]		-	60	-8	72	-	1,6	18
Maximal change in height and width of yielding support arches in relation to normal dimensions [m]	ΔH	-	0,01	0,22	0,46	0,27	-	0,13
	ΔW	0,84	0,13	0,08	0,22	0,06	-	0,08

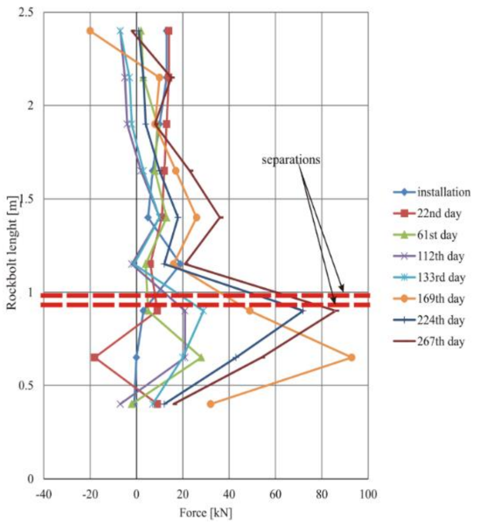

Fig. 4 Endoscope test results and instrumented bolt loads in excavation no. 4

surface between two rock layers is characterised by a lower degree of cohesion, which causes discontinuities. The analysis of the roof layer lithology showed that in nearly all of the excavation sites there was sandy mudstone, while mudstone (or mudstone with sand) proved to be much rarer. It should be noted that sandy mudstone has a high sand content, while mudstone with sand is characterised by a small amount of sandy material. The quality of the roof rocks translates into the results obtained from the measuring stations. The force in the dynamometers was 299 kN (in excavation no. 4) and in the instrumented rock bolt, it was 281 kN (in excavation no. 2). The high number of rock layers in the roof affects its displacement, which was 60 mm in excavation no. 2.

Based on tests in the mines, two new parameters were defined to describe the quality of the rock mass located in the immediate vicinity of the excavation site. The first parameter concerns the roof rock lithology and is referred to

Table 4 Reduction coefficient r

Rock lithology	Reduction coefficient *r*
Sandstone	1.00
Sandy mudstone	0.77
Mudstone (or mudstone with sandy)	0.66
Coal	0.27

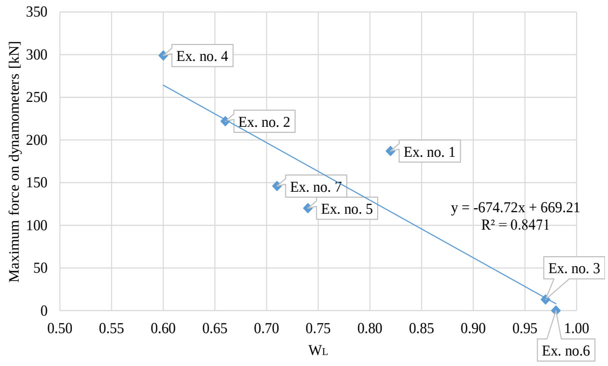

Fig. 5 Correlation between the maximum load of yielding arch support and roof lithology index W_L

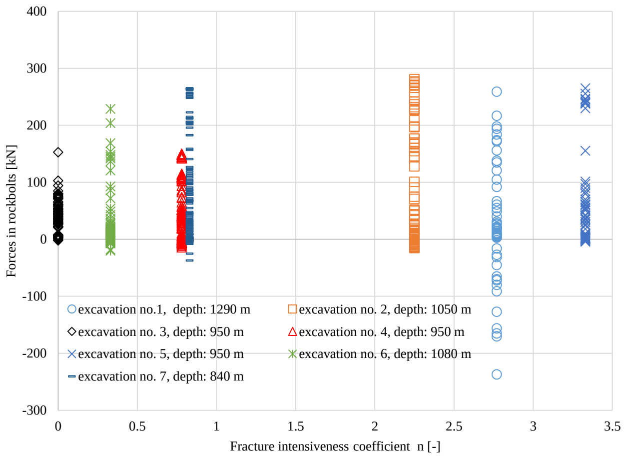

Fig. 6 Axial forces recorded by instrumented bolts depending on the crack rate

as the roof lithology index W_L. The index is calculated as a weighted mean of the rock layer thickness in the analysed excavation site (1). The thickness of individual rock layers is multiplied by the reduction coefficient for the specific layer. The product of this multiplication provides information on the layer that dominates the roof rock at the excavation site.

$$W_L = \frac{\sum h_i \cdot r_i}{\sum h_i} \tag{1}$$

where:

h_i – thickness of the i – rock layer, [m];

r_i – reduction coefficient for the i – rock layer, which depends on rock lithology [-].

The reduction coefficient r was determined on the basis of many in situ (used penetrometer) and laboratory tests on core samples. Laboratory tests included the following determinations: compressive strength, tensile strength, Young's modulus, and Poisson's number. In total, over 360 samples were tested. It was assumed that there is a specific correlation between the compressive strength and layer divisibility of the examined rock types. Table 4 shows reduction coefficients for individual types of rock and coal.

Our study showed that there is a correlation between crack intensity and the number of layers in the roof rock of the analysed excavation sites. The interface of two rock layers is characterised by lower cohesion, which can cause layer separation or cracks. Therefore, a densely layered roof at an excavation site can be susceptible to intensive discontinuities in the form of cracks and layer separation. On this basis, the crack rate n (2) was calculated as being the product of the number of cracks and the number of rock layers identified along a 10 m long section of the excavation site roof.

$$n = \frac{l_s}{l_w} \tag{2}$$

where:

l_s – number of cracks in the roof;

l_w – number of rock layers in the roof.

The above-mentioned characteristics describe rock mass quality. The first parameter, the roof lithology index W_L describes rock mass lithology and helps identify the dominant layer, while the crack rate n provides information about the discontinuities that exist in the roof and, in particular, their magnitude and density. Data collected during the observation of the excavation sites located deep underground were used to verify the above-mentioned characteristics and to assess their validity for the description of rock mass quality.

Our study showed that roof lithology largely contributes to the load carried by yielding arch supports. Fig. 4 shows the correlation between the maximum loads of yielding arch supports and the roof lithology index W_L. It is clear that the closer W_L gets to 1, the more stable the excavation site and the lower the support's exposure to significant loads from the surrounding rocks.

Fig. 5 shows the values recorded by instrumented bolts, depending on the crack rate. For each excavation site, the value of n varied because of the different numbers of lithological layers that were identified and cracks observed. The range of the axial forces exerted on bolt poles varied significantly. The number of points reflecting such forces depended on the measurement frequency, which was up to 17 readings over a period of 655 days (in excavation no. 2). For crack rates below 1, dispersion was low and the maximum force was 236 kN, which is lower than the load-bearing capacity of a bolt. It is clear that the lower the value of n, the greater the load on the bolt, which often exceeded its load-bearing capacity.

The characteristics describing rock mass quality, as described above, were used for the excavation site located 1,290 m below the ground. The excavation site employed a closed support with bolted binders in the roof and side walls. The length of the cable bolts in the roof was 4.5 m, while anchor bolts in the side walls were 3.0 m long. During excavation, site boring, penetrometer and endoscope tests were carried out. The tests results suggested an

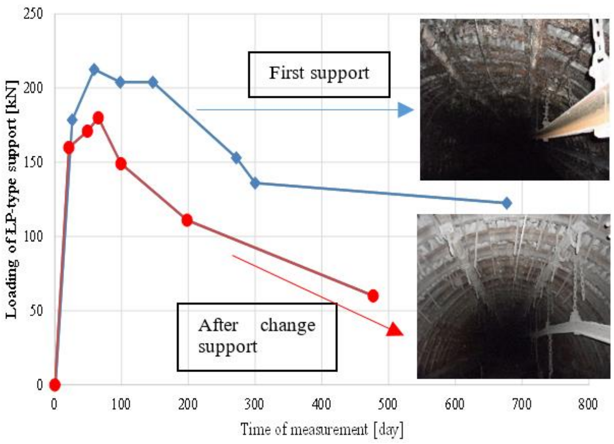

Fig. 7 Axial forces recorded by instrumented bolts depending on the crack rate

intensively layered roof and a dense and high-reaching network of cracks in the roof. At this stage of excavation (site driving), it was decided to use longer, 10 m cable bolts. Fig. 6 shows support loads separately for the site where shorter bolts and longer bolts were used, as well as site photographs of the excavation. It is evident that, as a result of modifications in the support used, support arch loads were significantly reduced. In addition, the dimensions and stability of the excavation sites were considerably improved, as can be seen in the photographs.

3. Conclusions

The analysis of the findings has arrived at the following conclusions:

• Penetrometer and endoscope tests provide a lot of useful information about the strength-related properties and condition of the rock mass in the immediate vicinity of the excavation site. The strength determined under mining conditions differs from that tested in a laboratory because core samples often come from the strongest rock layers.

• Knowledge of the extent and intensity of cracking makes it possible to correctly choose the support and use reinforcement, if necessary. Discontinuities in the roof of the excavation site often correlate with axial forces exerted on anchor bolt supports. The value of such forces near cracks or layer separation in a rock mass is increased nearly twofold.

• Information about the predominance of a specific lithological layer in the roof, as provided by the W_L index and the crack rate *n,* should be taken into consideration when selecting and designing excavation supports. If the value of the roof lithology index W_L is close to 1, this suggests that the roof is made of thick and strong sandstone layers. The higher the crack rate, the denser the discontinuities in the roof.

• Following the verification of rock mass quality during excavation site driving, it might become necessary to modify the support design. One example is an excavation site located 1,290 m below the ground, where, as a result of identifying a vast and intensive network of cracks, a decision was made to use longer cable bolts. This modification resulted in a reduction of the loads exerted on the supports by the rock mass and helped to maintain the dimensions and stability of the farthest section of the excavation site.

• The indicators presented in the article can also be used to support tunnel design. It is important to control the quality of the rock mass during the driving of both excavations in mines and tunnels. The support can then modify if there are deteriorating conditions.

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