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CASE STUDY OF ROCK BOLTING IN A DEEP COAL MINE IN POLAND

In 2017, the Central Mining Institute (GIG), Jastrzębska Spółka Węglowa SA (JSW SA), the largest producer of coking coal in Europe, and JOY KOMATSU, the producer of mining machinery, signed a consortium. The project's main goal was to reduce the costs of driving mine workings by reintroducing the rock bolt support. The works began in November 2019, and for the first time in the history of Polish coal mining, a Bolter Miner machine was used for the purpose. The paper presents the results of measuring the axial forces in rock bolts at the measurement base and their analysis with numerical modelling.

Keywords: coal mine; rock bolting; strain-gauged rock bolts; numerical modelling

1. Introduction

For the first time, bolts in mining were mentioned in 1872 in North Wales [35] and 1905 in the USA [23]. However, the actual development of rock bolting began in 1913 with Stephan, Fröhlich and Klüpfel's patent No. 302909. The primary purpose of the new type of support was to replace the traditional timber props set on the floor. To achieve it: "boreholes of sufficient depth will be drilled into the rock in which rods, tubes or cables made of a load-bearing material, for example, steel, will be inserted and properly fixed at the end or cemented along the whole length" [18]. The idea behind the solution is presented in Fig. 1.

After the first use of bolting in 1905, as mentioned in the introduction, it was successfully applied in the Sagamore mine in Southern Virginia, USA, in 1917. In the early 1920s, the first mining company, St. Joseph Lead Company, started to apply bolts broadly [3]. The actual expan-

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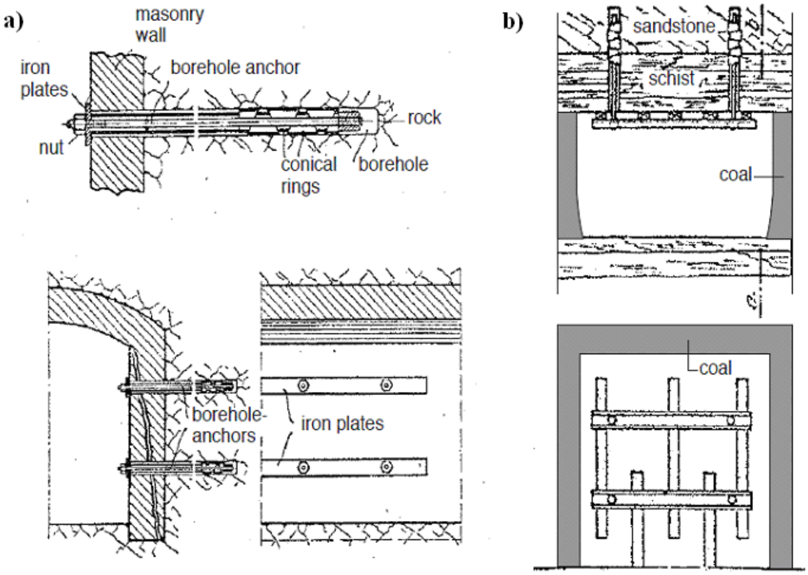

Fig. 1. First application of rock bolting support (a) in sidewalls, (b) in the roof of a mine working [18]

sion of the rock bolt support in the USA began in the late 1940s, with the significant contribution of the U.S. Bureau of Mines, which became a champion of this new technology [24]. The publication of Weigel [42], who presented the basics of rock bolting, which remain valid, also had a positive influence. The development of bolting technology was so rapid that more than 200 mines applied this type of support in less than two years [21]. As Fletcher [9] wrote: "roof bolting has been adopted more rapidly than any other new technology in the history of coal mine mechanization", and the statistical data prove it the best. In 1976, about 100 million bolts were used; in 1984, 120 million; in 1988, 88 million; in 1999, 100 million; and in 2005-2006, 68 million [8,21,30,37,38]. It is estimated that around 68-80 million units are currently used annually [16,36]. The decrease in demand observed over the years is related to the growing share of the longwall mining system, which uses only about a quarter of the number of bolts compared with the room and pillar method.

The history of rock bolts in Australia started with the construction (1949-1969) of the Snowy Mountains Hydroelectric Scheme and irrigation complex. Then the type of support was applied for the first time to strengthen the rock mass [7]. The first mine to use rock bolts together with timber support was the Elrington mine, located in New South Wales in 1949 [13]. In the 1970s, timber supports were predominant in the Australian mining industry. They were supplemented with steel supports and, relatively rarely, with mechanical anchors. Significant changes in the share of the individual rock bolt support came in the late 1970s and early 1980s due to, among other things, the increasing number of accidents with the arch support, the economic calculation, the introduction of modern adhesive loads and the increasing offer of rock bolting equipment. The next step in the early 1990s was the introduction of cable bolts as a form of reinforcement of the existing roadways and as the main element of the support system [12]. The rock bolt support is the principal method to strengthen rock mass in hard coal mines.

In Great Britain, in 1945-1946, a comprehensive publication was issued. Six books of Colliery Engineering, by Dutch engineer Z.S. Beyl, contained the positive experiences with rock mass anchoring in England in 1942-1943. Even then, based on convergence measurements, he noted that "it is necessary to insert the anchors as soon as possible after the exposure". Despite the studies, the type of support did not spread immediately, but they formed the basis for its development in the late 1950s [31-33]. The real revolution in the use of bolts in Great Britain began in the 1990s due to ownership changes. As a result of the activities, with an adequate level of safety, rock bolting was successfully introduced in the coal mining industry in Great Britain. In 2006, about 95% of all new mine workings had rock bolt support [43].

In China, rock bolts have been used in underground coal mines since 1956 [17]. In 1995, rock bolting was used in about 15% of roadways. In 1996-97, the Australian bolting technology was introduced, which introduced new materials, especially in the form of fully grouted bolts of significant load capacity. Another important date for coal mining in China was 2005 when a research program for supports applied in deep mines and in difficult geological and mining conditions began. The program resulted, among other things, in increasing the mining face advance rate, driven with rock bolting support. Currently, in the state-owned coal mines, about 70%, and, in some regions, even 90% of the mine workings have rock bolt support [16].

In Germany, before World War II, the rock bolt support was rarely used [33], and its first tests were conducted in 1948-51 in the coal mines: Consolidation, Neumühl and Diergradt-Merissen [31]. In 1960-1975, the use of rock bolting was almost abandoned. After 1975, again, there was interest in the type of support, but it never gained a significant share in the total length of the driven mine workings [33]. At the end of the 1990s, there was some rekindled interest in using rock bolt support, which was reflected in the publications [4,14,20,29]. However, since 2006, the type of support has hardly ever been used.

Other countries where the rock bolt support is applied are: the Czech Republic [15,19,40,41], Russia [1,10], India [2,25], the RSA [6] and Iran [34,39].

In Poland, the Pokój mine was the first one to use the rock bolt support as early as in 1916 [5,18,31]. It was applied in the mine working with the sidewalls damaged due to the impact of floor rocks. The applied bolts were 2.0-metre long, the spacing was 1.0 metres, and a flat steel bar connected them. In the same mine, there was an attempt to rock bolt the roof. However, such a support system was too innovative to be widely used. In the post-war period, there were other significant events related to the use of bolts in the Polish coal mining industry. In 1954, the first experimental roof bolting, based on scientific foundations, was conducted by the Central Mining Institute (GIG) [31]. In 1960, an instruction for the execution and control of rock bolting adapted to the mining needs was issued [28], and in 1976 the Ministry of Mining issued "Temporary guidelines for the use of rock bolt support, rock bolt and arch support, and arch support in hard coal mines", which were followed until the early 1990s. In the 1990s, an individual rock bolt support was used more commonly in Poland and for roadways. However, in 1994 and 1995, two of such workings collapsed, resulting in decreasing interest in the type of support until 1999. Again, there was some interest in rock bolting in 2000 ÷ 2002 related to the restructuring processes. Yet, from then on, the length of workings with the type of support systematically decreased until it was abandoned entirely as individual support in 2009 [26,27]. The situation is shown in Fig. 2.

The situation changed again in 2017, when a consortium between GIG, the largest producer of coking coal in Europe, Jastrzębska Spółka Węglowa SA (JSW SA) and JOY KOMATSU, the

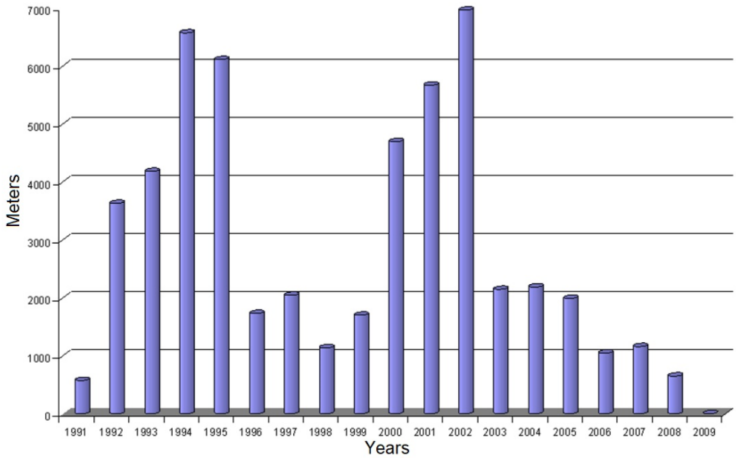

Fig. 2. Length of mine workings driven with an individual rock bolt support, 1991÷2009 [26,27]

producer of mining machinery, was signed. The project's main goal was to reduce the costs of driving mine workings by reintroducing the rock bolt support. The works began in November 2019, and, for the first time in the history of Polish coal mining, a Bolter Miner machine was used.

2. Geological and mining conditions

After detailed analyses of the geological and mining conditions in JSW SA, it was decided that the rock bolt support would be used in the Budryk mine. The mine is located in the south of Poland, in the Upper Silesian Coal Basin, and it started production in 1994.

The mine working supported with rock bolts was located at a depth of 880-900 m, inseam 401. It had a variable thickness between 0.5 and 2.0 metres. The seam in the part of the deposit generally fell southeast at 5 to 9° angle. In the first stage, it was assumed that about 2,000 metres of mine workings would be driven, which was completed, see Fig. 3. As part of the works, three turns were also made. The first two at obtuse angles of 135° and 146°, and one at an acute angle of 77°.

The basic parameters, determined by analysing the tests performed with a hydraulic borehole penetrometer and laboratory tests (RQD, slakeability r), are presented in Table 1.

TABLE 1

Results of measurements

	Depth [m]	**UCS [MPa]**	**UTS [MPa]**	**RQD_p penetrometer [%]**	**RQD [%]**	**Slakeability r [-]**
Roof	0.0-3.0	36.91	2.03	88.0	52.3	1
	0.0-9.8	32.06	1.76	64.0	29.4	1
Coal	0.6-3.3	16.18	0.89	—	—	—
Floor	0.3-0.8	20.03	1.10	—	—	—

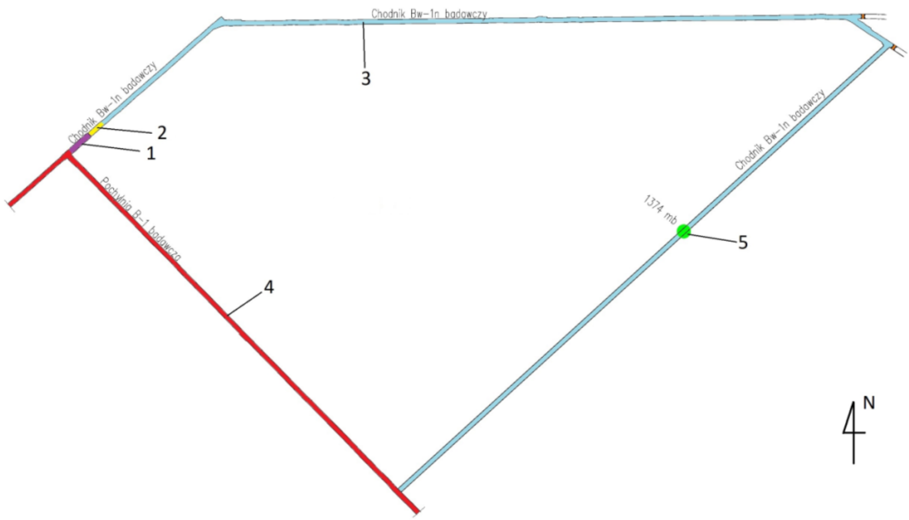

Fig. 3. Fragment of seam 401 map with the location of the measurement base
1 – Assembly room for Bolter Miner, 2 – Standing support + rock bolts, 3 – Rock bolt system,
4 – Arch yielding support, 5 – Measurement base No. 7

The basic dimensions of the mine working were 5.6 × 3.4 m (w × h), and due to changing geological conditions, there were places where its height increased to about 4.0 m. The rock bolt support was selected following GIG's analytical and empirical method and confirmed by numerical calculations. The primary rock bolting grid consisted of six 2.5-metre-long steel bolts of a minimum load capacity of 260 kN, fully grouted with DUO SPEED polyester loads. The row spacing was 0.8 m. Additionally, two 5.0 or 6.0 m long cable bolts of a minimum load capacity of 320 kN were used between the rows. There were six anchors installed in the sidewalls. Their parameters were compatible with the rock bolts in the roof. An additional bolt was used when the distance between the sidewall bolts and the floor exceeded 1.2 m. The rock bolting grid is shown in Fig. 4, and Fig. 5 shows the mine working.

3. Monitoring system

The continuous monitoring of the workings was based on dual height tell-tales, mounted in boreholes at the depths of 2.1 and 5.0 metre. They were installed in the mine working every 20 metres. Moreover, three-height tell-tales were also installed to monitor geological disturbances and other changes, e.g. while driving extensions. To monitor the behaviour of the mine working, measurement bases were established, consisting of the following elements:
- strain-gauged rock bolts, whose distribution, spacing, and length were compatible with the rock bolt support – 6 rock bolts per row,
- three height tell-tales, at the depth of 2.1, 5.0, 6.0 m,
- multi height tell-tales, at the depth depending on current needs,
- convergence measurements in properly marked sidewall bolts and roof rock bolts.

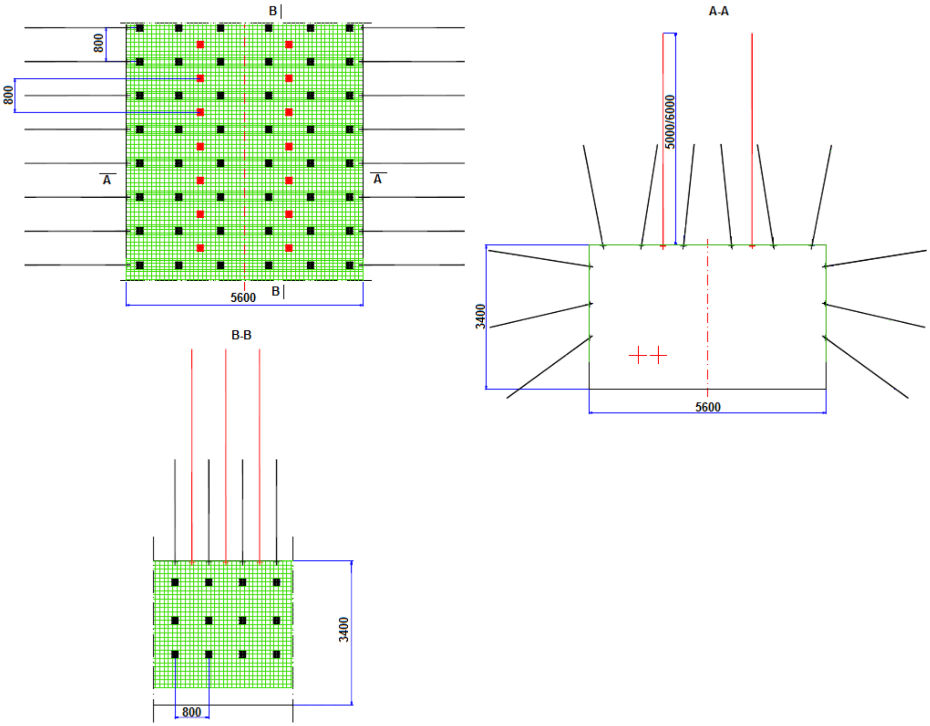

Fig. 4. Rock bolting grid in the Budryk mine

Fig. 5. Bolter Miner 12CM30 and the rock bolt support in the Budryk mine

In the measurement base no. 7, presented in Fig. 3, strain-gauged rock bolts and three height tell-tales were applied, see Fig. 6.

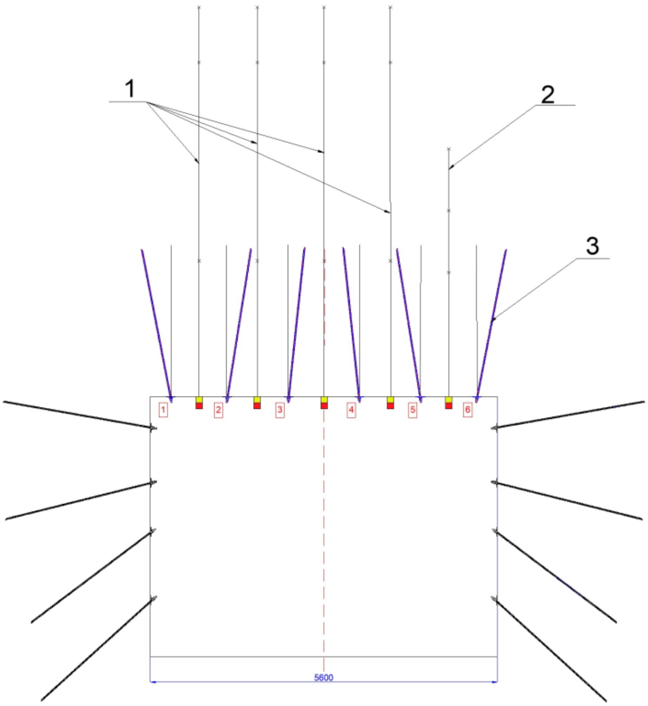

Fig. 6. Measurement base diagram
1 – three height tell-tales: 2.1, 5.0, 6.0 m; 2 – three height tell-tales: 2.0, 3.0, 4.0 m;
3 – 6 × 2.5 m strain-gauged rock bolts

Due to the limited volume, the article focuses only on the analysis of the results of measurements made with the strain-gauged bolts.

4. Results of underground measurements

Fig. 7 shows the results of measurements of the maximum axial forces in rock bolts depending on the distance from the mining face.

From analysing the results of the underground measurements (see Fig. 7), up to the distance of 36.4 m from the mining face, the values of the axial forces in the rock bolts change significantly. They both increase and decrease in axial forces. It is due to the dynamically changing conditions in which the rock bolts work. Initially, we have added support for the created rock beam at the mining face. As the work progresses, the beam is supported only on the opposite sidewalls, and with time the forces in the rock bolts stabilise. Similar behaviour of the axial forces was observed in all the measurement bases.

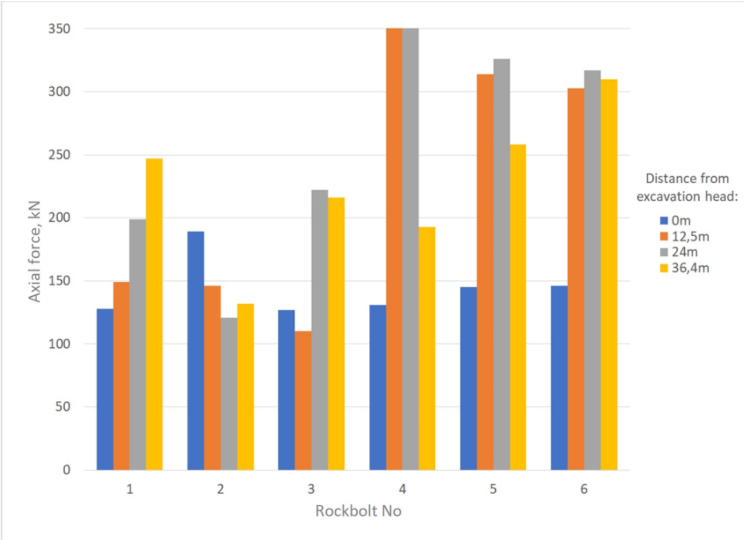

Fig. 7. Values of maximal axial forces in rock bolts depending on the distance from the mining face

Due to the variable values of the axial forces in the rock bolts, among which the dependence of their changes cannot be determined, it was decided to map, with numerical modelling, the situation when the distance between the measurement base and the mining face was 52.8 metres.

5. Numerical modelling

We applied the finite element method Phase2 software to map the geological and mining situation in the analysed mine working. The numerical modelling aims to map the axial forces in the rock bolts so that the calibrated numerical model could be used for subsequent calculations, e.g. related to the impact of the longwall face advance on a mine working with individual rock bolting.

The dimension of the numerical model was 75 × 100 m. The boreholes drilled in the roof, G7(2020)8; the floor, G7(2020)9, at the 1,375th metre of the mine working, and borehole G16(2006), were analysed. On this basis, the type and mutual deposition of the rocks were assumed. Fig. 8 shows a fragment of the numerical model of the rock mass. The Mohr-Coulomb strength criterion was adopted in the calculations. It was assumed that the modelled rock mass is an elastic-plastic and isotropic medium with no possible horizontal displacements on the vertical and horizontal boundaries of the model. Measurements of the primary stress state in the rock mass around the mine working with a rock bolt support were made by Golder Associates (UK) Ltd [11]. Based on them, the model determined the ratio of horizontal stresses to vertical stresses, which is, on average, $\lambda = 1.03$ and the ratio of horizontal stresses to vertical stresses perpendicular to the model, which was assumed to be 0.75.

Based on laboratory tests, the deformation properties of rocks in given rock layers, including Young's Modulus E, Poisson's ratio υ, were adopted. Strength parameters, i.e. tensile strength R_r.

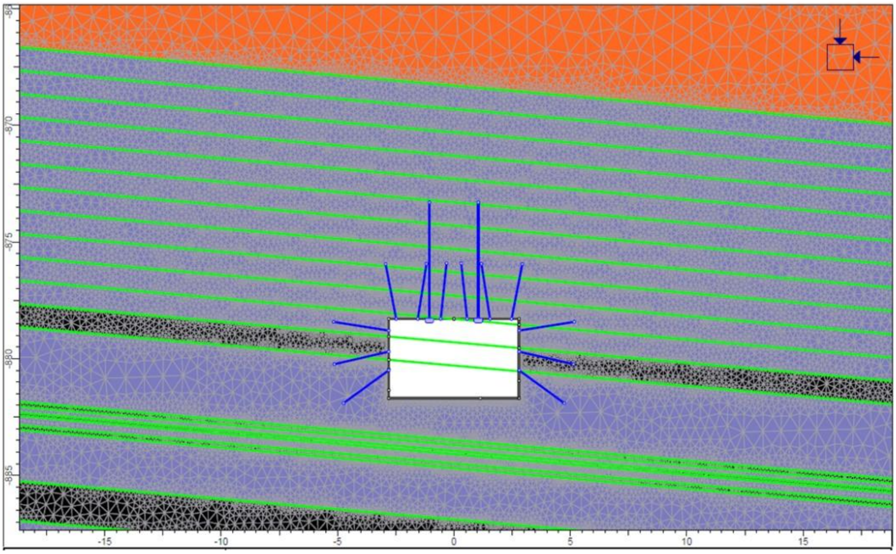

Fig. 8. Fragment of the numerical model of the rock mass

and compressive strength R_c, were determined with penetrometric tests. Cohesion c was calculated based on compressive strength and tensile strength [22]. The relationship is described by the formula (1). Table 2 shows the values of given properties of rocks based on which the numerical model was calibrated.

$$c = 0.5\sqrt{R_c R_r} \tag{1}$$

where:

R_c — compressive strength,

R_r — tensile strength.

TABLE 2

Strength and deformation parameters of rocks, assumed for numerical modeling

Type of rock	**Young's modulus E [MPa]**	**Poisson's ratio v**	**Tensile strength R_r [MPa]**	**Angle of internal friction φ [^{o}]**	**Cohesion c [MPa]**
coal	1900	0.3	0.9	30	1.9
Clay shale 1 – roof	6940	0.21	2.54	32	4.67
Clay shale 2 – roof	3580	0.24	1.8	28	3
Sandstone – roof	8250	0.19	3	35	5.73
Sand shale	9500	0.21	3.28	34	6
Clay shale 1 – floor	6860	0.22	2.4	28	4.23
Clay shale 2 – floor	6640	0.22	2.68	32	5

The developed numerical model assumes stratification of the clay shale layer in the roof, which, according to us, is the closest to the actual conditions of the described rock. The several-meter-thick clay shale roof layer consists of many thinner layers of various strength parameters, confirmed with penetrometric tests. Hence, clay shale in the roof of the mine working was modelled as alternate 1-metre thick layers of slightly different parameters, specified in Table 2.

The model also shows the support, consisting of a row of six 2.5-metre-long Ø21.7 mm steel rock bolts of load capacity 0.26 MN and Young's Module of 240 GPa in the roof. The primary rock bolting grid is complemented by two 5.0-metre-long Ø18 mm cable bolts of load capacity of 0.32 MN, installed between the rows of the primary grid. In the model, the influence of the cable bolts was simulated for their load capacity reduced by half, i.e. 0.16 MN.

The performed numerical calculations consisted of three stages. The first stage was to establish the equilibrium state in the model before driving the designed mine working. In the second stage, the state of stress in the rock mass was obtained, immediately after driving, before rock bolting, which allowed for partial destressing of the rock mass. The third stage reflected the stress distribution around the mine working after rock bolting.

6. Analysis of results of numerical modelling

The predicted maximum values of the axial forces in the rock bolts 1 to 6 (Fig. 9) are shown in Fig. 10.

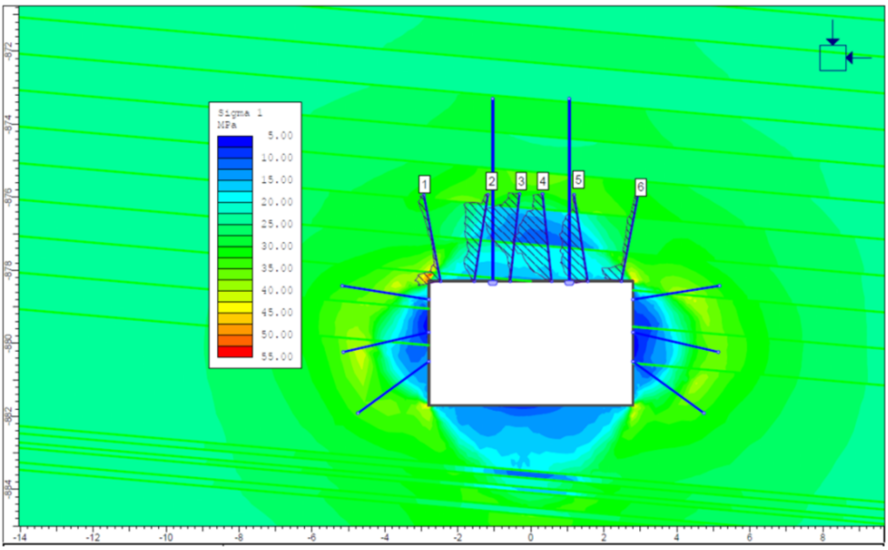

Fig. 9. Results of the numerical modeling

Based on the results of numerical calculations, it is concluded that the highest values of axial forces occur in rock bolts 3 and 4, i.e. the ones located closest to the axis of the mine working. The maximum axial forces are 250-260 kN. For rock bolt no. 3, the highest values are at 1.5-2.0 metres from the opening of the hole. Then they rapidly drop to the value of about 80 kN.

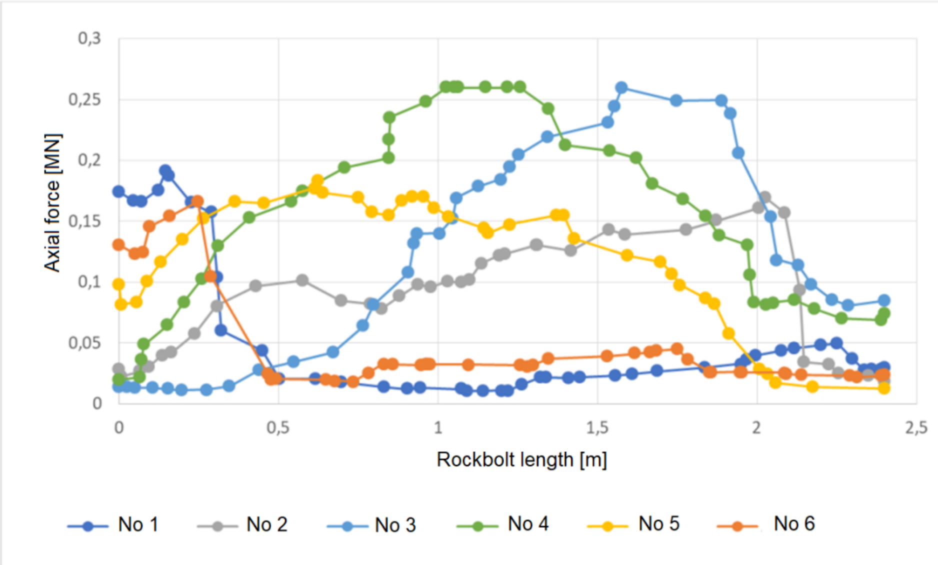

Fig. 10. Axial forces in the rock bolts

Rock bolt no. 4 shows the maximum value of axial forces at 0.8 to 1.3 metres of its length. Then it gradually lowers, reaching a value of about 70 kN at its end.

Rock bolts no. 1 and 6 (extreme bolts) reach the maximum value of axial forces at the initial length of the bar, i.e. 0.0-0.3 m. In rock bolt no. 1, the value is approximately 190 kN, while in rock bolt no. 6, approximately 170 kN. Along the remaining length, they show relatively low axial forces, which do not exceed 50 kN.

The last of the analysed rock bolts, no. 2 and 5, were installed, as shown in Fig. 8, between the extreme and central rock bolts. The maximum value of axial forces for rock bolt no. 2 is approximately 150 kN, and it occurs at the 2nd metre. In rock bolt no. 5, it reaches the value of about 180 kN at 0.7 metres.

The comparison of the maximum values of axial forces in the rock bolts according to the underground measurements and numerical modelling is presented in Table 3.

TABLE 3

Comparison of axial force values measured in rock bolts and obtained with numerical modelling

Rock bolts	**Maximum value of the axial force [kN]**		**Difference %**
	***In situ* measurements:**	**Numerical modelling:**	
1	198	192	–3.03
2	165	170	+3.03
3	248	259	+4.43
4	262	260	–0.77
5	244	178	–27.05
6	221	167	–24.43

Comparing the underground measurements of axial forces in the rock bolts with the results of numerical calculations, as shown in Table 3, it can be concluded that for rock bolts 1 to 4, the value of the tested parameter is compatible with the *in situ* tests and numerical modelling. The difference is slight and amounts to a maximum of 11 kN (4.4%). The rock bolts are located on the left side of the mine working (inby).

Significant differences in calculations and measurements, up to 27%, are characteristic for bolts 5 and 6, located on the right side of the excavation (inby). The reason is undoubtedly the effect of additional horizontal stresses, which was observed when the mine working was driven. Considerable sections of the right side (inby) required appropriate reinforcements, as characteristic deformations formed on the roof (see Fig. 11).

Fig. 11. The mine working (outby)

Fig. 12 shows examples of three different characteristics of the adjustment of the distribution of axial forces for numerical modeling and in-situ measurements along the length of the analysed bolts.

The first example (Fig. 12a) shows that the measured and the calculated courses of axial forces in rock bolts are almost identical. The force increases uniformly to the maximum value of approximately 260 kN at the 1.25th metre and then decreases to around 70 kN. The differences can only be observed at the 1.5th metre of the rock bolt, where, in the underground conditions, a local decrease in the value of the force was measured.

When analysing the axial force graph of rock bolt 5 (Fig. 12b), it can be noticed that the course of the axial force values along the bolt is very similar whilst there are differences in the maximum values, as shown in Table 3.

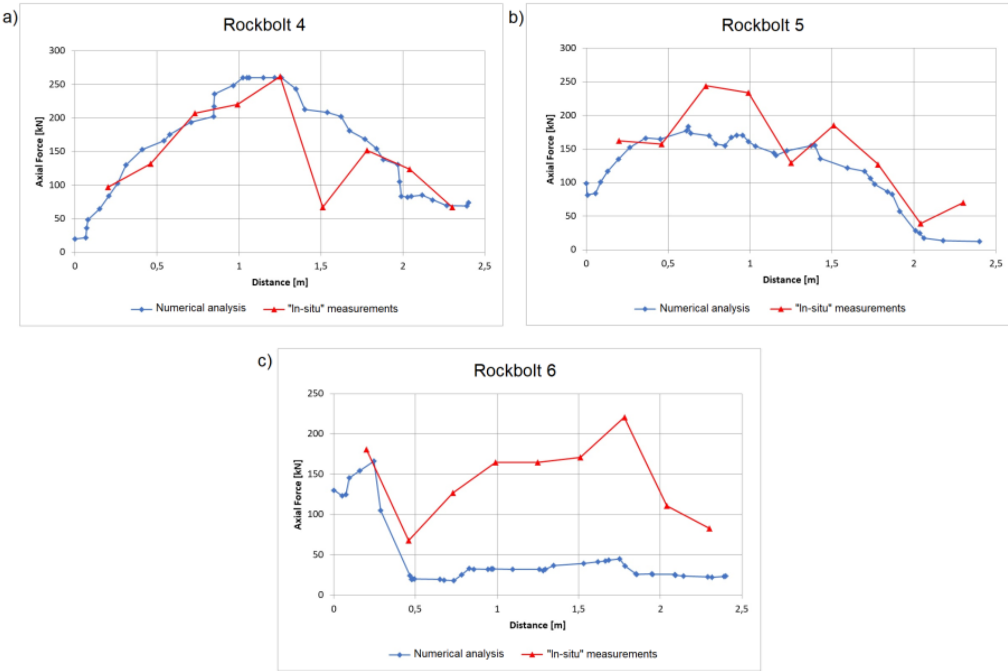

Fig. 12. Axial forces in rock bolts – comparison of underground measurements and numerical modelling

The comparison of the axial force for the in situ measurements with the predicted values of the rock bolt (no. 6) on the extreme right (inby) shows that the initial part of the rock bolt is subjected to a force of approximately 180 kN, which the numerical model confirms. Then its value drops to 70 kN (the 0.5th metre of the rock bolt), where, with the numerical model, it was possible to determine the force at 20 kN. For the remainder of the rock bolt, the value of the axial force increases until it reaches approximately 220 kN (the 1.75th metre of the rock bolt). The numerical model indicated that in the further section of the rock bolt, the value of the tested parameter would not exceed 45 kN (Fig. 12c).

7. Summary

Using the Bolter Miner, for the first time in the conditions of the Polish hard coal mining industry, was undoubtedly a breakthrough event. The comeback of rock bolt support more than a decade after its discontinuation, the state-of-art available tunnelling technology, different logistics were the challenges that all those involved in the project of reintroducing the support in the conditions of Jastrzębska Spółka Węglowa S.A. had to face. After the first stage of the implementation, i.e. completing approximately 2,000 metres, it appeared successful.

Due to its framework, the paper presents only a fragment of the issue of bolt application, i.e. the measurement of axial forces in rock bolts and subsequent analyses with numerical modelling. It allows, among other things, to use a properly calibrated numerical model for further calculations related to, for example, different mining conditions. Also, it was shown, the proper calibration of numerical models is not an easy issue. Under the conditions of the Budryk coal mine, local geological conditions, together with a significant influence of horizontal stresses on the behaviour of the roof, affect the obtained results of calculations. Therefore, the works to further calibrate the numerical models are continued.

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