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ANALYSIS OF YIELDING STEEL ARCH SUPPORT WITH ROCK BOLTS IN MINE ROADWAYS STABILITY ASPECT

ANALIZA SYSTEMÓW OBUDOWY PODPOROWO-KOTWOWEJ W ASPEKCIE STATECZNOŚCI WYROBISK KORYTARZOWYCH

The result of the search for new technological solutions in the field of support for roadways in coal mines has in recent years been the widespread use of steel arch with rockbolt support systems. The efficiency of these systems is affected among other things by the option of installing rock bolts after the actual driving the mine roadway, the increased load capacity that these systems can support, and their resistance to dynamic weight. Large variation in the way that these steel arch support can be connected using different types of rock bolts necessitates mining research revealing the effectiveness of such solutions. Although the steel arch with rockbolt support system is used in the majority of European coal mines, it is still not possible to apply templates of schemes due to the diversity of geological and mining conditions. Therefore, throughout a period of several years, the authors of this article conducted research *in situ* under conditions of different schemes related to connecting arched support frames with rock bolts, with only selected results being presented in the article. The measurements of convergence, load supported by the system frame, load supported by the rock bolts, and the stratification of roof rocks were analyzed, carried out in two roadways with yielding steel arch support in which strand bolts were applied. The article also proposes the index for working maintenance n_{uw}, used in preliminarily assessing the stability of a given working with a limited number of data concerning geomechanical conditions. Additionally considered are empirical methods used in Poland for designing steel arch with rock bolt support systems.

The results of mine research indicate that strengthening yielding steel support with strand bolts through steel beams maintains the stability of a roadway, even when exposed to the exploitation stress. Aside from the impact of exploitation, deformations of the support system are negligible, despite the fact that the tensile forces acting on the rock bolts can reach values of up to 160 kN. Under favorable geological and mining conditions, support system frames can be spread up to 1.5 m apart when using rock bolts between them. The conducted measurement of convergence during a three year period revealed a compression amounting to a few centimeters. The results obtained by the research fully confirm the effectiveness of combined yielding steel arch with rock bolt support systems under different mining conditions.

Keywords: mine roadways, mining support, stability of roadways, rock mass monitoring

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Efektem poszukiwania nowych rozwiązań technologicznych w zakresie obudowy wyrobisk kory-
tarzowych w kopalniach węgla kamiennego, jest w ostatnich latach dość powszechne stosowanie obu-
dowy podporowo-kotwowej. Na jej efektywność wpływa między innymi możliwość zabudowy kotew
w późniejszym okresie niż drążenie wyrobiska, uzyskiwanie dużej nośności, czy odporność na obciążenia
dynamiczne. Duża liczba możliwych do zastosowania wariantów połączeń obudowy podporowej z ko-
twami o różnej konstrukcji i rozmieszczeniu, wymusza konieczność prowadzenia badań kopalnianych
wskazujących na skuteczność zastosowanych rozwiązań. Choć obudowa podporowo-kotwowa stosowana
jest w większości europejskich kopalń węgla kamiennego, to ze względu na odmienność warunków
geologiczno – górniczych, nie można wprost przenieść gotowych schematów. W związku z tym od
kilkunastu lat autorzy artykułu prowadzą badania w warunkach *in situ* różnych schematów połączeń
odrzwi obudowy łukowej z kotwami, a w artykule przedstawiono wybrane wyniki badań. Poddane
analizie zostały wyniki pomiarów konwergencji, obciążenia odrzwi, obciążenia kotew i rozwarstwienia
skał stropowych w dwóch wyrobiskach z obudową podporowo-kotwową w których zastosowano kotwy
strunowe. W artykule zamieszczono także propozycję wskaźnika utrzymania wyrobiska n_{uw}, służącego
do wstępnej oceny stateczności projektowanego wyrobiska przy ograniczonej liczbie danych dotyczących
warunków geomechanicznych. Zamieszczono także stosowane w Polsce empiryczne metody projektowania
obudowy podporowo-kotwowej.

Wyniki badań kopalnianych wskazują, że zastosowanie wzmocnienie obudowy podporowej kota-
mi strunowymi poprzez podciągi, pozwala na utrzymanie stateczności wyrobiska nawet w przypadku
oddziaływania ciśnienia eksploatacyjnego. Poza zasięgiem oddziaływania eksploatacji deformacje obu-
dowy są niewielkie, choć siły rozciągające w kotwach dochodzą do 160 kN. W korzystnych warunkach
geologiczno-górniczych możliwe jest rozrzedzenie odrzwi obudowy nawet do 1,5 m przy zastosowaniu
kotwienia pomiędzy odrzwiami. Prowadzony pomiar konwergencji w okresie trzech lat wykazał zaciska-
nie wynoszące kilkanaście centymetrów. Uzyskane wyniki w pełni potwierdzają skuteczność obudowy
podporowo-kotwowej w różnych uwarunkowaniach górniczych.

Słowa kluczowe: wyrobiska górnicze, obudowa wyrobisk, stateczność wyrobisk podziemnych, moni-
toring górotworu

1. Introduction

Excavating of roadways requires support systems, which guarantee effective operation and
use during their time of functioning. Taking into account the significance of some opening-out
and development headings, which remain in use for long periods of times transport or ventilation
sites, the choice of support for these sites is hugely important – not only for one single area of
exploitation, but for the whole mining plant.

The activities surrounding the selection of support for roadways are primarily focused on
seeking the ones with a high load capacity. Striving to achieve greater durability is also important
(Majcherczyk et al., 2003; Brodny, 2012; Rotkegel, 2013), as well as limiting financial expen-
ditures by reducing the quantity of materials used. Experience has shown that is it difficult to
harmonize economy with safety and durability. Support system construction must simultaneously
be simple, easy to execute, and should take up as little space as possible in the mine working.
The system cannot inhibit the technological processes related to mining, transport, hauling, etc.
Furthermore, it should be as cheap as possible. One of the solutions meeting the above criteria is
a system of yielding steel support with rockbolts (Majcherczyk et al., 2011; Turek et al., 2012).

In the beginning of the nineties of the previous century, wide-ranging measures were taken
to introduce rockbolt systems for roadways in coal mines. Parallel to the implementation of roof
bolting, this time period also saw the application of mixed methods of roadway support, combin-
ing yielding steel arch support systems with rock bolts. This usually took place by bolting roof
arch beams with one or two pairs of steel rock bolt rebars.

Furthermore, the development of new solutions, including adhesives and rock bolts, brought about greater variation in bolting schemes. If bolting is done using steel beams with spaces for bolts to be fixed in, this yields better stabilization of a support system. This solution works especially well in areas of high tension, e.g. in gates, or junctions (Toraño et al., 2002; Majcherczyk et al., 2003, 2011; Prusek & Lubosik, 2007), as well as when dealing with heavy and dynamic loads (Horyl & Šňupárek, 2012; Turek et al., 2012).

Generally speaking, bolting the arch steel support system, as well as mounting rock bolts in between individual frames, will not only strengthen the rock mass and reduce the load supported by the system frame, but result in a more stable construction altogether (Majcherczyk et al., 2003). In coal mines, the use of steel arch with rock bolt support systems is fairly common in the majority of European countries (Toraño et al., 2002; Woll et al., 2004; Kovalevska & Fomychev, 2011; Horyl & Šňupárek, 2012).

Presented in this work is the application of design methods for steel arch with rock bolt support systems in Poland, and the results of such systems with respect to roadway stability analysis. The conclusions presented are the results of widely-conducted mining research over the course of a dozen or so years.

2. The stability of roadways

In the context of geomechanics, stability is understood as a state, in which mechanical processes occurring between the support and the rock mass are controlled and predictable. It is also said, that the criterion of stability in an excavation site is met when the load carried by the system in question is smaller than the maximum acceptable load, and when displacements of the surrounded rock mass are smaller than expected.

If one or both of these conditions are not fulfilled, then the stability of the excavation site is at stake. Loss of stability in a given site, manifested in the destruction of a support system or excessive displacement, is caused mainly by phenomena occurring in the rock mass surrounding the site, after it has been processed and during the time of routine site maintenance. The roadways' drivage, especially at great depths, results in the filling of empty space with deformed rocks, unless the process is halted, for example by the installment of a support system that provides an appropriate level of support. The support offered by a given system depends on its construction, which should be adapted to the specific geomechanical conditions so as to maintain the rock mass stability of the site (Majcherczyk et al., 2008).

For preliminary assessment of the potential of a support system to maintain a proposed excavation site within a specific time period – the amount of data related to geomechanical conditions being limited – we can use n_{uw} – the working maintenance index (Niedbalski, 2014):

$$n_{uw} = \frac{R_{clab}}{p_z} \cdot \frac{RQD}{100}$$ (1)

where:

n_{uw} — index of working maintenance obtained on the basis of laboratory-specified strength for uniaxial compression, -,

R_{clab} — the uniaxial laboratory compressive strength of roof rocks in a planned excavation site, MPa,

p_z — principal vertical stress, MPa,
RQD — Rock Quality Designation index, %.

For an average core divisibility value RQD totaling 50% – and assuming a resistance to uniaxial compression strenght of 30 to 80 MPa obtained on a laboratory basis, with depths varying from 500 to 1300 m – the values of site maintenance n_{uw} index change in accordance with the relationships outlined in Figure 1. Assuming that significant difficulties in maintaining an excavation site will appear for n_{uw} values lower than 1.0, it can be concluded that the divisibility of rocks and resistance to uniaxial compression strenght will be of decisive importance. For RQD equal to 50% and at low values of uniaxial compression resistance for roof rocks of 30 MPa, only at shallow depths of i.e. 500-600 m can we expect somewhat less difficulty in maintaining the excavation site. The value of n_{uw} index for site maintenance is then greater than 1.0. A larger depth can even prevent the maintenance of a mine working, or generate significant costs of establishment and maintenance.
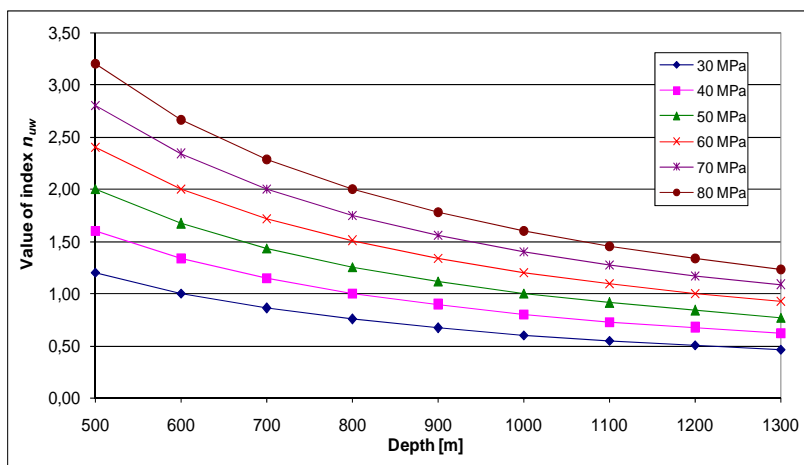

Fig. 1. Changes in excavation site maintenance n_{uw} index for $RQD = 50\%$

For a real evaluation of the state of an underground roadway, it is essential to take measurements in order to determine if it is stable (Xiu & Jiang, 2004; Thyrock & Dereli, 2011; Niedbalski et al., 2013; Álvarez-Fernández et al., 2013). It is generally accepted that the condition of stability is met when, during a specified period of time, changes in the longitudal and transverse dimensions of a working are lower than the acceptable critical values with regard to safety. In turn, the critical values depend on the geomechanical properties of the rocks surrounding a working. Hence the basic task that must be carried out before beginning mining work is identifying the geological-engineering conditions under which the excavation site will be run. Knowledge of the mechanical properties of coal and rocks surrounding the coal seam is used when preliminary evaluating the stability of a roadway in order to determine the size of possible cave-ins, displacement of the site floor and rock mass, as well as where potential fractures might occur (Toraño et al., 2002; Majcherczyk et al., 2008; Thyrock & Dereli, 2011).

3. Design of a steel arch with rockbolt support system

The design of mixed support structures using analytical methods is usually done in one of two distinct ways, each based on different assumptions. The first way is based on static calculations of the limited load capacity of yielding steel support, and is used mainly for steel frame support. Adapting it to the calculations of frame support with bolted-in roof beams involves treating rock bolts as rods fixed to the system or consideration of the reduction of weight coming from cracked rocks. The second method is based on L.A. Panek's (1956) solution on strengthening rock mass using rock bolts. We can understand the term "gain factor" as a reduction in the fractured rock load for excavation site support. In many countries indicators of rock mass quality such as *RMR*, *GSI*, *Q* and *RMi* are used in support system design for roadways; however, in Poland they have not yet seen wide use. Similarly, the use of response curves and the surrounding rock mass has not yet become widespread (Brady & Brown, 2004). This has partially to do with the global trend of using numerical methods instead (Jing, 2003; Palmstrom & Stille, 2007; Prusek & Bock, 2008; Feng & Hudson, 2011).

For steel arch with rockbolt support system design, we can use the methodology based on PN-90/B-03200, and specify the load for steel frame support with limited load capacity using the formula:

$$q_{gr} = \frac{q_1 \cdot f_d}{\dfrac{N_{1odp}}{\varphi \cdot F} + \dfrac{M_{1max}}{\varphi_l \cdot W_x}} \tag{2}$$

where:

q_{gr} — load for steel arch support with limited load capacity, kN/m,
q_1 — unit value for support system load, $q_1 = 1$ kN/m, for which N_{1odp} i M_{1max}, kN/m is determined,
φ — buckling coefficient, -,
φ_l — lateral-torsional buckling coefficient, -,
f_d — computational steel strength, kN/m^2,
— transverse surface area for a section of the steel arch support, m^2.

In this case, rock bolts are treated as rods fixed to the support system. The support strength of the system frame with respect to application of rock bolts is determined by the relationship:

$$q_k = \frac{q_1 \cdot N_k}{N_{1k}} \tag{3}$$

where:

q_k — support strength of the system considering rock bolt strength, kN/m
N_k — rock bolt load capacity e.g. for one pair $N_k = 240$ kN,
N_{1k} — value of the force acting on the rock bolt for a unit load q_1, kN,

and the system frame spacing d from formula:

$$d = \frac{\min(q_k, q_{gr})}{q_{obc}} \tag{4}$$

where q_{obc} — system load, kN/m^2.

According to other authors, a reduced force will act on the steel arch support after the application of rock bolts (Daniłowicz & Skrzyński, 2003). This also depends on the strength of the rock mass and the density of the rock bolting. Computational load q_{oz} will total:

$$q_{oz} = q_o \cdot K \cdot P \cdot m_c \qquad (5)$$

where:

     q_o — system load, MPa,

     K — coefficient of the rock mass attenuation (depending on the depth, divisibility, and slakeability) -,

     P — gain index (load reduction of the steel arch support as a result of bolting between frames), -,

     m_c — coefficient of rock pressure modification (depending on geological-mining conditions including but not limited to: seam edges, geological fault lines, etc.), -.

Rock mass gain index P is defined as the degree of unit load decrease of a steel arch support as a result of bolting between frames. To determine coefficient P, the relationship between rock compression resistance and bolting density is given, according to which the value can change from 0 to 0.9.

Research conducted by Piechota and Korzeniowski (2002) shows that the space between a steel arch support enhanced by rock bolts can be determined from the formula:

$$d = \frac{q_0 \cdot W_{wg}}{q_{zo}} \qquad (6)$$

where:

     d — frame span of the steel arch with rockbolt support system, m,

     q_0 — computational support strength of the system, MN/m,

     W_{wg} — rock mass gain index, -,

     q_{z0} — computational load of the support system, MPa.

The authors determined rock mass gain W_{wg} index on the basis of mine research, defined as:

$$W_{wg} = \frac{q}{q_k} \qquad \text{and} \qquad W_{wg} = \frac{e}{e_k} \qquad (7)$$

where:

     q — load of the system frame without rock bolts MN,

     q_k — load of the system frame with rock bolts, MN,

     e — roof stratification without rock bolts, MN,

     e_k — roof stratification with rock bolts, MN.

The authors recommend using the following values of rock mass gain W_{wg} index for a rectangular frame support system – $W_{wg} = 2.4$; but for an arch frame support system – $W_{wg} = 1.7$. If the roof is enforced by cable bolts, increasing W_{wg} to the next level is suggested: for a rectangular frame support system – $W_{wg} = 3.0$; and for an arch frames – $W_{wg} = 2.0$.

The next calculation proposition for a steel arch with rockbolt support system results from observations and measurements of mines, on the basis of which it was found that the load carrying

capacity of mixed support systems depends on the load capacity of the rock bolts (Niedbalski, 2014). Moreover, the load capacity of developed rock bolts has an additional effect on the compressive strength of roof rocks.

Assuming therefore, that the unit load of the excavation site roof is known, the spacing of the steel arch with rockbolt support system can be determined from the following empirical relationship:

$$d = \frac{\sqrt{(N_{LP})^2 + (a_w N_{Kot})^2}}{q_{s\,max}}$$

(8)

where:

d — frame spacing of the steel arch with rockbolt support system,
N_{LP} — working load capacity for 1 m of working span MN/m,
N_{Kot} — working load capacity of applied rock bolts per 1 m of frame width MN/m,
$q_{s\,max}$ — maximum roof load from fractured rocks and other weights MPa,
a_w — gain index dependant on the laboratory resistance to uniaxial compression of direct roof layers, $a_w = 1{,}0 \div 2{,}0$.

However the best way of checking the designed support system effectiveness is to carry out underground measurements (Xiu, Jiang, 2004, Prusek, Lubosik, 2007, Majcherczyk et al., 2011, Thyrock & Dereli, 2011, Álvarez-Fernández et al., 2013, Niedbalski et al., 2013).

4. Research results for steel arch with rockbolt support systems

4.1. Reinforcing by rock bolts with steel beams

Analyzed here is a drift excavation site, using yielding steel support type ŁP9/V29 (height 3.5 m, span 5.0 m) with a spacing of 0.75 m. The support has been reinforced by two rows of V25 shape steel beams, bolted to the roof using strand bolts with a total length 4.0 m (Fig. 2).
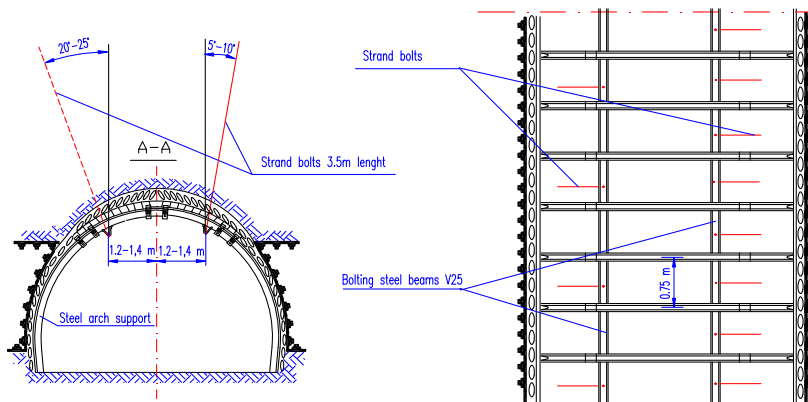

Fig. 2. Steel arch with rockbolt support system – reinforced by steel beams

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The stability of the excavation site was monitored throughout a period of around 18 months. Figure 3 shows the results of measured displacements of roof strata, which were carried out until the position of measurement was located around 160 m from the front wall.

The analysis shows that the use of bolted-in beams initially resulted in the dislocation of the roof towards the excavation area, after which it was then subjected to compression (Fig. 3). Only just above the mine working contour (around 0.4 m) did steady roof stratification ensue (moving into the excavation site). The maximum displacement values came to around ±40 mm. In terms of roadway stability, roof displacement values of 2-4 cm do not pose a problem for maintaining stability of the working. It is also worth noting that the longwall face, located 160 m from where the measurements were taken, did not generate any changes in the rock mass surrounding the research section of the working.
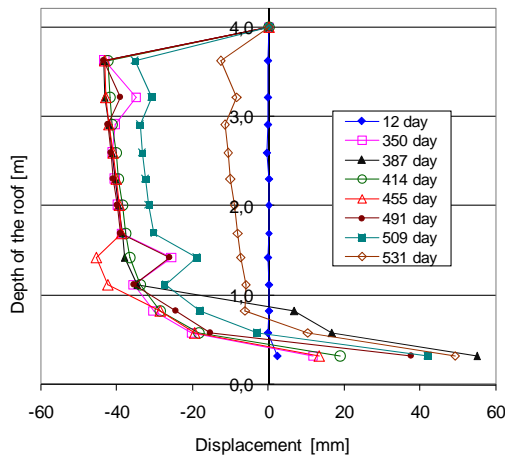

Fig. 3. Movement in roof strata of the analyzed roadway

The application of 4-meter long strand rock bolts and the presence of a sandstone roof caused small changes in the dimensions of the excavation site (Fig. 4 and 5). Before being subjected to excavation, the height of the drift decreased by about 70 mm, and the width by around 50 mm. After the disclosure of excavation impact and shift of the front wall, the height of the excavation site – depending on the position – decreased from 1100 mm to 1260 mm (Fig. 4), and the width from 300 mm to 710 mm (Fig. 5). It is also important to notice the fact, that the width of the excavation site located before the front wall at datum stands 986 and 966 m increased only slightly as the height decreased.

In order to determine the forces acting on the rock bolt support system, this research has been conducted using instrumented bolts (Fig. 6). Subject to stratification, the roof has also been reinforced by rock bolts, which can be fairly burdened by compressive forces –200 kN (initial section of the rock bolt) as well as tensile forces 160 kN (at the 0.49 m point of the rock bolt). The compressive force on the initial section of the rock bolt can be the cause of rock layers pressing against the support frame. This analyzed scheme for a steel arch with rockbolt support system thus successfully ensures the stability of a roadway, and is a testament to the active role of rock bolts.
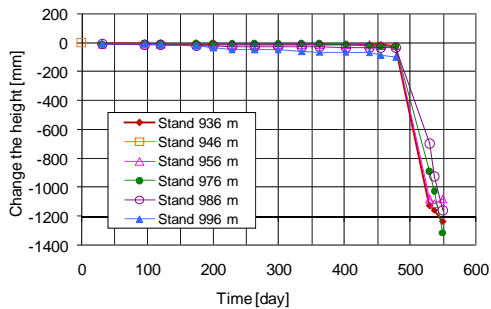
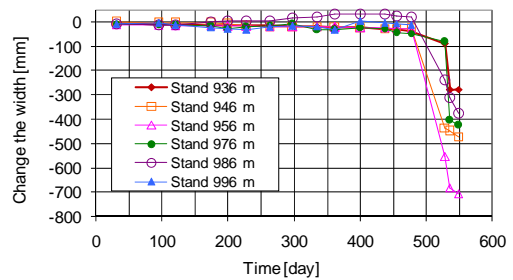Fig. 4. Changes in roadway height

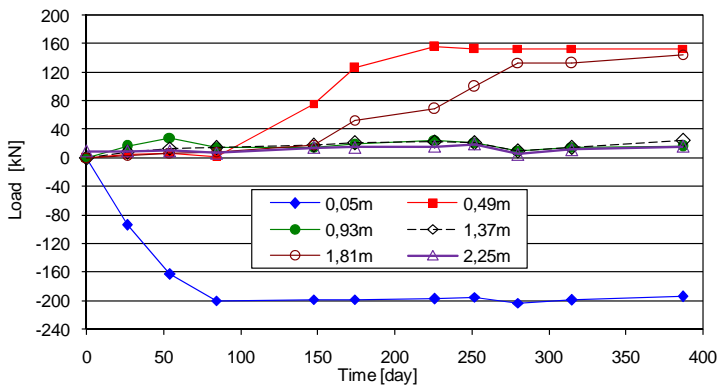

Fig. 5. Changes in roadway width

Fig. 6. Load supported by a measuring bolt in a drift site

From a depth of 800 m for the roadway, as well as from a resistance to the compressive strength of roof rocks (R_c = 71 MPa), it can be concluded that n_{uw} roadway maintenance index for RQD=11% totals 0.39, which would indicate significant difficulties in preserving its stability. Nevertheless, as proven by the research results, stability of the roadway remains fully intact before the effects of mining operations. Despite significant convergence after transition of the excavation wall front, the excavation site could then be used for the next longwall operation. If we take into account the results of endoscopic roof observation, which indicate that after the longwall face passed only two cracks were found there – i.e. RQD = 100%, the value of n_{uw} working maintenance index amounts to 3.55. This value fully reflects the predisposition of the rock mass to maintain stability, specifically when using long strand bolts.

4.2. Reinforcing support with bolts between arch frames

In the second of the analyzed excavation sites, two support schemes were subject to analysis. The first section featured yielding steel arch support type ŁP9/V29/4 with a frame spacing of 1.2 m, and likewise in the second section, with a frame spacing of 1.5 m. In both cases the roof of every second support arch frame was reinforced with strand rock bolts, 5 m in length (Fig. 7).
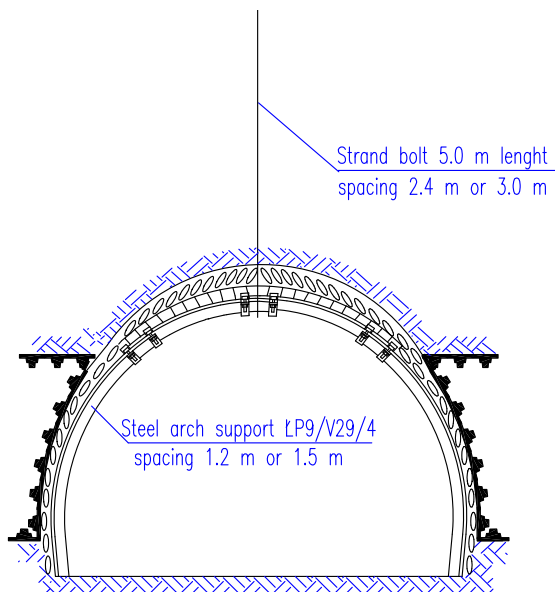
Fig. 7. Steel arch frame with rockbolt support system scheme – bolting between frames

The results of the load capacity research conducted on arch frames show that in both ana-lyzed sections, the load supported by the frame during the period of measurement was negligible and after about 10 months was about 30-35 kN maximum (Fig. 8 and 9). At the same time, we can also see an unevenness in the load supported by the frame, which is indicated by different values of forces acting between the sidewall arches and the floor. The resulting load values of the frame, despite an increased spacing of 1.5 m, are significantly lower than the maximum load capacity which the frame can carry.
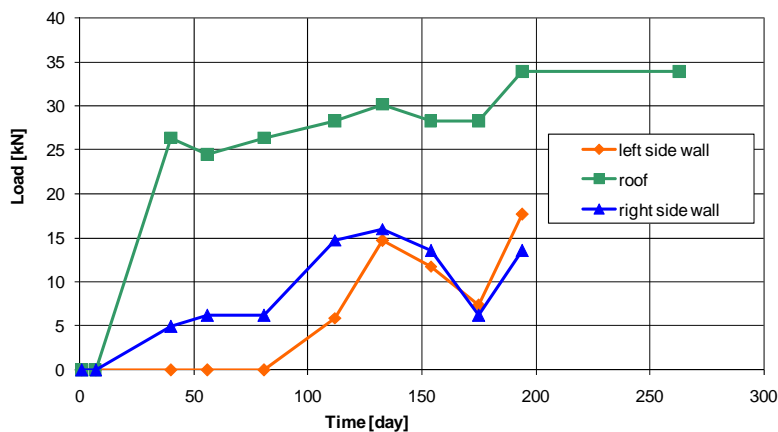

Fig. 8. Load frame support for the 1.2 m spacing between section of the support system
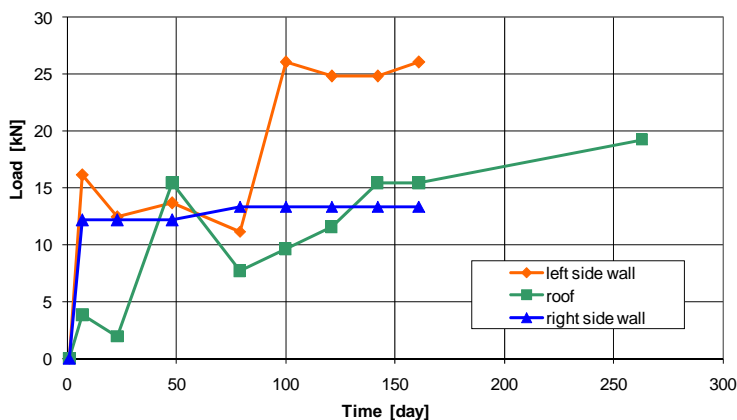
Fig. 9. Load frame support for the 1.5 m spacing between section of the support system

Small load values of the support frame were also confirmed by measurements of convergence, which were carried out over a period of about 3 years. The decrease in width, except for in one position of measurement, did not exceed 65 mm (Fig. 10). In the case of height, the final changes were somewhat larger and, depending on the datum, ranged from 0 to -220 mm (Fig. 11). At the same time, we can notice a small impact of the frame spacing on the vertical convergence of the roadway. At datum stands 4/2 and 5/2, where the support frame spacing amounted to 1.2 m, the change in the working height was lower than in datum stands 7/2, 8/2 and 9/2, where the frame spacing amounted to 1,5 m.

The values presented from the above research done on support systems reveal that the application of steel arch with rockbolt support, even with a frame spacing of up to 1.5 m, allows for safe maintenance of excavation site stability. The displacements in roof strata of the roadway
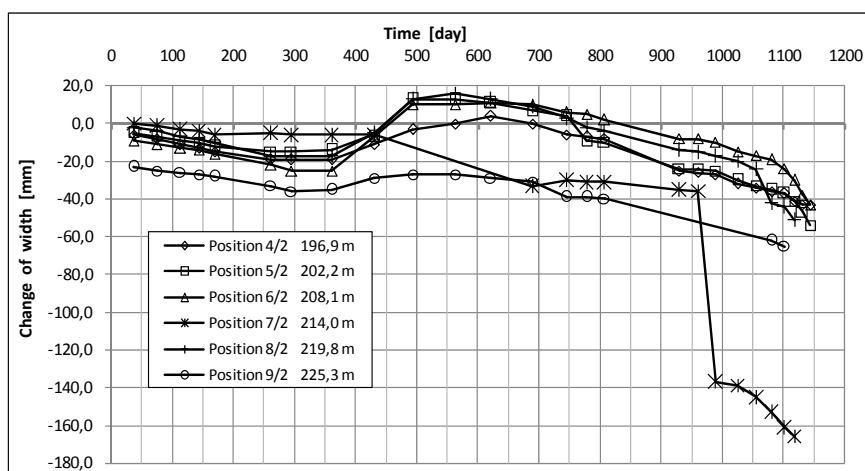

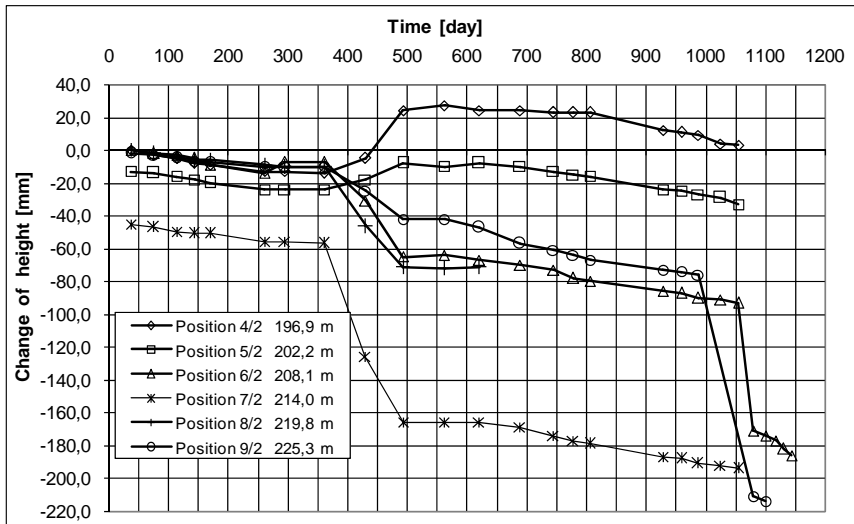Fig. 10. Change in roadway width in the researched section

Fig. 11. Change in roadway height in the researched section

were also small, as the endoscopic research conducted indicated that the range of the rock damage zone was a maximum of 0.3 meters, with a stratification of a few millimeters.

Considering roadway stability in the analysis, with a depth totaling 760 m, and a roof rocks compressive strength (R_c = 63 MPa), it was found that working maintenance index n_{uw} for RQD = 52% amounts to 1.71. This shows that the conditions conducive to the maintenance of an roadway exist in the area around it. The results of 3 years of monitoring the convergence of the working confirm its stability beyond a reasonable doubt. Comparison of the support system described in this work to standard and other support systems for the analyzed roadway can be found in Majcherczyk's book (Majcherczyk et al., 2008).

5. Summary

The following conclusions can be drawn on the basis of the performed analysis:
1. The stability of roadways in coal mines is of paramount importance. One of the ways of improving stability of the excavation site is the use of a steel arch with rockbolt support system. The use of such a support system is carried out mainly by: frame bolting using rebars; frame strengthening using bolted steel beams; and bolting between frames. In recent years, this has mainly been done using strand bolts up to 12 m in length.
2. Analytical, empirical, and numerical methods are used for designing steel arch with rockbolt support systems. Analytical methods are based on general principles and norms used for designing steel constructions. Empirical methods are most frequently based on the results of mine measurements, and usually account for reduction of the load supported by the frame, as well as the degree to which rock mass is strengthened using rock bolts.
3. The loss of roadway stability is most commonly a result of changes which, after the construction of the site, occur in the surrounding rock mass. In order to maintain the

correct dimensions of the excavation site, it is absolutely necessary to use support that is adapted to the corresponding geological and mining conditions. A preliminary assessment of the suitability of rock mass, indicating the difficulties of working maintenance under given conditions, can be specified on the bases of working maintenance index n_{uw}.

4. The results of mine research done on steel arch with rockbolt support systems show that strengthening the support frame with bolted steel beams maintains the stability of the working, even when exposed to the pressures of exploitation. Aside from the impact of exploitation, deformations in the support system are negligible, despite the fact that the tensile forces acting on the rock bolts can reach values of up to 160 kN.

5. Under favorable geological and mining conditions, support frames can be spread up to 1.5 m apart when using rock bolts between them. Simultaneously, the level of safety and stability of an roadway is upheld, since the measured load supported by the support system frame does not exceed 35 kN, and therefore is much lower than the maximum load limit. The measurements of convergence carried out over a three year period showed only small changes, i.e. ranging only a few centimeters. The results yielded by the research fully confirm the effectiveness of steel arch with rockbolt support systems under different mining conditions – proof of which is their widespread use, especially in longwall gates.

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