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BOLTER MINER MACHINE FOR DRIVING ROADWAY WORKINGS - POLISH EXPERIENCE

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1. Introduction

Artur Dyczko[1]

Jastrzębska Spółka Węglowa S.A. (Jastrzębska Coal Company) is the biggest European producer of coking coal, entered on a list of critical raw materials, indispensable for a heat of steel and thus for a production of a key material for the majority of the European industry branches. Simultaneously, the JSW S.A. is a capital group subject to a transformation towards low-emissions economy, involved in research projects oriented onto offering innovative products of the technologies of the future.

Being driven by appropriately understood need of the Company development, in particular in the technical-and-operational area, over the years 2017-2020, the JSW Board decided to invest available financial means and resources for an assessment of possibilities of adapting best available technologies, used in the world mining industry, to increase an efficiency of a production cycle. To achieve this objective the programme called JSW 4.0 was established. Its main objective consisted in an elaboration, testing and implementation of the latest technologies in the Jastrzębie coal mines by the JSW Innowacje S.A.

The collaborative formula of the JSW and of the suppliers of the mentioned technology consisted in a technical dialogue oriented onto an identification of the best solutions in the systematic approach as regards both mining operations, as well as run-of-mine haulage and monitoring of rock mass. Only such a formula of the programme realization was flexible and transparent enough to enable reaching the project objective of implementing independent bolting support in the Budryk mine successfully.

Taking into consideration the standing and difficulty of implementing independent bolting technology in the Polish conditions, realized with use of the cutting-and-bolting machine, many indispensable changes in the structure of the Company were made to create a possibility of taking advantage of certain technological chances and predominances. The first step, enabling the technological dialogue, was a scientific conference from the International Mining Forum 2017 cycle (realized in the framework of the Underground School of Mining) by the JSW S.A., to which the following participants were invited:

– representatives of leading suppliers of the state-of-the-art technological systems for mining,

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– scientists from national and international research institutes,
– managers from the leading companies producing minerals in Poland.

A development of mining due to a diversification of its activity based on knowledge, innovations, research projects and a collaboration between science and industry were discussed during the conference.

During a dozen or so sessions of the International Mining Forum 2017 a series of important relationships on the line: industry-science-suppliers of technology, which in the following quarters of the year were transformed in real projects, were established.

The first important project, realized in the framework of the accepted programme, was an implementation of the technology of driving workings with use of independent bolting technology, realized with use of cutting-and-bolting machine, commonly applied in the world.

This Monograph presents the experiences of the whole research teams, gained during a three-year period of the project duration, documenting in detail the biggest technological break-through in the Polish mining industry, which has not been achieved for years. It consists of 10 chapters. An idea of realizing the project of independent roof bolting support in the JSW S.A. mines is presented in the Introduction. In the second chapter an origin of the Bolter Miner machine and its use in the world mining industry in the process of driving roadway workings of rectangular cross-section, are discussed. In the third chapter of the Monograph a history of using bolting support in the national mining and in the world is described. This chapter is a certain introduction to the subject of bolting with use of independent bolting support. The fourth chapter contains a description of the project itself as regards implementing independent bolting support in roadway workings of the Jastrzębska Spółka Węglowa S.A. and an analysis of a full spectrum of possibilities as regards using this technology in the process of driving opening and development workings and also an extraction of residual parts of the deposit. Based on the analysis the venue of the project was chosen – the BW-1n test roadway in the Budryk mine. A detailed description of mining-and geological conditions and of natural hazards, occurring in the area of the driven working, is given, including an assessment of the following conditions, in particular:

– roof, floor, hydrogeological,
– tectonic faults,
– natural hazards.

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The sixth chapter contains a detailed description of the accepted technology of driving the Bw-1n test roadway with use of the Bolter Miner machine, and the seventh chapter is a description of the project of the elaborated bolting support for the BW-1n roadway with a discussion and monitoring. The eighth chapter presents a variant analysis of driving bends during the roadway drivage by the Bolter Miner machine, together with an assessment of alternative possibilities of driving crossings in the existing system of equipment collaborating with the machine. The ninth chapter is a deepened analysis of advances and an assessment of efficiency of conducted mining operations with a particular emphasis on an analysis of using the operational time of the whole technological system. The Monograph is ended with a synthetic summary and a rich literature of the subject.

Taking a decision about a realization of the implementation of independent bolting support, realized with use of the Bolter Miner cutting-and-bolting machine, the Jastrzębie miners proved that independent bolting support is not a problem for the Polish mining industry, but a challenge!

2. Bolter Miner Machines in the Practice of the World Mining Industry

Jacek Korski[1], Marek Majcher[1]

2.1. Introduction

An origin of the machines of Bolter Miner type results from a multi-year development of longwall systems in the hard coal mining. The beginning of longwall systems in the British hard coal mines dates back to the XVII century, but until the end of the XIX century it was a traditional mining – all the operations connected with an exploitation and transport of coal were performed manually. An application of conveyors (shaking, belt and in the end the scraper ones) in longwalls, at manual cutting and loading of the run-of-mine enabled to increase the output due to an elongation of the longwall and an employment of a bigger number of miners in it. Manually operated mechanical devices in a form of percussive hammers, shoveling machines and drilling machines were also used. Already in the XIX century some trials of applying coal cutters for undercutting coal in longwalls were undertaken. However, later on, in the end of the thirties of the XX century the first ideas of cutting-and-loading machines in coal longwalls, in a form of cutting out machines, appeared. An appearance of milling or drilling drum machines, operating on a longwall scraper conveyor (so called armoured one) at the beginning of the fifties of the XX century enabled to make a big progress. The scraper conveyor, introduced into operation in the forties of the XX century enabled to perform a few exploitational cycles within 24 hours [1]. When an operation of installing supports in the longwall became fully mechanized, in the sixties of the XX century, then fully mechanized longwall systems appeared. A new problem also appeared. It was connected with so called maintaining the mining capability of a mine, carrying out an exploitation with use of a mechanized longwall system – heading workings for the following longwall should be prepared before an operating longwall face reaches the ending line. An increase of the daily advance, resulting from a mechanization development, forced an increase of daily drivage advances and an introduction of a mechanization of these operations. It was a genesis of applying roadheaders with point cutting heads (roadheaders with axial or transverse cutting heads). In the European and Soviet coal basins chock supports, made of different materials (metal, timber), were commonly used in the gateroads. The shape of the working transverse cross-section was also different. Meanwhile, in the developing underground coal mining industry in the USA different types of room-and-pillar systems were commonly used. Since the forties of the XX century roof bolting instead of chock supports was more and more commonly used. Development and

[1] FAMUR SA

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exploitational workings had a rectangular cross-section. In the same time different producers in the USA developed and implemented narrow-face cutting-and-loading machines for room-and-pillar mining systems. This group of different machines was called Continuous Miners. The Joy JCM1 machines, were among them. They were functionally similar to roadheaders used in Europe – their cutting head was an extension arm with a multiple, parallel milling and cutting chain. The cutting head could move in the vertical and horizontal planes. Nowadays in the underground coal industry narrow-face machines of the Continuous Miner type with horizontal, so called linear cutting head, being a development of machines with milling multiple cutting chain, are used in mechanized room-and-pillar systems.

When in the seventies of the XX century some trials were undertaken to apply mechanized longwall systems in the coal mining industry of the USA, identical to those used in Europe, it turned out that there was a series of problems limiting the productivity of longwalls in better conditions than the European mining-and-geological conditions [2]. In the conditions of using mechanized longwall systems in Europe, the chock roadway supports limited a possibility of applying longwall powered roof supports only between the internal ribs of these roadways (from the rib side). It caused an occurrence 5of additional conflicts, activities and operations at the contact between the longwall and the roadway. A common use of the bolting support of roadways in the coal mining industry of the USA, at an application of the European solutions of longwall systems, forced a necessity of using an additional support in a form of steel props which hindered the work and increased a labour consumption of operations at the crossing: longwall – roadway (Fig. 2.1).

A rectangular cross-section of the roadway, driven in coal, enabled to implement a revolutionary change – the longwall powered roof support was installed as far as the external ribs of gateroads, which enabled to eliminate manual operations at the contact longwall - gateroad.

It led to an increase of a daily longwall production rate and simultaneously it increased the requirements in terms of advances achieved in driving galleries. Initially they were driven with use of the Continuous Miner machines with manual roof bolting. Then in the systems of two or more parallel roadways the Continuous Miner machines and mobile roof bolters were used. Similarly to the mechanized room-and-pillar system, they exchanged in the face, by turns performing operations of cutting the rock mass (together with loading the run-of-mine and its haulage) and of roof bolting.

Fig. 2.1. Tailgate with a scraper conveyor and additional
enforcement with steel props in the longwall area [4]

During the passes the Continuous Miner did not cut which was a loss, as its possibilities were not used.

Dynamically developing underground coal mining industry in Australia started its operational activity with room-and-pillar mining systems. However, possibilities of increasing an extent of the deposit use, due to mining the left coal pillars (retreat longwall mining) [5], were searched. The most advanced system, according to Australian views, turned out to be the retreat longwall mining system. For driving rectangular galleries in bolting support, in the conditions of nearly horizontal coal seams, the machines with point and linear cutting heads were used.

An application of very efficient longwall systems, similar to those used in the USA, in very advantageous mining-and-geological conditions caused that the problems with a regeneration of the mining front - advances of coal roadways were too small [6], appeared very quickly. A search for solutions, whose main objective was a maximization of the cutting time at high productivity, led to a generation of the first machine of the Bolter Miner type [3] and its implementation in the Australian Tahmoor mine in 1991.

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2.2. Basic reasons for constructing Bolter Miner machines

Bolter Miner machines are a development derivative of mechanized longwall systems for mining hard coal with use of rectangular galleries driven in roof bolting technology. The basic objective of creating such machines was to gain technical systems, which due to highly productive cutting-and-loading system, would be capable of cutting the volume of one web depth – step of the working drivage till the moment when a protection of the roof and ribs of the working with use of bolting, becomes indispensable. An achievement of big drivage advances also required a big share of time for cutting (Fig. 2.2).

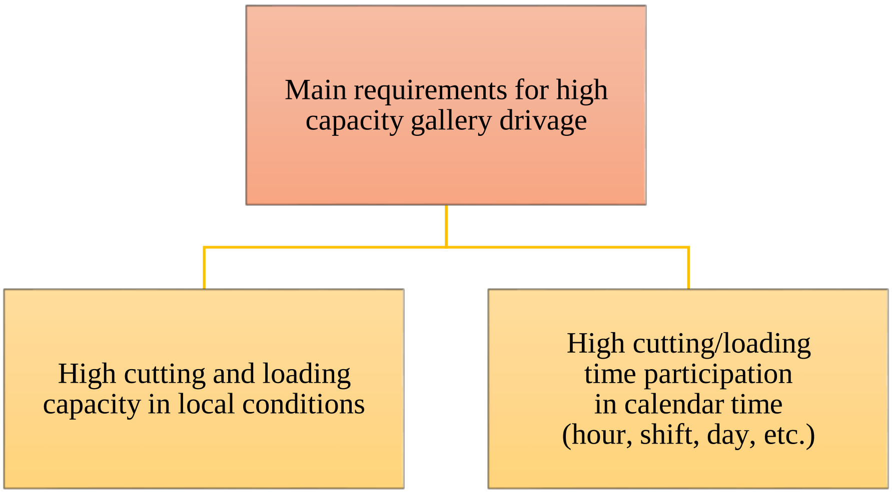

Fig. 2.2. Basic requirements for achieving a big advance of the gallery under driving

A productivity of mechanical cutting depends on the type of mined rocks and of their cuttability, including the uniaxial compressive strength and the structure of rocks for which the commonly used parameter is the RQD index [7].

In the majority of cases, for a given cutting machine the cutting productivity (m^3/min(h) or Mg/min) decreases as the compressive strength and/or the RQD index decrease. Therefore, for the majority of cutting machines permissible and optimal cutting parameters are determined.

For the needs of an analysis and for an improvement of the mechanical process of driving galleries in the Australian coal mining industry the structure of the face calendar time was elaborated [8]. It is shown in Fig. 2.3.

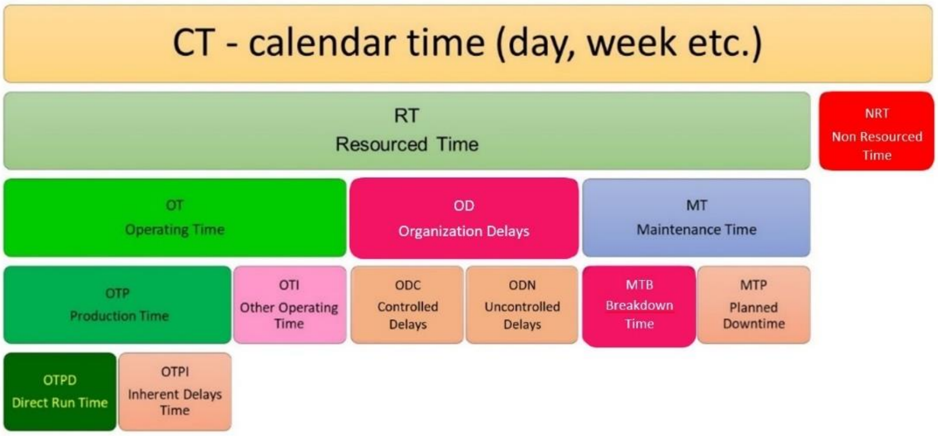

Fig. 2.3. Structure of the calendar time of roadway working acc. to Australian views [9]

This structure is used for searching solutions increasing an efficiency of driving galleries.

During a drivage of a gallery, the time spent on cutting, is most essential because the superior objective of development operations consists in making transport routes indispensable for coal mining in the accepted exploitation system. Only cutting and removing the run-of-mine from the face (understood as loading of the run-of-mine and its transport beyond the face) is an operation which serves directly "a generation" of a new working. In the case of the faces, driven mechanically, each activity or operation which requires stopping of cutting, decreases a share of time planned for cutting in the balance of the available time (hours, shifts, days etc.) Unjustified stoppage of operations, connected with a mechanical drivage of a gallery due to lack of the face personnel or unjustified operational breaks and failures, also reduces the time which should be spent on cutting [9].

The superior idea of improving the working drivage process is a maximization of the OTPD time share, i.e. in the case of mechanical cutting (an operation of cutting and removing the run-of-mine from the face). In the case of the working driven in the bolting support, the time of bolting, when a stoppage of cutting is indispensable, is also an unwelcome delay (a break of cutting), so the OTPI. The most welcome solution is an execution of cutting and bolting operations parallelly or if it is not possible, shortening, to minimum, of the bolting time due to an application of more efficient bolters and due to an increase of their number working simultaneously. For improving the drivage efficiency, it is indispensable to eliminate or to minimum reduce other activities, operations and breaks, which limit the time dedicated to cutting. Such an approach led to an appearance of the machines of the Bolter Miner type.

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A general idea consisted in a creation of the machine, which would integrate two devices on one chassis:

– a cutting-and-loading machine enabling the run-of-mine haulage beyond the face (the machine),
– multi-head bolting station with the heads enabling simultaneous, efficient bolting of the working roof and ribs. Nowadays the number of heads varies from 4 to 6.

In the topology of face cutting machines, the machines of Bolter Miner type are among the machines which integrate on one chassis all the activities connected with the roadway drivage [9].

However, before the Bolter Miner machines appeared some trials of creating the machines, based on a similar idea, had been undertaken.

In the eighties of the XX century the Joy Company developed the machine called Joy Sump Shearer (JSS) (Fig. 2.4) [11] to satisfy the needs of dynamically developing Australian coal mining industry.

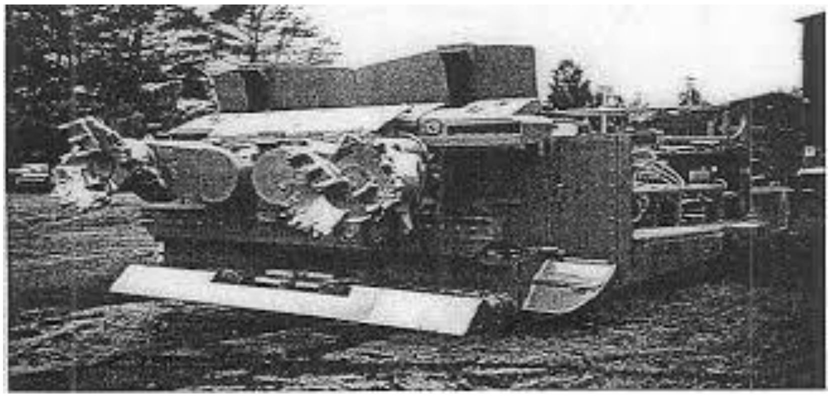

Fig. 2.4. Joy Sump Shearer [11]

The machine was designed for driving workings of rectangular-arch cross-section with roof bolting in the direct vicinity of the face front (1.5 m). Two bolters were used and the working cross-section was selected due to an optimal use of the bolters. The machine was equipped with two cutting heads installed on the arms, similarly to longwall shearers.

These heads enabled cutting the roof (with cutting stone) of an arch shape. It minimized the amount of stone cut in the roof because at that time coal longwalls of the height 2.2-2.8 m were mined in the Australian mining industry. A thickness increase of mined seams and limited benefits, resulting from use of the JSS, caused that this solution was relinquished.

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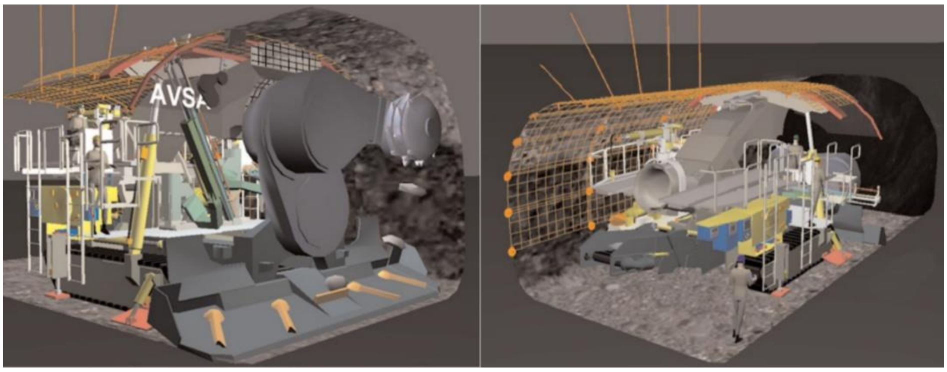

Fig. 2.5. Machine of AVSA type for driving galleries [12]

In Europe some trials were made with the AVSA system [13], shown in Fig. 2.5. Similarly to the JSS a milling cutting head was used for cutting, but it was installed on an articulated arm enabling cutting out of a half round face cross-section (Fig. 2.5) [14]. Also in this case the superior idea included parallel cutting of the rock mass and bolting of the roof and ribs directly behind the machine cutting head.

The working drivage systems of the AVSA type, developed by the Voest Alpine Bergtechnik, adapted for an application of both kinds of roadway supports (bolting and chock supports) were used in German hard coal mines (among others in the Schachtanlage Rossenray in 2003 or the West Mine). They were promoted on a large scale but they were not dissiminated.

2.3. Technical system of roadway drivage with use of Bolter Miner

Irrespective of the face cutting method and of the applied supports, each technical system of its drivage must ensure a possibly efficient execution of basic operations connected with the working drivage (Fig. 2.6).

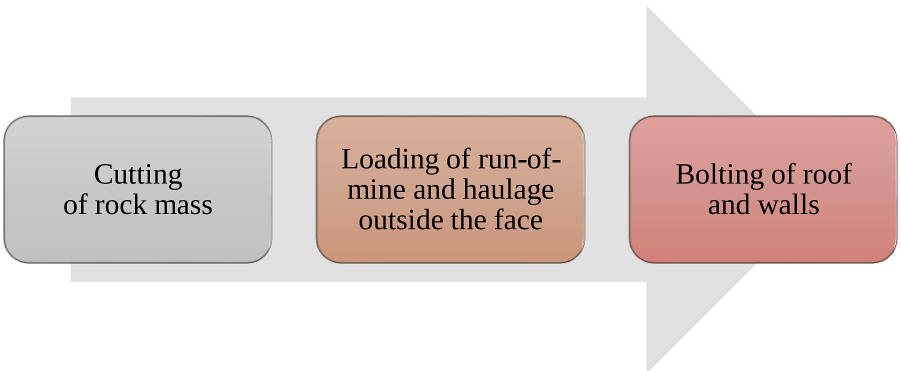

Fig. 2.6. Basic operations connected with a roadway drivage with use of Bolter Miner

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An efficient execution of these operations in the existing conditions requires:

– Haulage or transport system of the run-of-mine adapted for the characteristic of the cutting machine.

– Efficient media supply systems (electric energy, water etc.) of parameters ensuring a correct operation of Bolter Miner.

– Efficient system of transporting materials indispensable for a realization of the operations connected with the working drivage and with the technical infrastructure development.

A prerequisite also includes an availability of workers having indispensable skills and an availability of their sufficient number. The presence of personnel in the process of working drivage generates a necessity of ensuring the conditions for their living and working, including an application of indispensable system and solution (e.g. ventilation).

At present there is a necessity of protecting the working under driving, in fact protecting its functionality due to an application of chock supports and/or bolting (as in the drawing above).

An application of the first machines of the Bolter Miner type in the Australian coal mines, which combined a high productivity of coal mass cutting as the machines of Continuous Miner type as cutting machines in the Room & Pillar system with a possibility of protecting the working roof and walls after each cutting step, established high requirements concerning other technical subsystems collaborating with this machine. In Fig. 2.7 the most common system of the roadway drivage with use of Bolter Miner in Australian conditions, is shown.

An efficient application of Bolter Miner machines giving high advance rates, requires a very efficient logistic support. Although an application of roof bolting fundamentally reduces the amount of material needed during the drivage, a big face advance requires other support systems (haulage of the run-of-mine, supply of media and ventilation) to follow-up. That is why Bolter Miner machines operate very well in multi-headings systems (including double ones) connected with crosscuts at relatively small distances from one another. A maximum simplification of the technical system of faces, driven alternately with one (Fig. 2.7) or simultaneously with two cutting machines, is a target.

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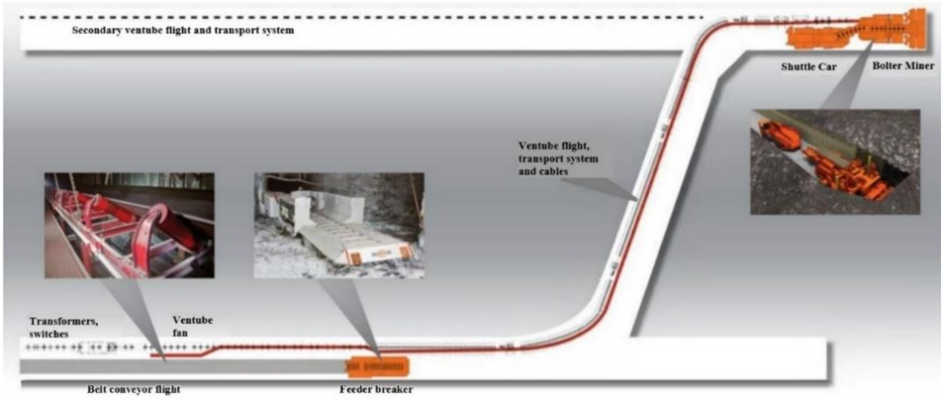

Fig. 2.7. Basic elements of heading face technical system with Bolter Miner machine [8]

In the case of faces driven with one machine, it is indispensable to ensure a free passage way from the face to the face. Due to big, in relation to the working transverse overall dimensions, transverse dimensions of the Bolter Miner machine, the passage way must be free and it should not have any elements impeding this passage. A high cost of the machine also requires an efficient displacement to a new place of operation and the shortest possible time of being out-of-productive operations. That is why the system of multiple workings, connected with technical crosscuts, enables to occupy only one of these workings with stationary system of the run-of-mine haulage and with the face supply. The ventilation flights (ventubes), which are installed in the sections of the blind drift, are also shorter.

2.3.1. Systems of run-of-mine haulage in faces driven with Bolter Miner machine

In the Australian coal mining industry, where there were technological solutions connected with use of narrow-face cutting-and-loading machines of Continuous Miner type, in the case of the faces driven with Bolter Miner the solutions of the run-of-mine, used in the Room & Pillar exploitation system, were duplicated.

The basic mechanization system of coal roadway drivage with use of the Bolter Miner is shown in Fig. 2.8. For an elimination of down-time in the case of longer distances from the Feeder-Breaker station sometimes two Shuttle Cars were used.

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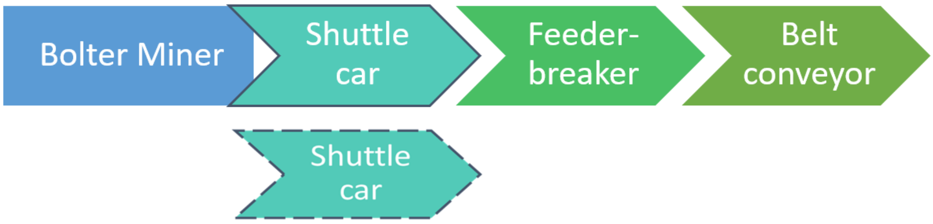

Fig. 2.8. Basic equipment of the face mechanization system with use of Bolter Miner

The Suttle Car was used for collecting the run-of-mine from the cutting machine and for delivering it to the discharge area, to the Feeder Breaker station (Fig. 2.9.).

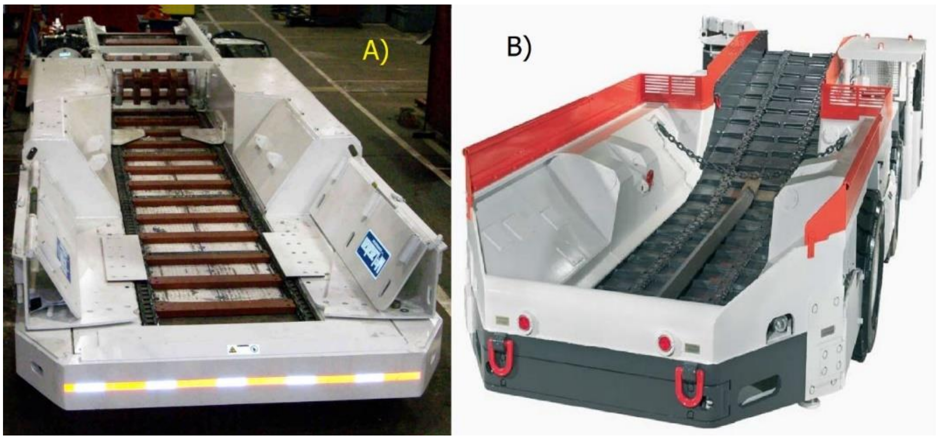

Fig. 2.9. Equipment of the run-of-mine haulage system from the face driven with use of the Bolter Miner: A) discharge – crushing station of Feeder Breaker type, B) vehicle of Shuttle Car type [13]

An advantage of such a solution consists in a possibility of a flexible connection of the Bolter Miner with the system of distant transport with use of belt haulage – the machine has a big, instantaneous efficiency of feeding the run-of-mine, interrupted by the periods when (during bolting) the run-of-mine is not fed. A storage volume of the Shuttle Car and the characteristic of the Feeder Breaker station enables to spread the run-of-mine stream over time and to reduce the requirements and loading of the conveyor. Additionally, the run-of-mine is crushed to adapt it for a transport on a belt conveyor (an elimination of oversized lumps of the run-of-mine).

A high, but short-lasting efficiency of cutting and loading the run-of-mine by Bolter Miner at a direct loading the run-of-mine to the collecting conveyor (belt or scraper) requires similarly high, instantaneous parameters of this conveyor and of the whole haulage flight. It requires big expenses for such a haulage system (adapted to instantaneous, and not average production) rates as it causes a risk of overloads of the run-of-mine haulage system (over-charging). Such a situation was confirmed in the case of applying the 12CM30 Komat'su machine in Poland, where a continuous run-of-mine system was used exclusively (Fig. 2.10).

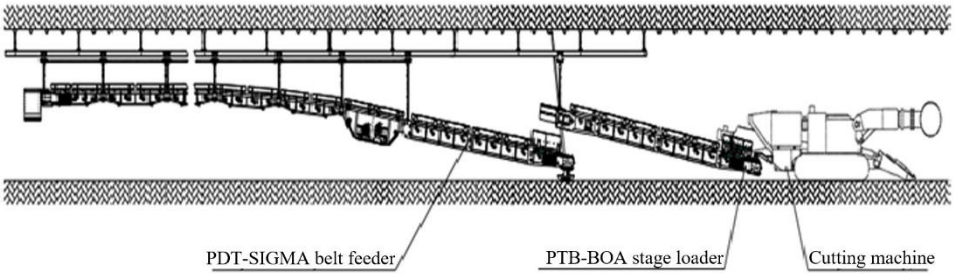

Fig. 2.10. The run-of-mine haulage system from the face driven with the 12CM30 Komatsu Bolter Miner machine in one of the Polish mines with use of suspended belt conveyors

Bolter Miner machines, characterized by a big bare weight require a positioning activity before starting cutting and bolting – the conveyor suspended to the machine impedes an execution of maneuvers and causes a damage to the working floor. Due to this a suspension of additional equipment to the Bolter Miner machine is avoided.

A solution of the run-of-mine transport with a Shuttle Car (or several ones) and a Feeder Breaker station is the most common solution in the world but it should be indicated that in the Chinese mining industry a solution with a belt haulage directly behind the machine was used. An internal feeder of the Bolter Miner machine with a movable, in the horizontal plane, discharge of "swing" type, spilled the run-of-mine to a voluminous, wide box of the Feeder Breaker without a mechanical connection with it.

2.3.2. Systems of face supply with materials and energy

An application of independent bolting support reduces significantly, in relation to the faces conducted with use of the steel chock supports, the requirements concerning a transport of materials to the face – it is one of important advantages of using bolting technology. In the world practice mainly tyre-mounted front loaders (LHD), including Scoop ones, are used (Fig. 2.11).

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Fig. 2.11. Front loaders (Scoop, LHD) as an auxiliary transport equipment

In the mines of Russia and China, and recently in the Polish Budryk mine a diesel monorail was used for a transport of materials and personnel to the face driven with use of Bolter Miner machine.

2.3.3. Ventilation systems of faces driven with use of Bolter Miner machines

In the practice of driving individual roadway workings of different type using roadheaders, a problem of applying separate ventilation (ventubes) occurs, because big lengths of the headings cause many problems related to maintaining proper ventilation conditions and a removal of auxiliary ventilation accesories later on, after having finished the working drivage. Such a solution was applied in the Budryk mine.

The systems of multiple headings connected with crosscuts (as in Fig. 2.7) limit, for example the lengths of the workings ventilated separately and thus an efficiency of the face ventilation. Auxiliary fans (ventube type) on wheeled chassis (Fig. 2.12) or slide base, which can be quickly displaced between the crosscuts, are used.

Fig. 2.12. Auxiliary fan on wheeled chassis [35]

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After having finished the drivage, there are no long sections of left ventubes and other materials impeding a removal of the machine from a long individual face.

2.4. Experience of the world coal mining industry in using Bolter Miner machines

Since an implementation of Bolter Miner machines for an exploitation in the Australian underground coal mining industry at the beginning of the nineties of the XX century, they have been applied in many advanced countries in the world as regards the mining industry. Already around 2012 over 300 Bolter Miner machines, of different producers, were operated in the mining industries all over the world. They often achieved very big daily and shift advances. At the end of nineties of the XX century Bolter Miner machines were used in the coal mining industry of the USA. A few years later these machines were applied in the coal mining industry in Russia and China. Single machines were operated in the coal industry of Great Britain, Germany, the Czech Republic, and recently also in Poland.

Below, some examples of efficient application of Bolter Miner machines in the countries, where they were operated on a bigger scale, are given.

2.4.1. Bolter Miner in Australia

The ABM20 Bolter Miner machines made by the Voest Alpine Bergbautechnik Company [16] were used as the first ones in Australia. Later, the machines of other producers were also introduced and in 2014 about 50 Bolter Miner machines were operated parallelly in Australian mines. Positive first experience [17] caused that for a further improvement of results, in the following years, the machines themselves, also whole mechanization systems and technology [18] as well as work organization [19, 20] were improved.

The basic mechanization system of coal roadway drivage with use of the Bolter Miner machine is shown in Fig. 2.7.

The mined coal is collected by one shuttle car, but in the case the faces of very big daily advance an application of the second unit may turn out to be necessary. A universal transport vehicle for transporting materials or front loader (LHD or scoop), systems of auxiliary ventilation and power supply unit (set of transformers and electric switches) belong to the additional face equipment.

A continuous improvement of coal or coal-and-stone roadways drivage encompasses also an improvement of the technical system devices and of work organization. There is also a search for solutions, which improve the bolting process, enabling to extend the time to be spent on cutting [20, 21, 22, 23]. Since the beginning, the activities in the field of technical system development of

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the face, driven with use of the Bolter Miner machine, have been concentrated on:

- a remote supervision of the machine operation (of the cutting system),
- a mechanized and then an automatic system of installing the mesh lining and of bolting the roof and ribs,
- ensuring a reliable, continuous run-of-mine haulage from the face, which is the best solution,
- an integration of operational systems and of the face (faces) supply systems, with longwall supply and servicing of the longwall under preparation.

All the undertakings were concentrated on reducing the duration time of individual activities and operations in the face, a generation of a possibility of a parallel performance of cutting/loading operations and of bolting as well as an elimination of cutting breaks caused by operational breaks for an execution of activities excluding the cutting process and thus stops caused by organizational breaks and unserviceabilities.

The implemented technical solutions enabled to obtain very big advances of gateroads drivage in some cases e.g.:

- hourly – up to over 7 rm/h [23],
- shift advance (in Australian conditions – 11 h) – to 55 rm/shift.

Within the framework of testing drivage processes of coal roadways in the independent roof bolting systems a big relationship between the achieved advances and the number and type of bolts, installed during one cycle of drivage (one web depth), was stated. During an improvement of the bolting process self-drilling, inserted bolts were introduced (which enabled to save time for an exchange of the drilling rod for bolts and vice versa).

In more advantageous roof conditions some tests on increasing the cutting web depth were conducted and then a few rows of bolts were installed one after another (e.g. 3 web depths of 1.0 m and then 3 rows of bolts). For many years some attempts of applying continuous haulage systems from the face and rapid, following systems of extending belt conveyors when the face advances, have been undertaken.

It should be noticed that in the Australian conditions two types of cutting-and-bolting machines can be distinguished:

- cutting machines equipped with an assembly of bolters enabling only alternate cutting/loading and an installation of mesh lining/roof and ribs – these machines are called Miner-Bolters,

– machines equipped with an advancing cutting-and-bolting system (Sump) and an assembly of bolters installed on the machine chassis – such a construction gives a possibility of parallel cutting as well as supporting the roof and walls. In the Australian mining industry these machines are called Bolter Miners.

In good mining-and-geological conditions the hourly advances of faces, irrespective of the type of the cutting-and-bolting machine, vary from 2.5 to 7.1 rm/h.

In the Australian research projects and tests the significance of the following activities is highlighted:

– designing and planning of the drivage process with use of cutting-and-bolting machine,
– a correct selection of the technical system devices according to the project expectations,
– a selection of the number and qualifications of the staff indispensable for achieving the expected results.

2.4.2. An application of Bolter Miner machines in Russian coal mines

Before Bolter Miner machines were used in the Russian Federation, Continuous Miner machines had been applied in the room-and-pillar mining system (in 2009 in 125 underground coal mines only 76 longwall systems were operated). The bolting support was commonly used.

Since 2006 an implementation of Bolter Miner machines has been started in Russian coal mines. The first machines were used in the Kuzbas Coal Basin. An implementation of the 12CM30D Bolter Miner of the Joy Global (at present Komat'su) in favourable mining-and-geological conditions of the Vorgasorskaya mine [24], with correctly selected mechanization system, enabled to obtain very big advances (in August 2009 more than 1000 meters of the monthly face advance were achieved. In the following month 1212 rm of coal roadway were driven in this mine, using another machine of this type. The width of the roadways was 5.2 m and their height was the same as the seam thickness: 3.6 m. Steel bolts in the roof were used with the bolting spacing 0.5 m and steel mesh lining. In the ribs the bolts of the length 1.8 m in the "W" configuration also with the steel wire mesh lining were used. Additionally, behind the Bolter Miner machine 5-metre rope bolts were installed. Shuttle Cars, similarly to Australia, operated together with the Bolter Miner machine. Two Bolter Miner machines, applied in this mine, obtained an average daily advance of 35 rm/d. According to available information the roadway faces, driven with use of the Bolter Miner, achieved not fewer than 300 rm of the monthly advance, at the average of about 600 rm/month.

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Since 2013 the MB-670-1 machines, made by the Sandvik Company, have also been used [25].

2.4.3. Bolter Miner in China

Around 2010 Bolter Miner machines found an application in Chinese hard coal mines, where technology and an organization of driving processes of galleries are being improved continuously. Although the average results (daily advances), in very differentiated mining-and-geological conditions, vary in the range 15-25 rm/d [26], much better results are also achieved. Bolter Miner machines, produced by the leading world manufacturers (Komat'su/Joy Global, Sandvik), are used. As opposed to the coal mining industry in the USA and Australia also long, single, linear coal workings [27, 28] are driven with use of Bolter Miner machines. Technical systems of driving galleries depend on driving conditions and expected results. Such an approach approach is shown in Table 2.1.

Mechanization systems in roadway faces with mechanical cutting and bolting support according to the Chinese approach [29]

Table 2.1.

Devices incorporated in the system	Conditions of using the mechanization system
Roadheader with a point cutting head, individual bolter, suspended conveyor (bridge-belt or scraper), stationary belt conveyor	Recommended for an application in stone or stone-and-coal faces, at the cuttability of rocks up to R_C=100 MPa (Protodiakonov f<10) and in the situation when there is no need of an immediate installation of the working support. The system can be applied in a big scope of mining-and-geological conditions.
Roadheader with the installed integral bolter, suspended conveyor, stationary belt conveyor.	Applied in stone and in stone-and-coal roadways, at the cuttability of rocks up to R_C=100 MPa (Protodiakonov f<10). In relation to the former solution it reduces the bolting time and improves work safety.
Roadway machine with a linear head (Continuous Miner), self-propelled multi-head bolter, Shuttle Car, front loader (scoop or shovel), reloading-and- crushing station (Feeder-Breaker).	System convenient for driving coal and coal-and-stone workings of big cross-sections in the system of double parallel workings in favourable mining-and-geological conditions. Alternate cutting and bolting in the

	face. Big cutting capacity. A small adaptation ability to changeable mining-and-geological conditions.
Bolter Miner, suspended conveyor, stationary belt conveyor.	System for driving single coal and coal-and-stone roadways in good mining-and-geological conditions. It is possible to carry out cutting and bolting simultaneously. A big cutting capability. A small adaptation ability to changeable mining-and-geological conditions.
Rapid driving system with use of the Bolter Miner machine and a follow-up system of collaborating equipment (so called BMRET – Bolter Miner Rapid Excavation Technology).	System applied in very good mining-and-geological conditions of the Ordos Basin in China. The system is integrated in terms of obtaining big advances: cutting-bolting and run-of-mine haulage, ensuring operational continuity.

Two last groups of solutions with use of Bolter Miner machines, as original case-studies, are presented below.

2.4.3.1. Bolter Miner of the Komat'su (Joy Global) in the Chinese mines

In the Chinese mines Bolter Miner machines are used in very differentiated mining-and-geological conditions, so it is difficult to compare the results obtained by the machines of different producers in the case of applying very differentiated systems of the face mechanization. In Table 2.2 examples of two faces, driven with use of the Bolter Miner machines, are shown.

Exploitational conditions and results obtained by the 12CM30 machines

Table 2.2.

Mine:	No. 1	No. 2
Type of machine	12CM30 Bolter Miner	12CM30 Bolter Miner
Seam thickness (m)	9.0	4.7
Kind of floor	aleurolite	aleurolite
Rocks in roof and ribs	weak	---
Working dimensions (height x width - m)	4.2 x 5.4	4.0 x 5.6
Longitudinal inclination (deg)	up to 15	horizontal
Layout of bolts	– 6 x 2.4 m bolts in the roof	– 6 x 2.2 m - roof bolts

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	– 4 x 1.8 m rib bolts (for each wall, altogether 8 pcs.) – 2 x 9 m - rope bolts installed outside the face – Spacing of bolts: 0.95 m, later 1.3 m	– 2 x 2.0 m – wall bolts (for each wall, altogether 4 pcs.) – 2 x 5.0 m - rope bolts installed every 3.6 m – Spacing of bolts: 1.0 m
Face coverage - shifts	4 shifts/day, (6 days/week)	4 shifts/day, (7 days/week)
Average daily advance (m/d)	20 rm/d	36 rm/d
Top advance in a month	600 rm	1000 rm
Average time of cycle (min)	49 min	29 min
Machine availability	About 90%	-

Lack of more detailed information renders it impossible to assess the obtained results, in particular in the context of mining-and-geological conditions and the depth of conducted operations.

2.4.3.2. Application of the Bolter Miner machines made by the Sandvik Company in the mines of the Shenhua Shendong Coal Group

In 2012 the state-owned group Shenhua Shendong Coal Group purchased a dozen or so of the MB-670 Bolter Miner machines made by the Sandvik Company. Since the very beginning some activities have been undertaken to implement highly productive roadway systems enabling to obtain big advances. The basic field of the mining activity in the scope of underground hard coal excavation is the Ordos Coal Basin with coal deposits situated on the border of the Autonomous Region of Inner Mongolia and the Shanxi Province in China. Some of the mines have coal deposits of very regular deposition of strata and often of big thicknesses. For a few years so called ultra-high longwalls have been mined in this Basin, recently of the longwall opening height of 8.5 m. For such longwalls, of very big panel length, systems of double gateroads of rectangular cross-section are driven. Bolter Miner machines are used for the drivage since the beginning of implementing these machines of big technical potential-cutting capacity, a search for solutions, enabling a full use of their possibilities, has been carried out. In 2013 in the Daliuta mine, belonging to the Shenhua Shendong Coal Group Co. Ltd., roadway faces with the MB670 Bolter Miner machines [29] were started. The mine is relatively shallow (less than 200 m), having nearly horizontal, regularly deposited coal seams and favourable mining-and-geological conditions in a form of good roofs and load-bearing floors without any tectonic dislocations. In such conditions an exploitation of so called ultra-high longwalls (over 7 m) of very big daily production rates is carried out [30]. In the existing conditions, due to an application of appropriate mechanization systems and mining technologies, it was possible to obtain very big daily advances of driving gateroads of very big panel length. In technical-and-technological solutions the main emphasis was laid on a maximal increase of the time spent on cutting the rock body during the daily available time. The superior idea of the driving process realization and the applied technical system of drivage was a harmonization of the three operations triad:

– Cutting, loading and haulage of the run-of-mine beyond the face,
– Installation of support (roof and ribs bolting in the working),
– Transport of the run-of-mine and materials,

in the way, enabling to conduct these operations in parallel to avoid holding up or slowing down by one operation of the other two.

The MB670 machine, made by the Sandvik Company of very big cutting capacity (up to 25 Mg/min) in coal of R_C = about 25 MPa and the technical system (in particular of the haulage), which realized a continuous, rapid machine advance without any breaks in cutting for e.g. an extension of the haulage system or a dislocation of collaborating devices, enabled a realization of such a process. A big emphasis was laid on such a technology of bolt support installation which would not hold up the cutting operation with the machine even to the smallest extent, because only the cutting operation ensured, as the matter of fact, an advance of the working. One of the solutions connected with bolting was a separation of the bolting operations into two phases:

- Phase I: Initial protection of the face with the bolters of the Bolter Miner machine, performed during the cutting operation.
- Phase II: Execution of supplementary bolting to achieve the final form in a small distance from the face front.

For an execution of the supplementary (final) bolting a mobile bolting station, equipped with 8 bolting heads, installed “astraddle” over the path of the mobile belt conveyor in a small distance from the face, was applied. An installation of the initial roof support (e.g. bolting), temporarily protecting the face and then supplementing it with the support finally protecting the working for the whole time of its life, was carried out. This solution is an often used method, but the local conditions may limit possibilities of its use (Fig. 2.13).

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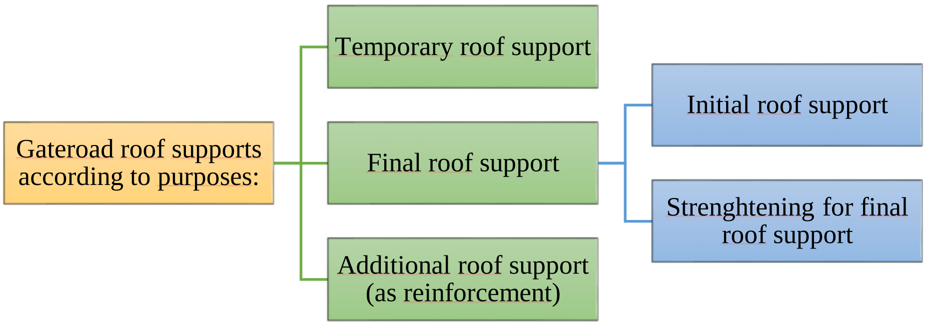

Fig. 2.13. Systematics of gallery supports due to their application and installation time

The machine fed the mined coal to a mobile feeder-breaker, equipped with a coal small retention bunker. The run-of-mine from the feeder-breaker was fed to a mobile, follow-up belt conveyor, which advanced together with the feeder-breaker, following the cutting machine. The follow-up belt conveyor, fed, in turn, the run-of-mine to the stationary belt conveyor equipped with a slidable return station with its own systems of sprags, which enabled a rapid extension of this conveyor.

In the same group of mines the Bolter Miner machine was also used for driving a gateroad in different mining-and-geological conditions of the Meneqing coal mine situated in the north-eastern part of the Ordos Basin. The seam, in which the working was driven had the average thickness of 4.78 m [31] and it was deposited very regularly at a small inclination of about 3°. Moderately grained sandstone was deposited in the seam roof. Improving the driving systems, the activities were concentrated on an improvement of the process organization and of the driving technology. A big depth of conducting the mining operations caused bigger requirements as regards the support – bolting of roof and ribs, so in the improvement process this element was concentrated on. Among the applied solutions the following ones should be mentioned:

– An aim of replacing the activities and operations carried out in series in the face with the activities realized in parallel to the biggest possible extent;
– A reduction of execution time of individual activities and operations;
– Changes of type of bolts and of bolting technology.

New organizational and technological conditions were introduced step by step. In the first stage some activities, realized in series, were replaced by the activities realized in parallel e.g. roof bolting together with cutting. It gave an

 https://doi.org/10.32056/KOMAG/Monograph2021.2

effect in a form a reduction of the basic drivage cycle from 63.3 to 48.2 minutes at identical time periods of individual activities and at the same bolting technology. It should be highlighted that due to a significant depth of the working 7 steel bolts of the length 2.2 m were installed in the roof with the bolting network 0.8 x 1.0 m. Additionally rope bolts of the length 6.3 m in the bolting network 2.4 m x 3.2 m were used. The walls were protected by 5 bolts of 2.2 m for each wall in the bolting network 0.8 x 1.0 m. The face step (web) was 1.0 m.

In the following development stage the bolting design was changed by replacing steel rod bolts in the roof (7 pcs.) with rope bolts at initial tension (high tension anchoring) in the number of 4 pcs. of the length 4.3 m with a steel profile beam (canopy). Rope bolts were installed with the Bolter Miner bolters directly in the face. A number of wall bolts was reduced. A change of the bolting system gave a positive effect in a form of a significant reduction of the basic drivage time at the same web (1.0 m) and the same face cutting time (9.5 min). In the case of realizing the activities in the face in series, the duration of the basic cycle was 45.8 min (so it was shorter from the in series – parallel time of the basic cycle in the former bolting scheme!) and at the in-series-parallel execution of activities it was reduced to 22.7 min. The obtained results did not require any changes in the face mechanization system.

A further search of mechanization systems for driving coal roadways in exceptionally favourable mining-and-geological conditions required an increase of the cutting capability.

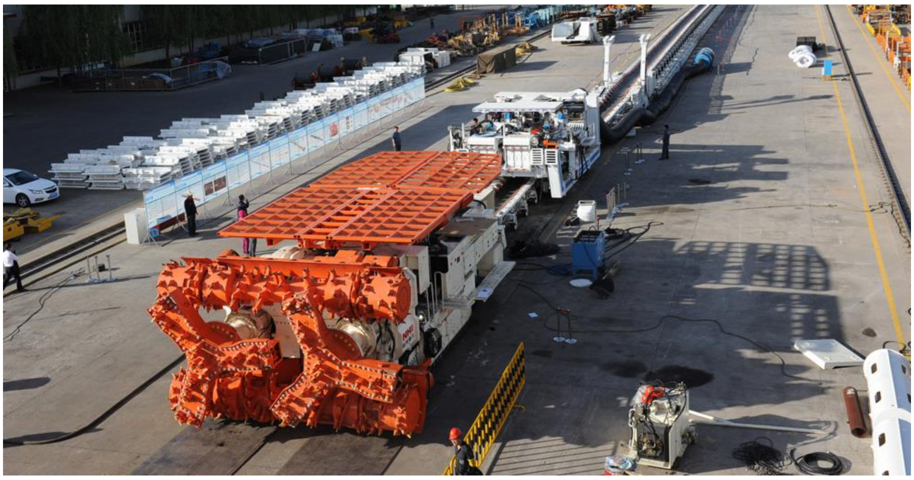

Fig. 2.14. Full-face QMJ4260 machine for rapid driving of coal roadways with mobile multi-head bolter in the Daliuta mine [32]

To achieve this objective the full-face, boring machine of the Borer Miner (Marietta Miner) with the system of temporary roof support on the machine, was applied (Fig. 2.14). Based on the analysis of images, it can be noticed that the

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QMJ4260 machine has the cutting system very similar to or even identical with the solutions of the MF420 machine made by the Sandvik Company. Due to a very big advance of the machine roof and rib bolting on a continuously advancing machine became impossible in practice. Due to that the possibilities of the independent bolting machine, following the cutting machine, were increased by equipping it with 10 bolting heads.

2.4.4. Bolter Miner in Europe

In the European coal mining industry the machines of the Bolter Miner type found an application in underground mining of Great Britain, Germany, the Czech Republic [33, 34] and also in Poland in 2019.

In the closed down in 2013 British Daw Mill mine the 12BM15 Bolter Miner machine, made by the Joy Company, was used. It is known that in difficult mining-and-geological conditions the machine achieved the daily advances on the level of 6-7 rm/d, in the case of the cycle duration time of 40 minutes, the web (step) of 0.8 m. The bolting network is not known, but for each cycle 48 m of holes for bolts were bored in total. Mining operations were conducted at the depth of 700-800 m.

In the British Riccall mine, closed down in October 2004, incorporated in the Selby mining complex, the Bolter Miner machine was also used for driving coal roadways in the Stanley Main Seam deposited at the depth of 750-850 m. It is known that the weekly advance of drivage reached up to125 rm.

It lacks reliable information about an application of the Bolter Miner machine in the German Walsum mine, apart from the information that the average daily advance was 8.3 rm/d. After having analyzed the results and experience, the Bolter Miner machine, made by the Joy Company, was used in the Auguste Victoria mine. A follow-up, slidable return station of the belt conveyor was used there, the supply and run-of-mine discharge systems were implemented and all the manual activities in the face were reduced to minimum. Due to the introduced improvements the daily advance was up to 22 rm/d.

Since 2014 the 12CM30 Bolter Miner, made by the Joy Company, as the basic machine in the room-and-pillar mining system of a coal seam in one of the Czech coal mines, has been used. The applied mining system was different, as regards many details, from typical solutions with a participation of the Continuous Miner machines, operated alternately with mobile multi-head bolting stations (Mobile Bolter) in a few faces (5-9) due to a possibility of executing initial bolting from the Bolter Miner. The applied system of workings (Fig. 2.15) and solutions of the technical system are similar to those used in the driving systems of the four parallel galleries in the case of longwalls operated in the mining industry of the USA.

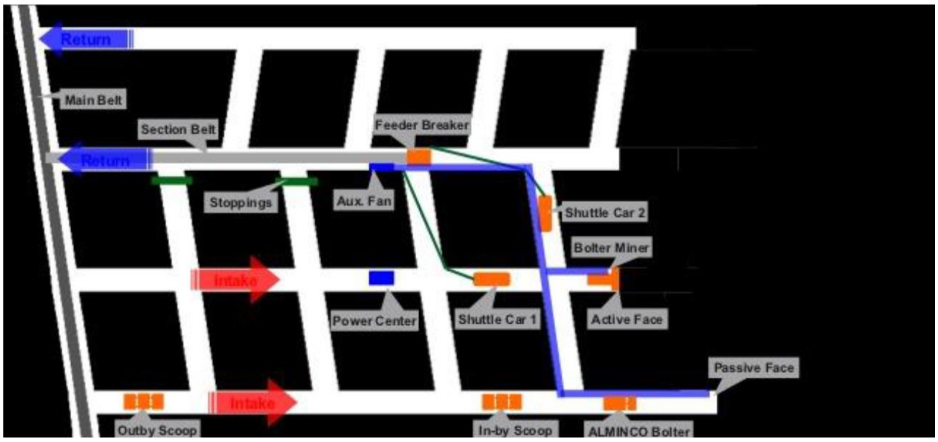

Fig. 2.15. Room-and-pillar mining system with use of the 12CM30 Bolter Miner machine used in the Czech CSM mine [33]

Two of the workings served for a supply of fresh air to the system of faces and two of them - for carrying away the used air. In the accepted mining system two of the faces always required an application of separate ventilation with a single air duct, with branching and a possibility of controlling the air propagation. An exhaust ventilation was used. In the face, where the Bolter Miner was operated, two ventilation variants were applied:

– stationary air duct with elongating end, enabling to keep a constant distance from the face front,
– stationary air duct connected with a flexible air pipe with an integral fan of the dust control station, being an integral part of the Bolter Miner.

The roadway crossings with the driven connecting crosscuts are made at the angle different from 90° due to big overall dimensions of the Bolter Miner and for facilitating an operation of the Shuttle Car. An occurrence of crosscuts between parallel roadways caused that the sections ventilated with separate ventilation (auxiliary) were relatively short. In the face, where the Bolter Miner machine was operated the amount of air was about twice as big as in the passive face (without cutting) with a mobile bolter in operation.

The face mechanization system, apart from the 12CM30 Bolter Miner, included additionally:

– 2 vehicles of 10SC32B Shuttle Car type,
– mobile Feeder Breaker of BF 14Q58 type,
– GE Fairchild 35C-WH2-30-AC scoop,
– mobile roof bolter – Alminco Scorpion Bolter,

– handheld bolter – Hilti,
– belt conveyor,
– CFT air duct fan with dust collector,
– adapted transformer – breaking assembly (installed in the fresh air current) – Hansen Electric,
– installations of electric energy supply, of technological water and compressed air.

The reason of applying such a mining system (room-and-pillar) consisted in creating a possibility of excavating high quality coal in the shaft pillar, where for a protection of numerous functional workings, it was not possible to use the longwall system. The system was introduced when it lacked experience in using such a mining method and such a technical system. The staff had to be trained from the grounds. In such conditions, within about a year of using this system, 61.8 thousand Mg of coal were mined and about 2200 m of exploitational workings were driven. The bolting system was adapted for changeable conditions-basic elements of the bolting support were installed with the Bolter Miner bolters, but also additional bolting, with use of handheld bolters, was used as well as supplementary bolting of ribs using the mobile roof bolter.

In November 2019 in the Polish hard coal Budryk mine a roadway face driven with the 12CM30 Bolter Miner machine was started. The initial objective consisted in checking a possibility of applying independent roof bolting support at a big depth in the conditions of this mine and getting experience in using the Bolter Miner for driving galleries in the Polish conditions.

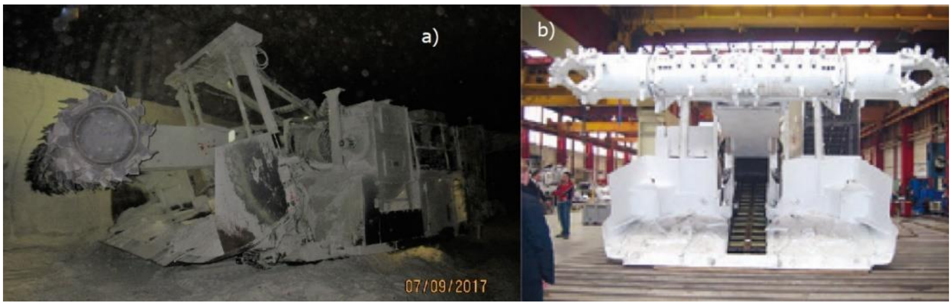

Fig. 2.16. MB670 Bolter Miner in salt mines: a) applied for an exploitation of rock-salt in the Polkowice-Sieroszowice cooper-ore mine belonging to the Polish KGHM, b) prepared for an operation in one of the German mines of potassic salts

The Bolter Miner machines, due to their big cutting capability were also used in the European mines of rock-salt and potassic salts (Fig. 2.16). In such applications the Bolter Miner machine does not have to be equipped with the bolting system.

2.5. Conclusions

The Bolter Miner is a cutting machine, enabling also a very efficient roof and rib bolting of the working in a direct vicinity of the face under driving. In advantageous conditions, operating together with harmonized haulage systems of the run-of-mine, the Bolter Miner can achieve the best daily advances among all the known systems of driving gateroad workings in hard coal mines.

Among the basic conditions, indispensable for getting good results of using the Bolter Miner, the following ones should be mentioned:

– Advantageous mining-and-geological conditions, in particular load-bearing floors, advantageous mechanical properties of rocks enabling an application of not-too-complex bolting support, low levels of the methane hazard as well as outbursts of rocks and gases.

– Well-designed mechanization system enabling a reliable, efficient haulage of the run-of-mine, reliable supply of media (electric energy, water, compressed air).

A good organization of work, operations and appropriate staff, which in connection with big production capacity of the Bolter Miner, enables to use maximally the time available for the face cutting.

3. Bolting technology in the world

Wojciech Masny[1], Jerzy Ficek[2,3]

The first notices, concerning an application of bolts in the mining industry, according to the review presented in the publication [35], are dated for the years 1872 – New Wales [36] and 1905 – the USA [37]. However, a real development of the bolting technology started in 1913 when Stephan, Fröhlich and Klüpfel obtained the patent No. 302909 according to which the main objective of applying a new type of support was a replacement of traditional wooden props, installed on the floor, used in those times. To reach this objective: "*It is necessary to drill holes of appropriate length in the rock, into which rods, pipes or ropes, made of the material ensuring the appropriate load-bearing capacity e.g. steel, will be inserted and which will be fixed at the end in the appropriate manner or cemented along their full length"* [38].

The history of the Polish coal mining industry is inseparabely bound with the beginning of using bolts as in 1916 bolting support was used for the first time in the Pokój mine [38-40], where due to an interaction of floor rockes, the working side walls were damaged. The length of applied bolts was 2.0 m, their spacing – 1.0 m and their connection with each other was made with a steel flat bar (Fig. 3.1).

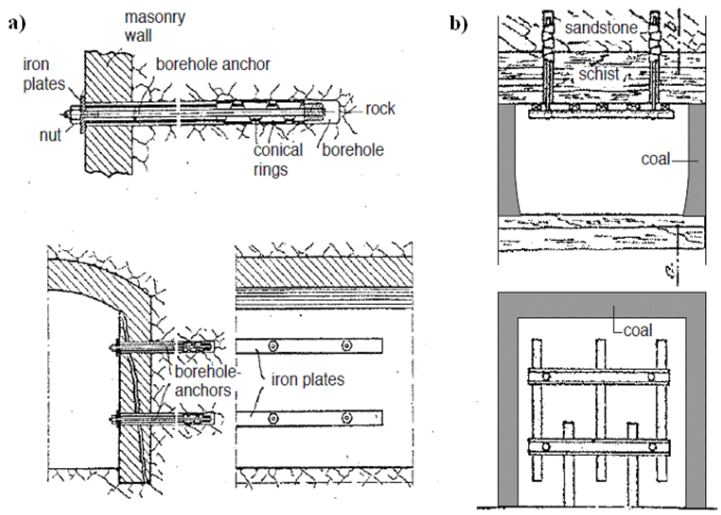

Fig. 3.1. The first application of the bolting support:
a) in the working walls, b) in the working roof [38]

[1] Central Mining Institute
[2] GEOFIC Technical and Scientific Office
[3] Enterprise of Shafts Construction JSC

The trials of roof bolting (Fig. 3.1b) were undertaken also at the same mine. An application of this type support system was too innovatory as for that time and the following report on using the bolting support in the Polish mining industry comes only from the year 1938, from the Michał mine, where bolts, fixed with cement, were used for protecting the side walls of the main rock drift situated at the depth of 340 m [40].

In turn the following essential events, which were connected with an application of bolts in the Polish coal mining industry, took place in the post-war period. In 1954, under the auspices of the Central Mining Institute, the first roof bolting, based on scientific principles [40] was conducted. In 1960 an operational manual of executing and checking up the bolting technology, adapted to operational needs [41], was edited and in 1976 the Ministry of Mining edited the "Temporary guidelines of using Bolting, Bolting-and-Chock and Simple Chock supports in Hard Coal Mines" (Polish "Tymczasowe wytyczne stosowania obudowy kotwiowej, kotwiowo-podporowej i prostej podporowej w kopalniach węgla kamiennego"), which were in power until the beginning of the nineties.

At the turn of sixties and seventies the bolting support was used as the basic support in the mines of Legnicko-Głogowski Copper Basin. In the same period numerous successful attempts of roof bolting in the hard coal mines (Siemianowice, Bielszowice, Katowice, Dębieńsko, Michał, Czeladź and Milowice) were undertaken. Nevertheless they were not finished with an implementation of roof bolting technology on an industrial scale. Among the reasons of such a situation a low work efficiency, caused mainly by a use of outworn bolting equipment, should be mentioned [42-44].

In the nineties of the last century an application of independent bolting support was started in a wider scope in Poland, including the roadways. Two roof falls of the workings happened in 1994 and 1995 (02.11.1994 – Anna mine, 5.07.1995 – Staszic mine), which caused an interest decrease in this type of support until 1999. A renewed interest in roof bolting, over the years 2000-2002, was connected with restructuring changes. However, since that moment the length of driven workings with use of this type support, has decreased systematically until a total forbearance of its use as independent support in the year 2009 [45-46]. This situation is presented in Fig. 3.2.

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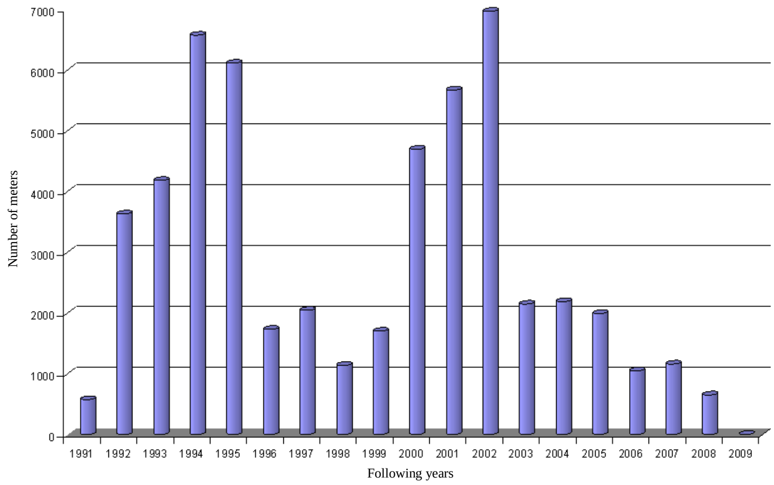

Fig. 3.2. Length of workings driven in independent roof bolting system in the years 1991÷2009 [45-46]

Only within the period 1991-2000 roof bolting, in different forms, was used by 44 underground mining plants. However, in the independent roof bolting 300 km of workings were driven. A possibility of an extensive use of mining bolts was conditioned by [43]:

– amendments to legal regulations concerning issues of supporting underground workings,
– a dynamic market development of bolts, bolters and glue charges,
– trainings of engineering personnel in the scope of bolting of workings,
– a selection of qualified experts on issues of roof bolting by the President of the State Mining Authority (WUG – Wyższy Urząd Górniczy),
– a realization of indispensable research work in the scope of cooperation of bolts with the rock mass and an application of bolts for a protection against the effects of dynamic phenomena,
– an organization of scientific-and-technical conferences concerning the bolting of workings subject-matter.

The experience in using roof bolting at the beginning of XXI century was positive explicitly, in particular in the case of roof bolting in the start-up crosscuts of longwalls and roadways drawn off behind the longwall front. In the roadways, maintained behind the longwall front, no problems with the roof control were observed. However, a phenomenon of the floor upheaval turned out to be a problem. The basic advantages, resulting from using the independent roof bolting support, obtained in the period under discussion, include [43]:

- a reduction of materials' use,
- a reduction of time for a reinforcement of start-up crosscuts of longwalls,
- an increase of work safety,
- an improvement of produced coal quality due to a reduction of the rock cutting in the roof.

A possibility of an essential costs reduction of workings drivage is also significant, because for each mine in the total economic account not only longwall workings but also roadway workings, whose drivage and reinforcement in the appropriate support, have a big share in the total production costs.

Based on different sources it can be stated that applications of independent roof bolting support, instead of a typical chock support, caused a reduction of the drivage costs [47-49]:

- Jankowice mine, Tailgate Z-1, Seam 413/1+2 23.9%,
- Jankowice mine, Raise Z-1, Seam 413/1+2 28.5%,
- Jankowice mine, Haulage Roadway, Seam 411/1 24.8 %,
- Jankowice mine, Gravity Incline Z-2, Seam 411/1 24.3 %,
- Chwałowice mine, Tailgate, Longwall I, Seam 403/1 19.5 %,
- Mysłowice mine, Incline 3 western, Seam 418 20.5 %.

It should also be noticed that bolting equipment, till the time of starting the working drivage in the independent roof bolting technology in the Budryk mine, where for the first time in the history of the Polish coal mining industry the Bolter Miner machine was used for this purpose, was based mainly on hand-operated bolters (e.g. UW-1, BM-8, Gopher, BM-505, Wombat, KHTP [49-50]), which had an undoubtful impact on the achieved production results

An independent bolting support, closest to the Polish borders, was used in the Czech part of the Upper Silesian Coal Basin, where over the years 1996-1997 a trial excavation of the safety pillar of the Northern Plant shafts was conducted. The mining method was based on gateroads and for cutting and bolting the ABM 20 machine, produced by the Voest Alpine [34], was used. No sooner than 17 years later in the year 2014, the idea of mining the coal seam, deposited in the mentioned safety pillar, returned. However, this time it was decided to use

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the room-and-pillar method, modified for the needs of Czech conditions. A part of the mining panel is presented in Fig. 3.3.

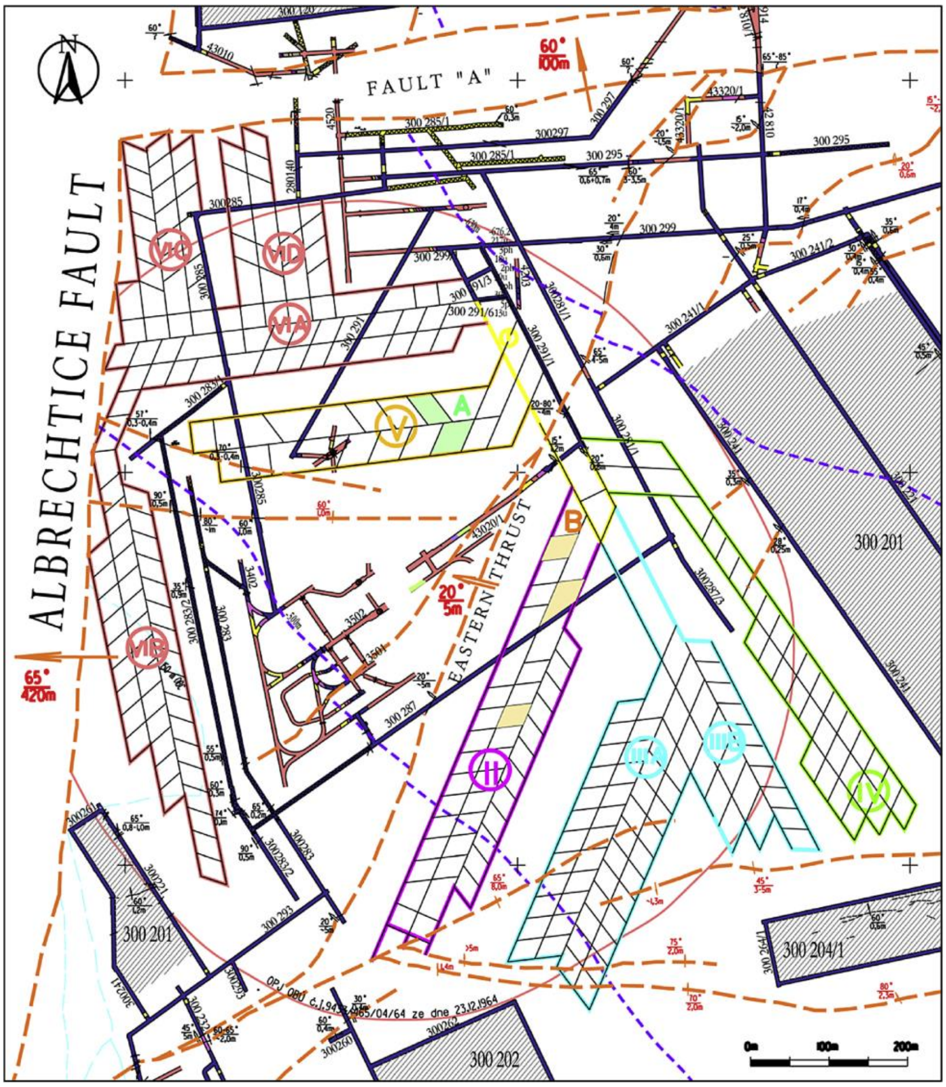

Fig. 3.3. A part of the panel mined with use of the room-and-pillar method [51]

The basic geological-and-mining data in the exploitational panel, in the seam No. 30, were as follows [34, 51-54]:

- the mining depth: 850 m, 700-900 m,
- the seam thickness: 2.5-5.2 m (combined seams 30+31+32), on average 3.5 m,
- the inclination: 8-17°, locally up to do 20°,

- the roof: 0.10-0.15 m sanded argillaceous schist, which went into gray sandstone and laminated mudstone of 5.0 m thickness, above 6.0 m of moderately grained sandstone,
- the floor: mudstone, locally with claystone and below finely – and moderately grained sandstones,
- the strength to uniaxial compression:
  - the roof – mudstone: 69 to 129 MPa, on average 105 MPa,
  - the floor – mudstone locally with claystone: 45 to 80 MPa, on average 60 MPa,
  - the coal – 9.2 to 22.0 MPa, on average 14 MPa,
- the working dimensions: 5.2 x 3.5 m,
- the pillars in the shape of rhombuses.

Similarly to the Budryk mine the Bolter Miner machine12CM30, having the operational height range 2.4 – 4.5 m and the head width of 5.4 m with the basic difference because the run-of-mine haulage was realized with use of self-propelled Shuttle Car 10SC32, which transported the run-of-mine to the discharge-crushing device of the Feeder Breaker BF-14Q-58-64C type and further on to belt conveyors, was installed. All the machinery system was supplemented with the transport vehicle Scoop 35C-WH-AC with an electric battery drive and the Alminco Scorpion [34] caterpillar drill rig.

Six APB-1-k bolts of the load-bearing capacity 300 kN, and the length 2.4 m in the roof (in rows 4+2) and 3 bolts of the same type in the walls were used for protecting the workings in good geological conditions. In the case of deteriorated conditions, apart from six roof bolts APB-1-k of the length 2.8 m two rope bolts of the length 6.0 m were applied, and in the walls – three bolts of the length 2.8 m. Besides, behind the machine the bolting scheme was supplemented with 2.8 m additional wall bolts and one roof bolt. In the extreme cases eight bolts were used in the roof [34].

The Czech experience indicated explicitly that there was a possibility of conducting an exploitation in the conditions of the Upper Silesian Coal Basin, the more so as the workings safely crossed the zones of overfaults twice. The biggest advance was achieved in April 2015, when 255 m of the working were driven [34]. Due to an intensive monitoring programme in the area of applying independent bolting support, it is worth presenting the selected observations and remarks:

- the measurements of axial forces in the instrumented bolts showed very small loads which maximally reached: 20 kN – compressive forces and 40 kN – tension forces. The first ones were located mainly at the ends of bolts (the

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hole bottom and the working outline), whereas the compressive forces – about 0.5 - 1.2 m from the hole inlet [53],

- low values of loading the bolts are also confirmed by the measurements taken with use of hydraulic dynamometers on rope bolts, which most often did not exceed 10 kN and maximally they reached 30 kN [53],
- the measurements taken with use of vertical hand displacement meters showed the values most often below the limit of accuracy, i.e. 0.5 mm. Maximally it was 7.5 mm and this value was measured in the distance of 2.0 m from the roof. In turn, horizontal hand displacement meters, installed in the walls, indicated significantly bigger values from 200 to 270 mm [53],
- the measurements of pillars, taken with use of 3D lasers indicated that during a ten-month period of observations, the biggest deformations of the coal walls occurred in their bottom part and in the extreme case they reached 50 cm of the horizontal displacement. Such a situation was caused by significant floor upheavals and an impact of gravity. It is important to mention that no displacements of the roof were recorded [52],
- a deformation of coal pillars was caused by significant vertical stresses, which in the pillar of the surface 860 m^2 and the height 3.5 m, were 49 MPa maximally – Fig. 3.4 and also by the slip planes over and below the seam. In the result of such a situation a significant floor upheaval, a convergence of walls and a change of stresses inside the pillars occurred, which had an impact on the coal seam weakening and on yielding of the pillars. Nevertheless, as it has been indicated by the Authors, the pillar core remained undisturbed and ensured a sufficient load-bearing capacity [51, 55-56].

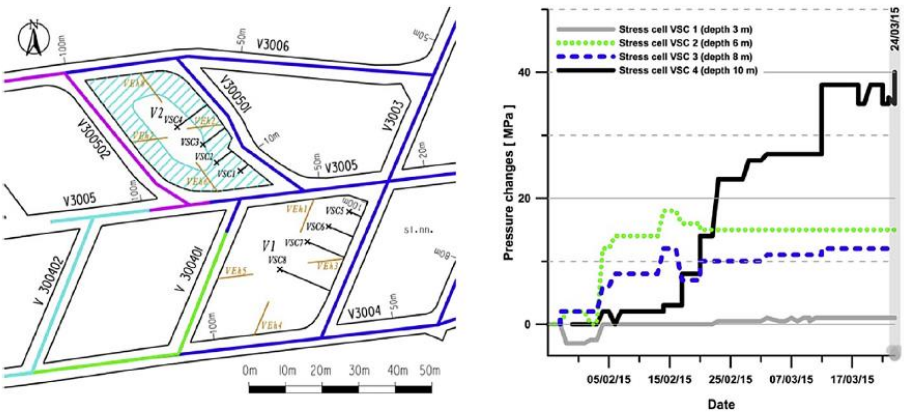

Fig. 3.4. Measurement results of stresses in the V2 pillar [51]

In Germany the bolting support was rarely used before the World War II [57] and the first trials of bolting were conducted over the years 1948-51 in the

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hard coal mines: Consolidation, Neumühl and Diergradt-Merissen [40]. A view of a typical roadway from the fifties of the former century in the independent bolting support is presented in Fig. 3.5.

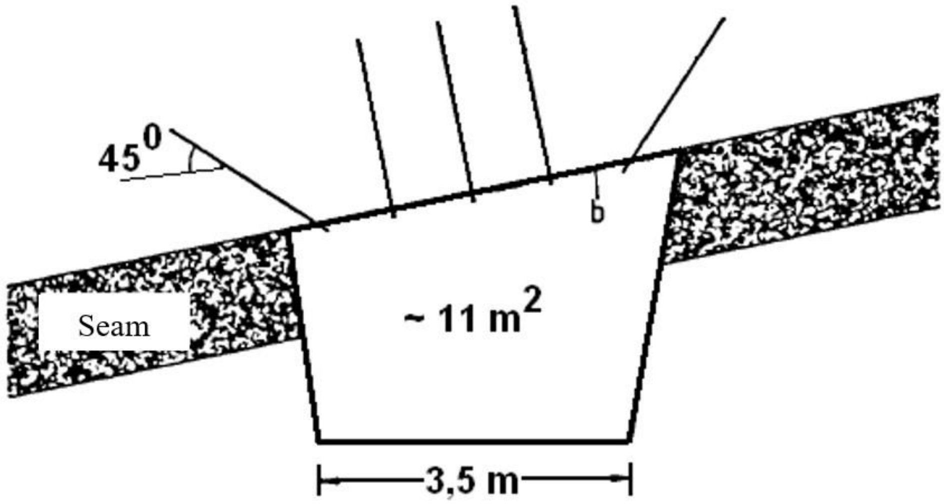

Fig. 3.5. Roadway in the independent bolting support – the fifties [57]

Within the years 1960-1975 an application of independent bolting support was nearly relinquished, what was connected with low labour costs in relation to significant expenditure of money indispensable for a mechanization of the bolting process. After the year 1975 this type of support regained an interest, however it has never reached a significant share in the general metric length of the driven workings [57]. In the end of nineties of the last century an increase of the interest in using independent bolting support (Fig. 3.6) reappeared which is confirmed by the publications [58-61]. However, as it is shown in Fig. 3.7, since 2006 an application of this type support has been nearly completely relinquished.

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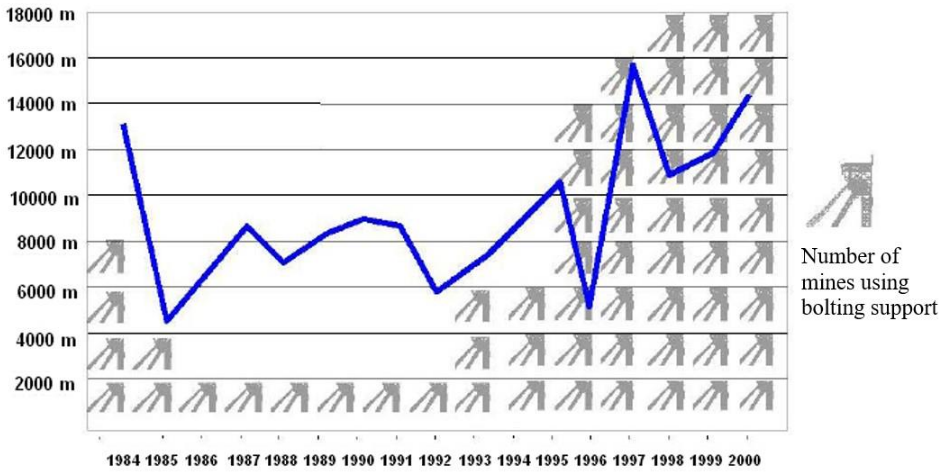

Fig. 3.6. Total metric length of workings in independent bolting support in Germany over the years 1984-2000 [62]

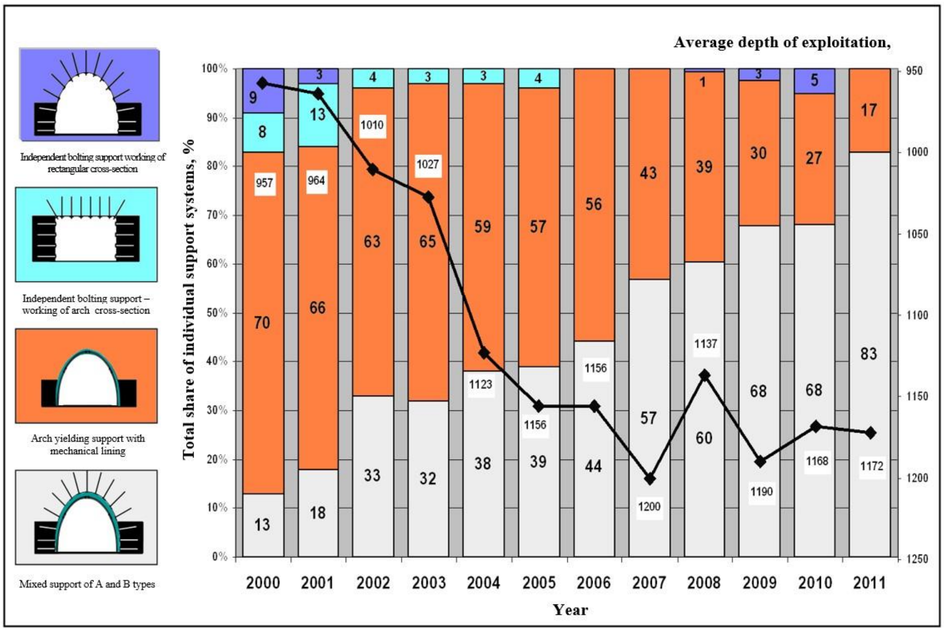

Fig. 3.7. Support of gateroads in Germany in relation to the depths and years [63]

The main reason of relinquishing a use of independent bolting support in German mines was a continuously increasing depth of exploitation and exploitational past events as well as a disadvantageous distribution of stresses in the rock mass, connected with that situation. As it had been predicted (Fig. 3.8), the dominating support of gateroads in German mines was the mixed support (Kombiausbau type A – Fig. 3.9).

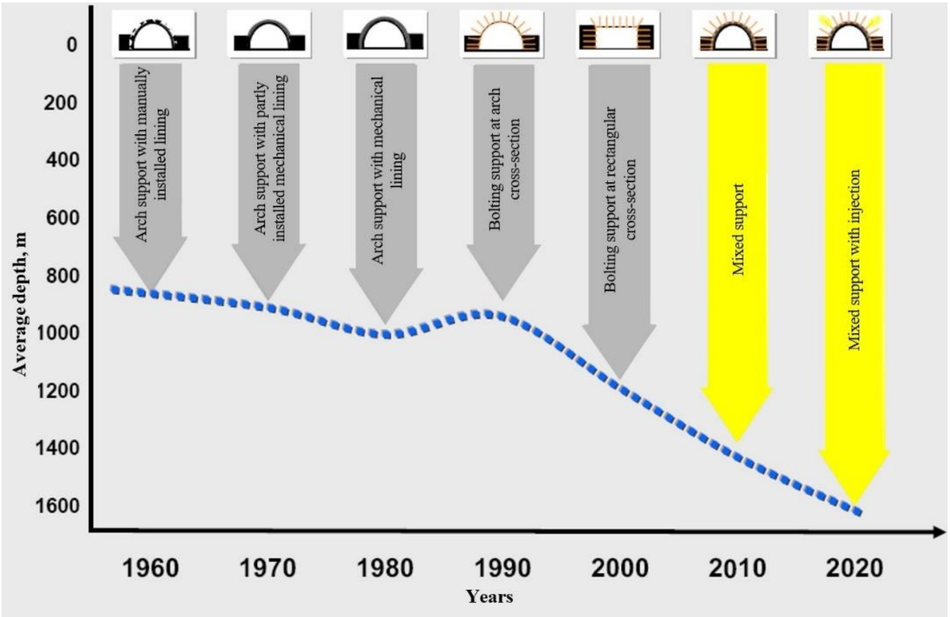

Fig. 3.8. Support of gateroads in Germany in relation to the depths and years [64]

Fig. 3.9. A view of gateroad driven in the mixed support of type A [64]

In Great Britain within the years 1945-1946 a comprehensive publication, contained in six fascicles of the "Colliery Engineering", by a Dutch engineer Mr. Beyl, on positive experience gained due to rock bolting, conducted in England over the years 1942-1943, was edited. Already at that time, based on the measurements of convergence, he noticed that "it is indispensable to apply bolting immediately after an exposition of the rocks". Despite these tests bolting supports were not disseminated at once, but they formed the grounds of their development at the end of fifties of the last century [40, 57, 65]. According to Newson [66] in 1959 there were nearly 100 km of workings driven in independent bolting support in Great Britain.

In mid-sixties, due to repeatable cavings in the workings, where mechanical bolts were used, this type of support was completely abandoned. Further trials with the bolts, fixed with glue charges this time, took place in the seventies and early eighties of the last century [67]. In 1982 the coal mining industry consumed 330 thousand of steel bolts and nearly a million of wooden ones [57]. Slightly later on, in mid-eighties some trials with the support systems, used in the Australian and American mining industries, were conducted. Based on these tests, it was stated that for the conditions of the hard coal mining industry on the British Isles, the Australian systems containing bolts of significant load-bearing capacity, glued-in with use of the polyester resin charges, were most appropriate.

A real revolution in applying independent bolting support started in Great Britain in 1987. Then the mining companies were owned by the State (The British Coal Corporation, owned by the State Treasure was active till 1994, when it was privatized) and exactly from the government part there was a strong pressure on a reduction of costs and a limitation of subsidies in relation to an increasing share of imported coal. A reaction to this demand was a comprehensive research-and-development programme, including, among others, issues related to the support. One of the conclusions, resulting from this programme was a statement that for achieving positive financial effects, which could create possibilities of competing with foreign coal producers, it was necessary to apply a rectangular cross-section of the working, in gateroads, enabling a use of Continuous Miner machines and of the independent bolting support [68-69]. In the result of those activities, in the nineties, bolting support was, at ensuring the appropriate level of safety, successfully implemented in the coal mining industry in Great Britain, and in 2006 about 95% of all the new gateroads were driven in independent bolting support [68]. Typical dimensions of the workings, driven in independent bolting support in Great Britain were contained in the ranges: the height: 2.5 to 5.0 m and the width: 4.5 - 6.0 m [70]. The bolts of the length 2.4 m and long bolts of the length about 4.0 m were used for supporting the workings. The bolting density was 1.5 bolts/m^2 on average, whereas in some cases it reached 3.0 bolts/m^2. A characteristic feature of the British mining industry [71] was a common use of wide pillars between two following mining panels, which was to ensure a maintenance of satisfactory overall dimensions of gateroads driven in independent bolting support.

According to Ma et al. [27] an application of the JOY12BM15 machine at the depth of 700-800 m, at the spacing of bolts 0.8 m, enabled to achieve about forty-minute cycle and advances from 6 to 7 meters per day. In turn, the Riccall mine achieved advances of about 125 m per week at the depth of 750 do 850 m.

An exemplary scheme of the bolting system in the English mining industry is shown in Fig. 3.10.

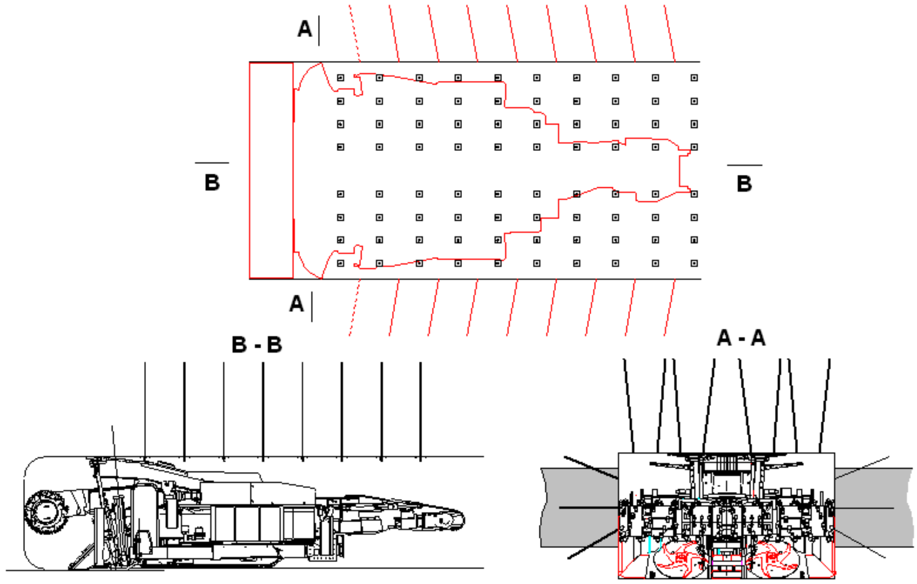

Fig. 3.10. An exemplary bolting scheme in one of the gateroads at the Thoresby mine [72]

At the United States after the first use of bolts in 1905, mentioned at the beginning of this publication, the following pieces of information about their positive application concern the Sagamore mine in South Virginia, which took place in 1917. At the beginning of twenties of the XX century the first mining enterprise - *St. Joseph Lead Company* started to use bolts on a large scale [73]. A real expansion of the bolting support use in the USA started at the end of forties. At the same time the U.S. Bureau of Mines deserved a credit for it as it became a spokesman of the new technology [74]. Undoubtedly the Weigl's [75] publication also had a positive impact. He presented the main principles of bolting which are valid up till the present time:

– strengthen the weakened rock mass below the natural arc plane of loosening,
– bolt together (combine) weak, narrow strata to create thicker and stronger strata,

– install bolts at the beginning of the cutting cycle.

A development of the bolting technology was so rapid that during less than two years over 200 mines applied this type of support [76]. As Fletcher [77]: put it "*bolting has been approved by the mining industry at the bigger speed than any other changes since the moment of an introduction of mechanization*". The statistical data confirm this fact very well, because in 1956, 26.4 million bolts were used per month and the bolting support was used in 424 mines [40]. It is estimated that in 1976 about 100 million bolts were used a year and in 1984 - about 120 million, in 1988 – about 88 million, in 1999 - about 100 million and over the years 2005-2006 - about 68 million [76-81]. It is assessed that at present about 68-80 million bolts are used a year [82-83]. A reduction in the demand, observed over the years, is connected with an increased share of longwall exploitation, which uses only about one fourth of the number of bolts in relation to the room-and-pillar system.

Up till the present time an independent bolting support is the basic type of support used in the hard coal mines in the United States. An additional strengthening of the basic support (Fig. 3.11), is used only in the case of excessive deformations of roadways, connected with increased values of stresses, for example in the zones of exploitational pressure impact [6, 70, 84-85]. In such cases the following strategy is used [87]:

– *Primary support* – typically steel bolts glued in at their full length, possibly with profiled linings and with mesh linings.

– *Secondary support* – installed after the primary support, most often at crossings, where a significant span of the workings occurs and typically it includes an installation of rope bolts.

– *Supplementary support* – executed after the secondary support due to bad geological-and-mining conditions and typically it includes an installation of the bolt-strand and chock supports in a form of wooden chocks, pillars or other solutions whose characteristic is described among others in the Barczak's [54] publication.

Fig. 3.11. Roadway driven in independent bolting support strengthened with wooden chocks [85]

An application of independent bolting support in hard coal mines in the United States is connected with the fact that apart from the Australian mining industry, the best production rates in the world are achieved there. In such a situation it is necessary to drive gateroads rapidly, the more so that in the American longwall system from two to five gateroads are driven for each panel, however the systems with three gateroads dominate. The width of gateroads is contained in the range 4.5-6.0 m and they are driven mainly with use of Continuous Miner machines – Fig. 3.12 [87-88].

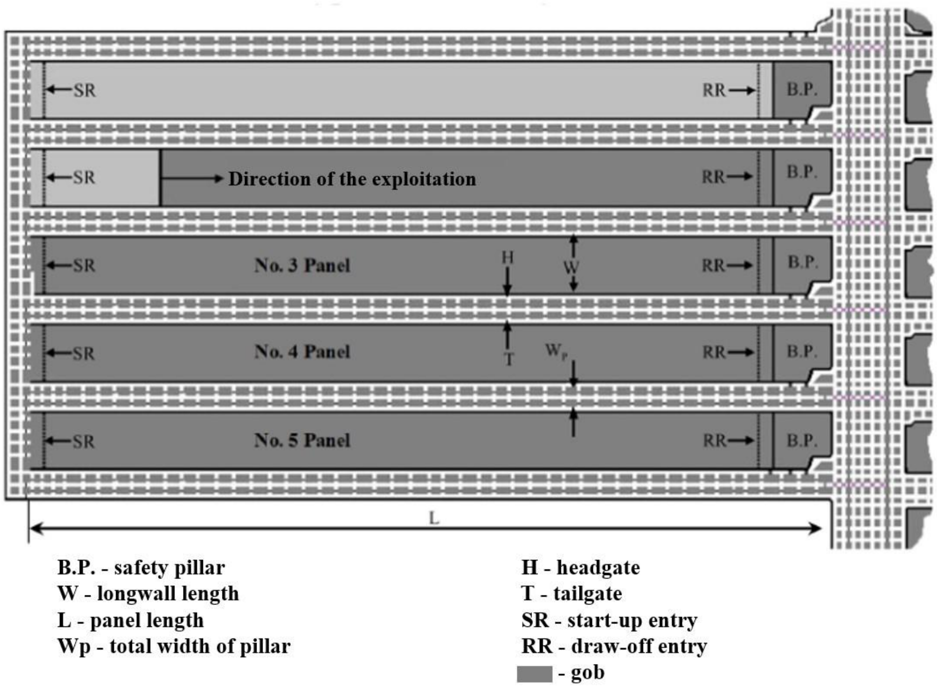

Fig. 3.12. Typical scheme of the exploitational panel development in the United States [6]

In the United States a broad range of bolts is used, among which bolts with initial tension and without any initial tension, fixed pointwise and glued-in at their full length, with tightened-up bolt end and different immovable ends, steel and rope as well as specialistic solutions such as bolt-strand supports [83]. According to Dolinar and Bhatt [78] five main basic bolts can be specified in the American mining industry:

– mechanical of pointwise fixation and mechanical bolts with additional gluing-in (Fig. 3.13),
– steel, glued-in with use of glue charges with an initial tension (*torque-tension bolts*),
– connected bolts consisting of a smooth and ribbed rod connected with a link (*combination bolts*),
– steel, glued-in with use of glue charges.

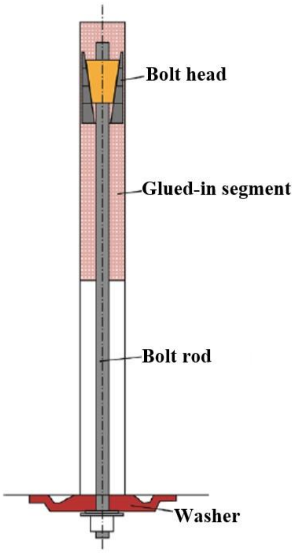

Fig. 3.13. Mechanical bolt with glued-in segment [83]

Steel bolts of 19 mm dia. and of continuous gluing-in have over 80% applications in gateroads. Their length is in the range 1.8-2.4 m and they are installed in the spacing of 1.2 m. The load-bearing capacity of bolts of 19 mm dia. is about 180 kN, however for the 20.4 mm dia. - about 260 kN [6, 78, 86, 89].

The history of using bolts in Australia started at the moment of a construction (1949-1969) of the hydroelectric and irrigation complex “Snowy Mountains Hydroelectric Scheme”, when for the first time this type of support for strengthening the rock mass [90] was implemented. However, the first mine which in1949 started to use bolts together with wooden support was the Elrington mine, located in New South Wales [91]. In the seventies of the last century wooden supports, supplemented with steel supports, dominated in the Australian mining industry. Relatively rarely mechanical bolts were used. Significant changes in the share of independent bolting support happened in the end of seventies and at the beginning of eighties, what was connected, among others, with an increasing number of accidents which happened during an installation of chock supports, economic calculations, an introduction of the state-of-the-art glue charges and an enlargement of the offer for the bolting equipment. At the beginning of nineties the following step was taken, consisting in an introduction of rope bolts, not only as a form of strengthening the existing roadways but also as an element of the primary support system [92]. In 2011 the Australian mining industry used about 7 million bolts [91].

At present, in the conditions of the Australian hard coal mining industry, the shape and support of gateroads are mainly determined by a possibility of a rapid drivage of these workings and their protection with an appropriate type of support. In the conditions of longwall mining these factors have an impact on the production rates [93]. That is why gateroads are driven in the rectangular cross-section, mainly with use of bolting support, both at the stage of drivage (primary support) as well as during probable strengthening of the working (strengthening support). Typically the scheme of the panel cross-section for longwall mining is very similar to the scheme used in the United States and it is based on a two-roadway system – Fig. 3.14.

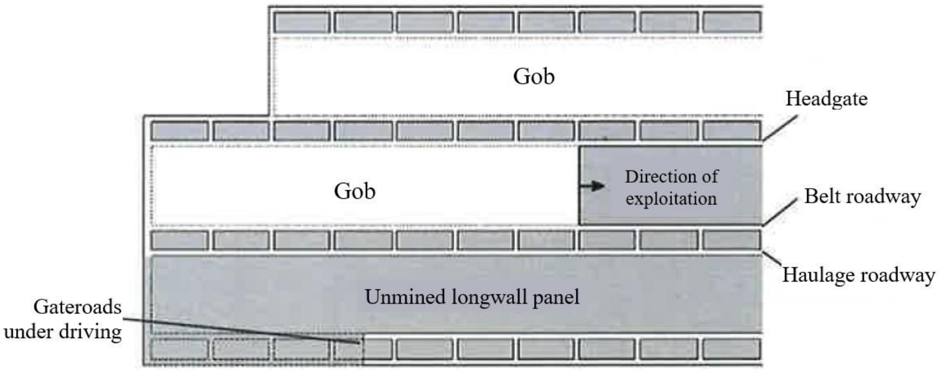

Fig. 3.14. Scheme of exploitational panel in the Australian mine [94]

The width of roadways in the Australian system is contained in the range from 4.0 to 6.0 m at the height from 2.0 to 4.0 m and driving is executed with use of Continuous Miner or Bolter Miner machines [9, 88, 95].

Steel bolts, made of steel of the tensile strength 770 MPa, 22 mm dia. and of the load-bearing capacity of 260 kN (real load-bearing capacity exceeds 300 kN) are most often used for an installation of independent bolting support in Australia. The bolts glued-in at their full length with glue charges, are based on appropriately selected, specialistic polyester resins. Similarly to the conditions in the Polish hard coal mining industry slow – and quick-fixing of different time of gelation charges are used. Mainly roof bolts of the length 1.8 to 2.4 m in the number from 4 to 8 pieces and wall bolts from the range 1.2 to 2.1 m in the number from 2 to 6 pieces are applied. The whole scheme of the basic bolting system is supplemented, most often, with profiled linings of the cross-section in the shape of the “W” letter (*W-shaped straps*) and where it is necessary with steel mesh linings or in the case of walls, in the place of future exploitation, with cut mesh linings [84, 96]. An exemplary scheme of independent bolting support is presented in Fig. 3.15.

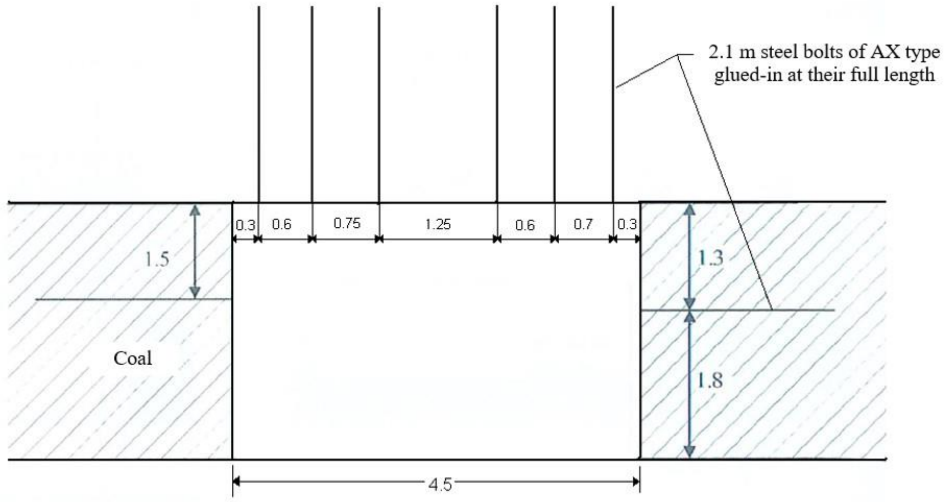

Fig. 3.15. An example of the scheme of independent bolting support – Angus Place mine, Western Coal Field, New South Wales, Australia [96]

In difficult mining-and-geological conditions long, rope bolts of significant load-bearing capacity are additionally used in the bolting system. They were introduced to the Australian mining industry in the early eighties, initially for strengthening the crossings of roadways and in gateroads where weak roof rocks occurred. They had the dia. of 15.2 mm and they were installed in the holes of 50 mm dia. with use of cement mortar. At present the bolts of the Flexibolts type of 23.5 mm dia. and the load-bearing capacity of 580 kN, whose length is 8.0 m and the Megabolts, implemented in 1997, which occur in different variants: 500, 630, 840 and 900 kN, are most often used as primary and supplementary supports. For example the version 630 kN (Fig. 3.16) consists of 9 wires and it has the dia. of 40 mm. An installation takes place in the bolting hole of 45-55 mm. An assembly of bolts is conducted with use of glue charges and after their binding, it is possible to apply initial tension (conventionally 250 kN) and an injection with cement of the left length of the bolt [97-98].

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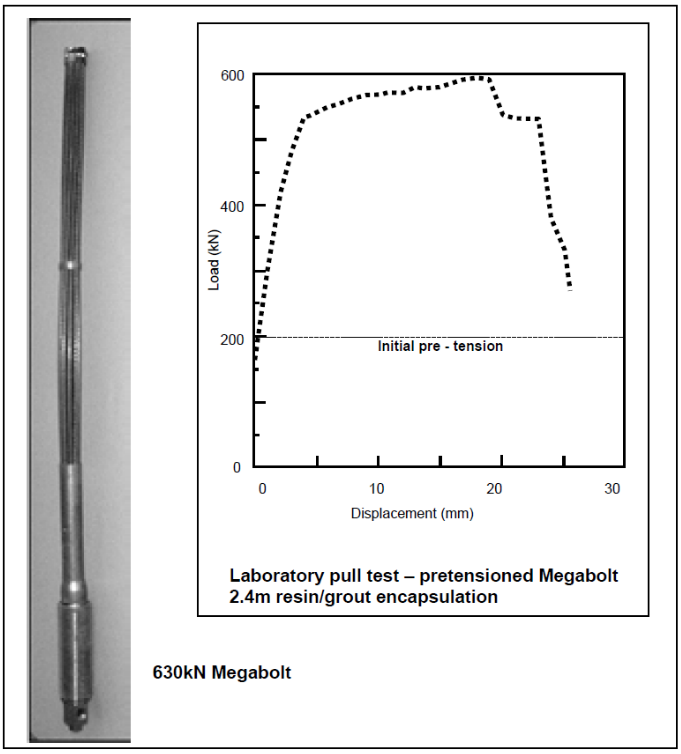

Fig. 3.16. Characteristic of the load-bearing capacity of the 630 kN Megabolt [98]

An interesting solution, which has not been used in Poland so far and it is used in Australia and the United States, is the system of the bolting-strand support (Truss). A design of such a support is not complicated and expensive as the bolts in connection with the rope or ropes themselves are used. It is possible to distinquish two basic systems, i.e. monopartial [99] and three-partial [100] – Fig. 3.17.

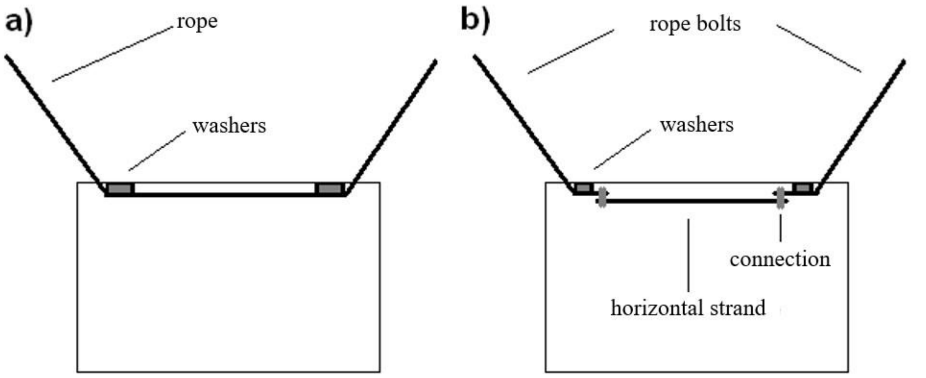

Fig. 3.17. Typical scheme of bolting-strand support: a – monopartial, b – three-partial [101]

According to Kidybiński and Nierobisz [102] a typical length of bolts, incorporated in the bolting-strand support, installed in the roof, is about 2.5 m and bolting holes of 25 – 35 mm dia. are drilled in the distance of about 0.6 m from the walls. The installation angles, in relation to the roof plane, according to different sources, are similar and they are in the range of: 45 - 60°. A comprehensive survey of literature in this scope was conducted by Cała et al. [35].

The bolting-strand support is most often used in hard coal mines as additional support, in the places where weak roof or significant stresses in the rock mass occur and where the bolting support itself can be insufficient for protecting against roof falls, resulting from generated fractures and fissures over the bolting horizon [103]. In such cases bolting-strand support can be indispensable for maintaining the working stability. An exemplary application of this type of support is presented in Fig. 3.18 and 3.19.

Fig. 3.18. Bolting-strand support with steel ropes before applying initial tension [95]

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Fig. 3.19. Bolting-strand support with steel ropes in the heading working [104]

According to Turek [95] the advances of 44 meters in the Crinum mine were achieved with use of the JOY 12SCM30 machine during an 8.5 – hour shift and an average advance of the roadway, including the time spent on an extension of the belt conveyor, reached 15 meters per shift. An application of the ABM 20 machine, in the case of different mines, gave the effect in a form of average advance per shift in the range from 7.0 m to 20 m, which gives daily advances in the order from 14 to 40 m. Turek [95] also presented a list of advances which are shown in Table 3.1.

Advances of roadways made in independent bolting support based on the Turek's list [95]

Table 3.1.

Mine	Bolting scheme	System of shifts	Average advance per shift	
			Gateroads	Main drifts
1	6 x 2.1 m – roof bolts 4 x 1.5 m – wall bolts Spacing – 1.2 m	3 x 8 h /5 days a week	15.0 m	12.0 m
2	6 x 2.1m - roof bolts 6 x 1.4m – wall bolts and mesh lining Spacing – 1.0 m	8.75 h /15 shifts a week	13.07 m	9.28 m
3	4 x 2.1m - roof bolts and "W" steel canopy, Spacing – 1.5 m	9 h /9 shifts every two weeks	8.0 m	9.0 m
4	4 x 1.8 m - roof bolts and mesh lining 1 x 1.2 m - wall bolts Spacing – 1.2 m	10.0 h /15 shifts a week	25.0 m	15.0 m

5	4 x 1.8 m - roof bolts Spacing – 1.0 m	8 h /15 shifts a week	20.0 m	Lack of data
6	4 x 1.5 m - roof bolts Spacing – 2.0 m	7 h	23.0 m	12.6 m
7	4 to 6 x 2.1 m - roof bolts Spacing – 1.0 m 2 x 1.2 m - wall bolts Spacing – 1.5 m	8 h	7.6 m	7.4 m
8	6 x 2.1 m - roof bolts 4 x 2.1 m - wall bolts Spacing – 1.0 m	9 h	7.0 m	6.0 m

Based on a fifty-year experience in using independent bolting support in Australia, Frith et al. [105] specified eight basic principles according to which the rock mass should be strengthened correctly:

1. The place of bolts installation should consider the distance from the face front.
2. An appropriate selection of bolts lengths and correctly selected, as regards geotechnical conditions, their scheme of spacing.
3. Minimization of pressure acting on glue charges during an installation of bolts to avoid a disadvantageous impact of increased pressure on strengthened rock strata.
4. Proper mixing of glue charges to obtain a correct connection between bolt-rock mass.
5. An application of rock charges of adequate parameters.
6. Maximal use of the initial pull effect by tightening up the nut during an installation of the bolt.
7. A protection of the staff against any rocks falling down from the roof, especially those which may occur among the bolts.
8. An application of continuous processes both for a correct installation of bolts as well as for a management and control, in the conditions of uncertainty, connected with strengthening of the rock mass.

In China, in underground hard coal mines, bolts started to be used in 1956 [106]. Initially only mechanical bolts and bolts of the split-set type, which were characterized by low strength parameters, were used. In 1995 roof bolting was applied in about 15% of gateroads. Within the years 1996-97 an Australian bolting technology was started to be introduced, which enabled to use new materials, in particular in a form of glued-in bolts at their full length, which were characterized by significant load-bearing capacities. Another important date in the Chinese mining industry is the year 2005, when a research programme on

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supports, used on significant depths and difficult mining-and-geological conditions, was started. The programme enabled, among others, to increase advances of the faces, driven in independent bolting support. At present, in the state-owned hard coal mines, about 70%, and in some regions 90% roadways are driven in independent bolting support [83]. Basic schemes used in Chinese mines are presented in Fig. 3.20, and mechanical parameters of steel bolts are given in Table 3.2.

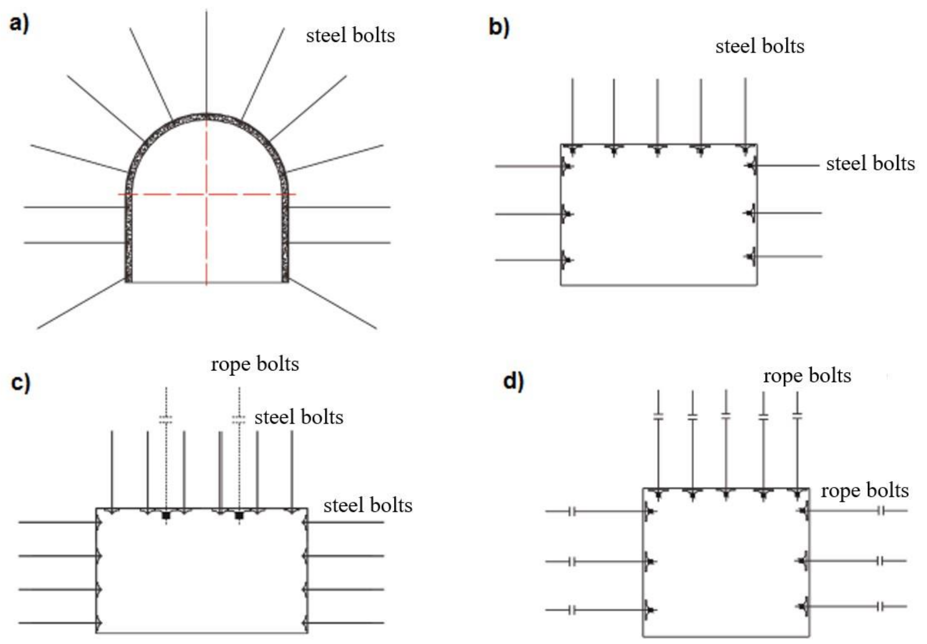

Fig. 3.20. Basic schemes used in Chinese hard coal mines: a) steel bolts and shotcrete, b) steel bolts, c) steel and rope bolts, d) rope bolts [83]

Basic mechanical parameters of steel bolts used in the Chinese mining industry based on the Kang's statement [83]

Table 3.2.

Type	Diameter [mm]	Yield point [MPa]	Tensile strength [MPa]	Elongation [%]
Q235	14-20	235	380	25
BHRB335	16-22	335	490	22
BHRB400	18-22	400	570	22
BHRB500	18-25	500	670	20
BHRB600	20-25	600	780	18

Detailed results of laboratory tests of bolts and washers, used in China, can be found in the publication [107] – Fig. 3.21.

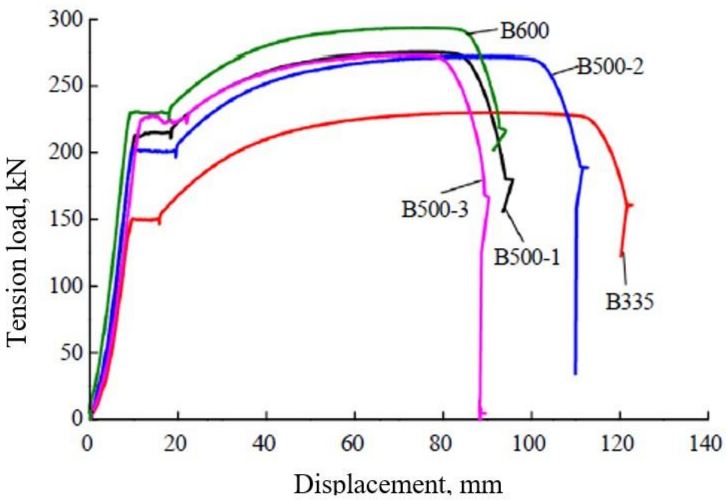

Fig. 3.21. Results of laboratory tests of bolts used in China [107]

The rope bolts, developed for the needs of Chinese hard coal mines, are characterized by significant load-bearing capacity and a possibility of their installation in the holes of small diameters. They are made of nineteen-wire steel strand and they are available in diameters: 18, 20, 22 and 28.6 mm. For example, a bolt of 22 mm achieves the load-bearing capacity of about 600 kN, at the elongation of about 7%. The washers of dimensions 300 x 300 x 16 mm and steel mesh lining, made of wire 6.5 mm [108-109], are used for these bolts.

A scheme of the working support, made in independent bolting support with use of ropes, is presented in Fig. 3.22. In this scheme the bolts 22 mm, of the length 4.3 m, which had already been mentioned, were applied. They were assembled with use of glue charges, at the initial tension of 200-250 kN and then the cement grout was forced in to obtain full gluing-in. In the roof bolts and in the walls three bolts at spacing 1.2 m were used. It is characteristic that the roof bolting holes had the dia. of 28 mm, and the wall ones - 42 mm. All the bolts were installed rectangularly to the outline [109].

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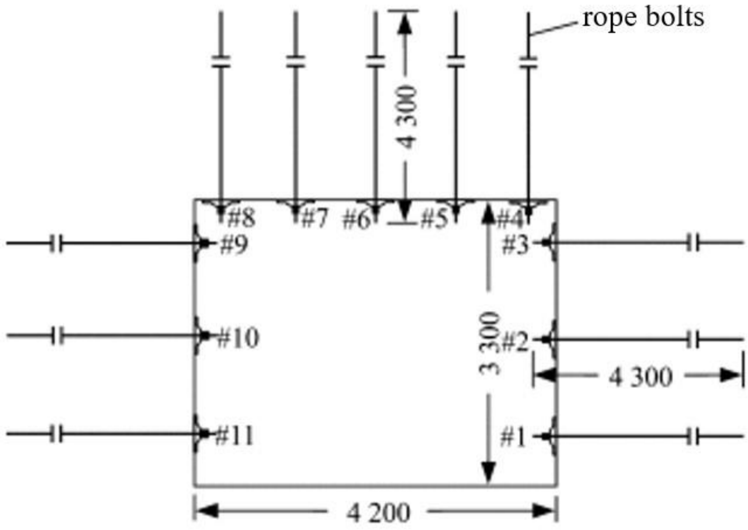

Fig. 3.22. The working made in independent bolting support with use of rope bolts – units in mm [109]

Due to the fact that highly productive Chinese mines need a technology which will enable rapid and efficient driving of gateroads, special attention is paid to this subject-matter. Average advances in underground coal mines in western China exceed 500 m/month, what is not satisfactory [110]. One of the applied solutions is the BMRET (*Bolter Miner Rapid Excavation Technology*), which is a technology of rapid driving of workings with use of the Bolter Miner type of machines. Average advances in deep mines, using this technology, are in the range from 15 to 25 meters per day [111]. As it is shown by the data presented in the publication [27] this technology, in advantageous conditions, enables to achieve nearly 99.72% increase of advances, which in effect gives an average monthly advance of 36.15 m/day and 1080 m/month.

One of the basic BMRET elements is a change of the bolting system (Fig. 3.23) and the fact that due to these changes a reduction of the cycle duration from 48.2 to 22.7 minutes is obtained.

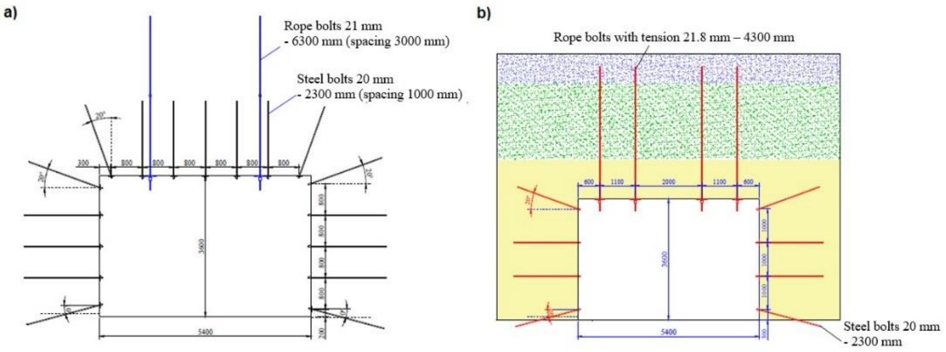

Fig. 3.23. Changes in the bolting system a) originally used b) an application of the BMRET technology [27]

The independent bolting technology is also used in Russia. Among companies, using the technology of independent bolting support, the following ones can be distinquished: Raspadskaya, Severstal, JSC Yzhkuzbassugol [112]. According to Artemyev and McInally [113] the SUEK Company (Siberian Coal Energy Company) purchased, among others, 12 Bolter Miner machines. In the Volgaskorska mine, in the area of the Russian town of Workuta in 2014 over 1400 rm of the roadway advance was achieved during a month [9].

Among other countries, using independent bolting support, it is possible to mention India [114-115]. According to the data presented in the publication by Mukhopadhyay and Sharma [116] in the mines, belonging to the state-owned company Coal India Limited, eight Continuous Miners were in operation and in the following nineteen ones this technology was under implementation. Coal mining is based on room-and-pillar system. For example in the Tandsi mine, typical for the Indian mining industry, for a mechanization of cutting the Joy 12CM Continuous Miner, the Joy Quad Bolter, 2-3 Shuttle Cars, the Feeder Breaker and the Scoop were used. A system of workings with faces is shown in Fig. 3.24.

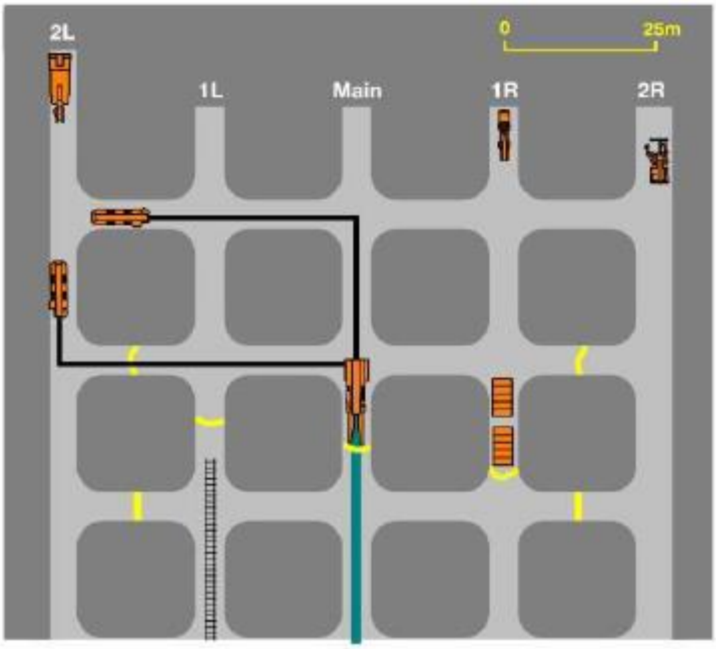

Fig. 3.24. Room-and-pillar system in the Indian hard coal mine [116]

The dimensions of pillars, at the depths of exploitation 220-400 m were 35 m x 35 m, 40 m x 40 m and 40 m x 35 m, and the dimensions of workings 3.6 x 2.7 m, 4.8 m x 2.7 m and 4.8 m x 3.5 m. For protecting the roof four steel bolts were used [116].

In South Africa, in the forties of the twentieth century, mechanical bolts were introduced in the mining industry for the first time. Since that time a dynamic development of this type of support has caused that it became the basic method of strengthening the roof and walls in hard coal mines. The basic types, applied at present, are steel bolts, glued-in at their full length both with use of charges having the same time of binding, as well as rapid-and slowly-binding and steel bolts glued-in segmentarily and mechanical bolts. The bolts of diameters 16, 18 and 20 mm [117-118] are mainly used.

In practice in South Africa four 20-mm bolts of the load-bearing capacity of 180 kN in a row are applied. The spacing of rows is 1.5 to 2.0 m. In the case of selecting the length of bolts and their spacing, the notion of maximal thickness of weakened strata [117] is employed, according to the data presented in Table 3.3.

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Selection of the lengths of bolts and of the spacing of rows [117]

Table 3.3.

Distance between rows of bolts [m]	Length of bolts [m]	Maximal thickness of weakened strata [m]
1.5	0.9	0.59
	1.2	0.78
	1.5	0.98
	1.8	1.17
	2.0	1.30
2.0	0.9	0.21
	1.2	0.29
	1.5	0.36
	1.8	0.43
	2.0	0.48

In Fig. 3.25 an example of the face, in the working, made in independent bolting support, in the mine producing hard coal, is presented.

Fig. 3.25. Bolting in the face in a hard coal mine in the Republic of South Africa [119]

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In Iran the Tabas mine uses independent bolting support. The workings of rectangular cross-section shape, of the surface 15 m^2 and 18 m^2, are driven there. The seams have the thickness from 1.8 to 2.2 m, at the inclination 11°-26°. The strength of roof rocks is in the range 32-73 MPa. The basic bolting scheme embraces an application of 13 bolts of AT type of the length 2.4 m, in the system 7 + 6 pieces for each running meter of the working. The walls are strengthened with three bolts of AT type of the length 1.8 m and in the case of the opposite wall – with four cutting bolts also of the length 1.8 m [120-121] – Fig. 3.26.

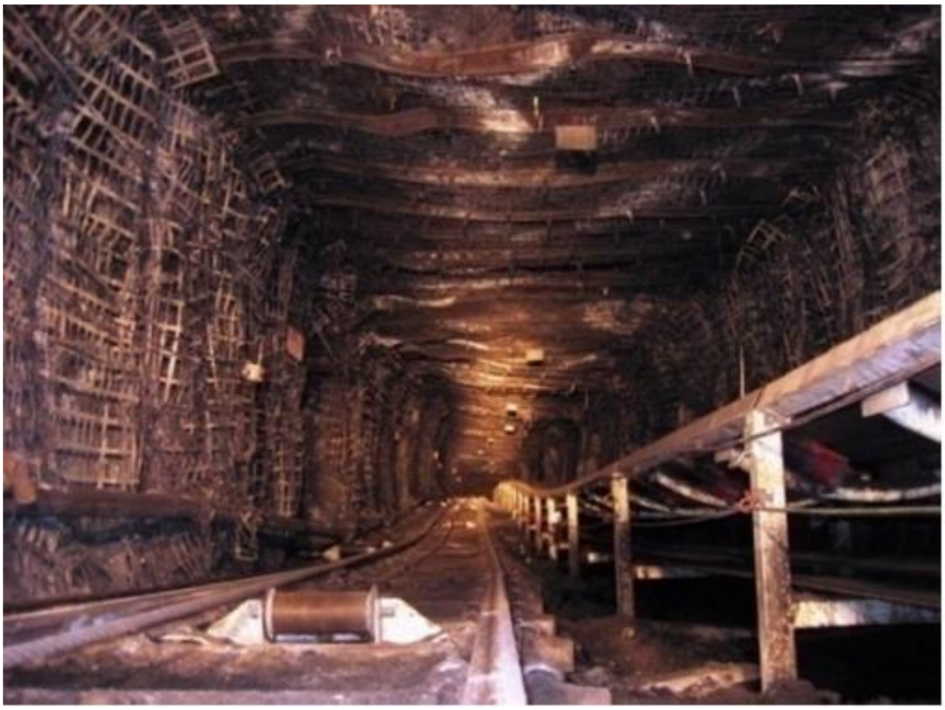

Fig. 3.26. The working driven in independent bolting support in the Iranian Tabas mine [122]

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4. The history and the present of roof bolting technology

Aleksandra Otto[1]

4.1. Mechanism of roof bolting operation

At present the basic type of support, used in the Polish hard coal mining industry, is standing support made of steel, yielding arches. Standing support has a passive character, which means that its co-operation with the rock mass starts no sooner than a significant displacement of the working contour begins. Its task consists in carrying loads coming from the zone of cracks situated above the working, thus protecting it against roof caving. A co-operation mechanism of roof bolting support with the surrounding rock mass is completely different. The main task of roof bolting is a generation of a self-supporting arch in the working roof and a prevention against decreasing the rock mass parameters around the working. A correctly designed roof bolting support should mobilize the rock mass for self-supporting to the maximum possible extent [35].

Two basic co-operation models of bolts with the rock mass can be distinguished in a stratified rock mass occurring in the majority of underground mines in Poland. The first one consists in tacking the strata having low strength-and-deformation parameters, however the other one consists in fastening (suspending) a stratum, having low strength-and-deformation parameters to the strata of high strength-and-deformation parameters, situated above [35].

In practice, in mine conditions a combination of both mentioned models occurs. A task of bolts is to connect rock strata with one another in such a way that makes their displacement in relations to one another impossible and thus to generate a stratified rock beam, consisting of strata having different strength-and-deformation parameters and ensuring the working stability [35].

4.2. Historical outline of using roof bolting support in the world

The first pieces of information about using bolts in underground mining industry date back to the year 1872 and they come from Wales, but a dynamic development of this type of the support system took place in the American mining industry. In the United States bolts were used for the first time for protecting workings already about 1905. Nevertheless an application of systematic roof bolting for the first time took place no sooner than the twenties in the mine St. Joseph Lead Company [35].

At the turn of forties and fifties roof bolting support revolutionized the American mining, becoming the basic type of support used in the coal mining

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industry in the end of fifties, efficiently replacing standing support. Only in 1951 roof bolting replaced steel and wooden props in nearly five hundred American mines. An introduction of roof bolting systems caused a significant increase of productivity and also an improvement of safety connected with a reduction of roof falls and a decrease of accident rates [35, 123].

A possibility of an easy introduction of roof bolting support for protecting workings in American mines was connected with the following mining-and-geological conditions [123]:

– A possibility of performing a full deposit development exclusively in coal mainly resulting from horizontal deposition of strata – through cutting out workings in coal a reduction of horizontal stresses in pillars at nearly unchanged horizontal stresses in roof and floor strata occurs, having an advantageous impact on roof stability due to a reduction of tensioning forces, occurring at bending the roof beam.

– Occurrence of flat, stratified roofs connected with a possibility of generating a stable roof beam – an initial tension of bolts enables an introduction of the perpendicular stress, increasing friction on splitting planes.

An additional factor, conditioning a possibility of applying roof bolting support in the United States included an economic situation. As opposed to the Polish mines in the similar period, the American mines had private owners, obtaining no financial support in a form of a subvention from the government. Due to that it was indispensable to preserve a tight discipline in the domain of production costs to achieve a positive economic effect. That is why it was indispensable to limit a use of standing support and also to limit a development of expensive rock workings to required minimum [123].

An increasing popularity of roof bolting technology in the United States caused a development of this type support systems in mines all over the world and also a dissemination of mining room-and-pillar system for extracting deposits of seam type. The lowest production costs obtained in American, South-African and Australian mines, commonly using independent roof bolting support, confirm an efficiency of this technology [43].

4.3. Historical outline of using roof bolting support in Poland

For the first time in Poland roof bolting support was applied in 1916 in the Upper Silesian Pokój mine. A use of bolts consisted in bolting cracked walling to the stratum of compact rock mass and in roof bolting in the sedimentation roadway. An application of this type support system was too innovative as regards that period of time and the following report on using the roof bolting support in the Polish mining industry comes from 1938 and it concerns the Michał mine, where bolts were used for protecting the main drift situated at the depth of 340 m [35].

In the after-the-war period some work on using roof bolting support was conducted by Główny Instytut Górnictwa (Central Mining Institute), realizing for the first time experimental roof bolting in 1954. In 1960 an operational manual on an installation and inspection was published, but only in 1976 the Ministry of Mining edited "Temporary guidelines of using roof bolting support, bolting-and-standing and of simple standing support in hard coal mines", which were in force till the beginning of the nineties. At the turn of the sixties and seventies the roof bolting support was used as the basic support in the mines of Legnica-Głogów Copper Region. In the same period numerous, successful trials of roof bolting in hard coal mines (Mines: Siemianowice, Bielszowice, Katowice, Dębieńsko, Michał, Czeladź and Milowice), were undertaken, however they were not finished with an implementation of roof bolting support on an industrial scale. The reasons of such a situation might result from, among others, a low work capacity, caused mainly by a use of time-worn roof bolting devices [42-44].

In the nineties of the former century, in the Polish hard coal mining industry an interest in using independent roof bolting support increased again, among others, for protecting headgates and tailgates, affected by disadvantageous impact of secondary rock pressure. Nevertheless, two serious roof falls (2.11.1994 – Anna mine, 5.07.1995 –Staszic mine) reversed this trend. With a short break in the years 2000-2002, in hard coal mines a gradual decrease in the length of workings, developed in the independent roof bolting support, occurred until its use was completely stopped in 2009 [42-43].

At present in the Polish hard coal mines bolts are used as follows [44]:

– for bolting machines and equipment to ensure their stabilization,
– high bolting consisting in bolting horseheads and straight connection pipes from V sections,
– high bolting among frames of standing support,
– bolting of canopy arches at crossings of a longwall face with gates,
– bolting of starting passages of longwalls,
– bolting of crossing of roadway workings.

4.4. Past experience resulting from using independent roof bolting support in Polish hard coal mines

In nineties intensive activities, oriented onto a development of roof bolting systems were undertaken in the Polish hard coal mines. Only in the period 1991÷2000 44 underground mining plants applied roof bolting support in

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different forms, however 300 km of workings were developed in independent roof bolting support system. A possibility of an extensive application of mine bolts was conditioned by [43]:

- amendments to legal regulations concerning issues of underground workings supports,
- a dynamic market development of bolts, bolters and glue charges,
- trainings of engineering staff in the scope of workings' bolting,
- nominations of qualified experts on issues of roof bolting support by the President of the WUG (State Mining Authority),
- a realization of indispensable research projects in the scope of a bolt co-operation with the rock mass and an application of bolts for protecting against dynamic phenomena effects,
- an organization of scientific-and-technical conferences concerning the subject-matter of bolting workings.

The experience in using roof bolting support at the beginning of the XXI century was unmistakably positive, in particular in using the bolting technology in starting passages of longwalls and gates liquidated behind the longwall front. In the gates, maintained behind the longwall front, no problems were experienced with the roof control, however the phenomenon of floor upheaval turned out to be a problem. The basic advantages resulting from an application of independent roof bolting support, obtained during the period under discussion, are [43]:

- a reduction of materials consumption,
- a reduction of reinforcement time of starting passages,
- an increase of work safety,
- a quality improvement of produced coal due to a limitation of ripping dirt in the roof.

The experience of the Polish mines in using independent roof bolting support at the beginning of the XXI century, is presented on the examples of Mysłowice, Chwałowice and Jankowice mines.

Mysłowice mine

In Seam 418 of the Mysłowice mine, situated at the depth from od 460 to 590 m, the Incline 3 western was located. The area of locating the incline is between the Mysłowicki Fault in the west and the Brzęczkowicki Fault on the south-eastern side. It was decided to install independent roof bolting support at the section situated at the depth of 560-590 m. The seam thickness in this area varied from 1.6 to 2.3 m and the inclination – from 2° to 10°. Physico-mechanical parameters of the rocks, surrounding the seam and of the coal were [43]:

– coal laboratory strength to uniaxial compression – R_c = 25 MPa,
– penetrometric compressive strength of roof rocks – R_c = 46 MPa,
– laboratory compressive strength of floor rocks – R_c = 26 MPa,
– rock jointing index – *RQD* = 68%,
– soakability coefficient – *r* = 1.

In the independent roof bolting support 400 m of the working, driven with the AM-50 roadheader, were developed. Bolts of 2-metre length were installed in the rock mass, in 6 lines to protect the roof. The distance between bolts in line was 1.1 m, and the distance between lines - 0.9 m. The walls were protected by 4 bolts of 1.5-metre length at each 1.1 running metres of the working [43].

Installation costs of the independent roof bolting support in the Mysłowice mine, including labour, material, equipment and transport, were 20.5% lower than the costs of using yielding arches [43].

Chwałowice mine

The gate of Longwall 1, of Seam 403/3 at the Chwałowice mine was located in the western wing of the Chwałowicka Trough in the vicinity of the Michałowicki Overthrust. The seam was at the depth of 430 m. The seam thickness was 1.6 m on average, and an inclination varied from 16° to 29°. It was the headgate used for ventilation purposes and it was liquidated currently at the longwall advance. It was decided to use an independent roof bolting support in it. Physico-mechanical parameters of the rocks surrounding the seam and of the coal were [43]:

– coal laboratory strength to uniaxial compression – R_c = 12.3 MPa,
– penetrometric compressive strength of roof rocks – R_c = 39÷45 MPa,
– laboratory compressive strength of floor rocks – R_c = 35÷38 MPa,
– rock jointing index – *RQD* = 50÷55%,
– soakability coefficient – *r* = 1.

In the independent roof bolting support 361 m of the working, driven with the AM-50 roadheader, were developed. Bolts of 2.6-metre length were installed in the rock mass, in 6 lines to protect the roof. The distance between bolts in the line was 0.84-0.9 m, and the distance between lines - 0.9 m. The walls were

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protected by 3 bolts of 1.9-metre length at each 0.9 running meter of the working [43].

Installation costs of the independent roof bolting support in the Chwałowice mine, including labour, material, equipment and transport, were 19.5% lower than the costs of using yielding arches [43].

Jankowice mine

In Seam 413/1 of the Jankowice mine the headgate was located. It was liquidated currently at the longwall advance. The seam in the area of building the gate in the independent roof bolting support was deposited at the depth of 350 m. Its thickness was 1.2÷2.5 m and its inclination - about 8°-12°. Physico-mechanical parameters of the rocks surrounding the seam and of the coal were [43]:

- coal laboratory strength to uniaxial compression – R_c = 7.9 MPa,
- penetrometric compressive strength of roof rocks – R_c = 21÷27.1 MPa,
- laboratory compressive strength of floor rocks – R_c = 31 MPa,
- rock jointing index – *RQD* = 21%,
- soakability coefficient – $r > 0.8$.

In the independent roof bolting support 1633 m of the gate were driven. The roof was reinforced with bolts of 2.5-metre length, steel mesh and steel linings. The walls were protected with bolts of 1.8-metre length, workable bolts of 1.8-metre length and steel mesh [43].

A reinforcement of crossings in roof bolting support was carried out according to the following scheme [49]:

- an installation of rope bolts of 5-metre length, in the distance up to 2 m, the bolts were installed at the crossing length and 5 m behind and before the crossing,
- a vertical installation of extreme roof bolts from the side of crossing under designing,
- before starting the crossing an installation of two horseheads made of V25 sections at the length of the crossing under designing as well as 2 m behind and before the crossing,
- an installation of two rows of bolts in the branched off working (so called “zero” bolts),

- an installation of rope bolts of 5-metre length and in the distance up to 2 m in the branched off working to the distance of 5 m from the crossing.

Installation costs of the independent roof bolting support in the Jankowice mine, including labour, material, equipment and transport were 23.9% lower than the costs of using yielding arches [43].

Summing up, an application of independent roof bolting support for protecting underground workings in the Polish hard coal mines enabled to obtain satisfactory economic results. Installation costs of the workings, compared to a traditional support made of yielding arches, were reduced by about 20%. Individual kinds of costs were limited by [43]:

- labour costs – 36.5%,
- materials costs – 49.7%,
- equipment – 13.0%,
- transport – 0.8%.

Besides, in the conducted analysis the following additional benefits were not included [43]:

- an improvement of work safety and comfort during an installation of the crossing support,
- an improvement of coal quality achieved due to an elimination of dirt ripping in the roof.

Before starting a project design of the roof bolting support, it was necessary to conduct a detailed analysis of the geological and mining situation. Workings, located in the rocks of adequate strength parameters, where there were no impacts of edges and goaf, were most suitable for an application of the independent roof bolting support [124].

Simultaneously, a manufacture of the independent roof bolting support required an observation of a strict technological discipline. It was indispensable to follow exactly the bolting grid fixed by the designer of the bolting grid and a precise installation of glue charges as well as a location of bolts in the holes to obtain satisfactory technological results. Besides, it was very disadvantageous to execute a web above the size determined in the technical project. It was demonstrated that any deviation from the accepted technology of the support installation might cause a loss of the workings' stability [42, 124].

The experience, gained due to an application of independent roof bolting support, showed a necessity of a good recognition of coal strength parameters and an application of probable stabilization of the walls. Excessive falls of the walls

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caused an excessive, unforeseen in the project, exposure of the roof beam which might cause problems with maintaining the roof stability [124].

Besides, a necessity of conducting current monitoring of the roof bolting support and of the surrounding rock mass behaviour was demonstrated. It was stated that it was unquestionably necessary to conduct a current checkup of the roof stratification, giving an image of efficiency of roof strata fastening. In the case of workings, planned for a long-term use it was shown that there was a need of periodic tests of pulling out of the bolts which enabled to determine a condition of rock strata in the vicinity of the bolt glued into the rock mass. Conducting of current monitoring enabled to determine a correctness of the support project design, thus enabling to recognize changes occurring in the surrounding rock mass and an application of possible corrective measures, improving safety of activities conducted in the working [42, 124].

5. Application of *Continuous Miner* type machines

Artur Dyczko[1], Paweł Kamiński[2]

The first machine of *Continuous Miner* type was introduced to the market of mining machines by the JOY Company in 1948. For the following decades the machines, produced by JOY, have been improved and modernized and the Company itself is up till now the biggest producer of this type machines in the world [125].

A drive of a typical *Continuous Miner* consists of two motors. The following two motors are responsible for a rotation of the cutting head, one or two form a drive for the loader (as an alternative, instead of the loader, a loading wedge can be used), one drives the hydraulic pump and the following one has a drive function for the dust collector (if the machine is equipped with it) [125].

Due to a correct designing of cutting picks location on the cutting drum, of cutting pick holders as well as of the cutting picks as such, an optimization of the forces‘ distribution is ensured. The feedback between the motors driving the cutting drum and the machine driving system enables on optimum use of the machine capabilities without any overloads which has a positive impact on its reliability and life [125].

The most modern machines of *Continuous Miner* type, apart from the standard equipment mentioned above, are additionally furnished with dust control devices, a cooling and spraying system of the cutting drum, a haulage system and remote control systems. A protection of the exposed roof is realized immediately by hydraulic props and the roof bolting system consisting of hydraulic bolters for roof bolting and additionally for bolting the walls – Bolter Miner machines.

It is planned that an introduction of Bolter Miner machines in the Polish hard coal mining industry will enable a fast driving of development workings for exploitational longwalls and a significant reduction of their realization costs. Additionally, after having gained the appropriate experience, Bolter Miner machines will be applied in the conditions, when mining of longwalls is inefficient economically or technically impossible e.g. due to a necessity of area surface protection [125].

Machines of *Continuous Miner* type are widely used in the world hard coal mining industry for conducting development and exploitational operations. An exploitation with use of this type machines is usually carried out in a shortwall system or in a room-and-pillar system. During exploitational operations a haulage

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of the run-of-mine is most often realized with use of an assembly of haulage cars of *Shuttle Car* type equipped with a scraper conveyor used for unloading the run-of-mine from the car load-carrying body.

The most important advantages and disadvantages of the *Continuous Miner* machines in comparison with other cutting methods are presented in Table 5.1.

List of advantages and disadvantages of respective cutting

Table 5.1.

Development technology	Advantages	Disadvantages
Continuous Miner cutting linearly	High mobility	High unit pressure on the floor
	High speed of machine dislocation	A possibility of driving workings only in a rectangular cross-section
	Maximum cutting power calculated for 1m^2 of the face surface	Difficulties with cutting rock of compressive strength exceeding 40 MPa
Roadheader cutting pointwise	Optional shape of the working transverse cross-section	Convenient for mining thick and medium coal seams
	Ability of cutting rock of compressive strength exceeding 40 MPa	
	Maximum cutting power calculated for 1m^2 of contact surface of the cutting head with rock mass	
MW blasting method	The biggest scope of applications	Uneven circumference of driven working
	Possibility of cutting rock of big and very big compactness	A generation of cracked zone around the working caused by blasting operations
	Optional shape of the working transverse cross-section	Lack of the process continuity

An application of *Continuous Miner* machines in the world hard coal mining industry is presented on examples of South-African, Australian, British and Czech mines.

South Africa

A use of heavy machines of *Continuous Miner* type, adapted for cutting very hard coal in high seams, occurring in South-African hard coal mines, enabled to increase productivity and decrease costs in relation to a conventional blasting method applied before. South-African coal seams belong to the seams of big thickness (more than a half of them is above 3-metre thick) and an application of shortwall systems with an extraction of pillars turned out to be more flexible and more efficient in those conditions (a use of the seam reached even 95%) in comparison with longwall systems [125].

Australia

In Australia machines of *Continuous Miner* type are used as mining machines in the room-and-pillar system and as machines driving development workings in longwall systems. Big longwall advance forced producers of mining machines to elaborate a project design of a heading machine capable of keeping pace with the longwall advance. To meet this requirement a series of machines of *Continuous Miner* type was developed and equipped with bolters for a simultaneous cutting and bolting – machines of Bolter Miner type. The machines: JOY 12CM20, JOY 12 CM30 and Voest Alpine ABM20 [125] belong to this group.

United Kingdom

In the nineties of the XX century British hard coal mines experienced a period of big and difficult changes. Due to a big pressure on a reduction of production costs, the methods used successfully in the United States, Australia and in the Republic of South Africa started to be implemented. A rectangular cross-section of development workings, driven with use of *Continuous Miner* machines and independent roof bolting support with bolts glued at the whole hole length, were introduced. Roof bolting support was also used in the mines, which before had been considered to be too deep for its application. An introduction of roof bolting support improved roof conditions to a big extent, and in many mines a problem of the floor up-heaving ceased to occur. Besides, advantageous economic results were achieved [125].

Czechia

In 2014, in the Northern Plant of the Czech CSM mine, at present called DZ2, OKD a.s. an experimental exploitation of the seam, modified to comply with the mine needs, was started with use of the room-and-pillar system. The exploitation was continued so that to verify possibilities of applying this method for an extraction of the deposit residual parts from protective pillars, drifts and shafts, at a simultaneous impact minimization of operations on the area surface [34].

An experimental exploitation was conducted in Seam No. 30 (Polish equivalent: 418/2). It was classified as a seam with no bumping hazards. The seam was deposited at the depth of about 850 m and its thickness varied from 2.5 to 5.0 m. The seam inclined towards the eastern direction, at a local inclination reaching 20°. Besides, local tectonic dislocations occurred, e.g. an overfault of 5-metre thrust. The tests, conducted in testing holes, demonstrated an occurrence

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of very good roof conditions and based on the RMR index acc, to Bieniowski, it was stated that properties of roof rocks enabled to apply independent roof bolting support [34].

In connection with positive results of the geological survey it was decided to start an exploitation in the protective pillar of the shaft in the Northern Plant. Nevertheless, a classical room-and-pillar system had to be subject to the following modifications [34]:

- it was decided not to part down technological pillars due to a necessity of protecting the surface and shafts,
- the dimensions of pillars were increased in relation to calculated dimensions to ensure a stability of the pillars,
- a multi-task cutting-and-bolting machine of Bolter Miner type was used instead of a *Continuous Miner* machine.

The JOY 12CM30 machine was chosen for cutting the face and bolting the roof as well as the walls. A haulage of the run-of-mine was carried out by a self-propelled haulage car of the Shuttle Car 21SC04 type supplied with an electric cable wound on a drum. The car transported the run-of-mine to the discharge-and-crushing device of the *Feeder Breaker* type, from which the crushed material was transported to the district coal bunker with use of belt conveyors system. The *Feeder Breaker* device was advanced step by step following the front advance and it was synchronized with the range of *Shuttle Cars*. Additionally, the Alminco Scorpion drill-rig was a supplementary machine, which executed special tasks e.g. drilling of bolt holes to strengthen crossings or in the case of deteriorated geological conditions [34].

The workings were conducted in independent roof bolting support. Steel bolts of 21.7 mm dia., 300 kN of load-bearing capacity and of the length 2.4 m or 2.8 m were used – depending on the conditions. The bolts were glued at the hole total length and installed together with steel mesh lining. Cuttable bolts were used for protecting walls in the places of future crossings. The roof was protected with 6 or 8 steel bolts in relation to mining-and-geological conditions, the walls – with 6 steel bolts, additionally when it turned out to be needed rope bolts of 6-metre length were used [34].

Two- or three-level stratification meters, located at the distance of maximum 20 m, were applied for monitoring independent roof bolting support. The stratification meters were installed at the height of 2 and 5 m (two-level ones) or 2.5 and 7 m (three-level ones). In fact 5% of the height under observation were regarded to be a critical stratification. Depending on the place of the critical stratification occurrence, it was decided about concentrating the bolting grid (a low stratification), about increasing the length of bolts (a medium stratification) or about replacing the roof bolting support with standing support (a high stratification). The critical value for convergence measurements was accepted as 10% of the working dimensions. To check a load-carrying capacity of bolts in-situ tests were performed, so called pull-tests, where the force combining the bolt, glue and rock mass was checked. For a current observation of bolt loading indicatory properties of a washer were used – straightening and bulging at given values of an axial force in a bolt [34].

Within a year, since September 2014 till September 2015, in the framework of an experimental exploitation 2197 m of the working in the independent roof bolting support were driven. It was the first trial of applying the Bolter Miner machine in difficult conditions of Górnośląskie Zagłębie Węglowe (Upper Silesian Coal Basin). No serious problems were experienced during an operation of the independent roof bolting support although an overfault was passed twice. An analysis of costs showed moderately advantageous results and the achieved technical-and-economical results indicated an increasing efficiency of driving workings with use of a Bolter Miner machine [34].

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6. Project of implementing independent roof bolting support in roadway workings at the mines of Jastrzębska Spółka Węglowa S.A. (Jastrzębska Coal Company J.S.C.)

Stanisław Prusek[1], Sylwester Rajwa[1]

Between July 2017 and March 2018 in the mines of the Jastrzębska Spółka Węglowa (JSW S.A.) 90 km of roadway workings were developed. Total costs of driving them reached about 1.1 billion PLN. In individual mines of JSW S.A. the percentage share of roadway working drivage with use of roadheaders was different and varied from 28% in the Zofiówka mine to 94% in the Budryk mine. The total length of roadway workings in the individual mines of the JSW S.A., during the period under discussion, is presented in Table 6.1 as well as in Fig. 6.1 and 6.2.

Length of roadway workings in running metres in the individual mines of the JSW S.A. in relation to the driving method over the years 2017-2018

Table 6.1.

Mine	**Length of roadway workings [rm]**		
	Mechanical cutting	**Mechanical cutting and blasting**	**In total**
Borynia	6873.0	6073.5	12946.5
Jastrzębie	3866.0	1329.0	5195.0
Zofiówka	3056.0	7780.0	10836.0
Budryk	13088.0	766.9	13854.9
Knurów-Szczygłowice	25910.0	1901.0	27811.0
Pniówek	18014.5	1371.5	19386.0

[1] Central Mining Institute

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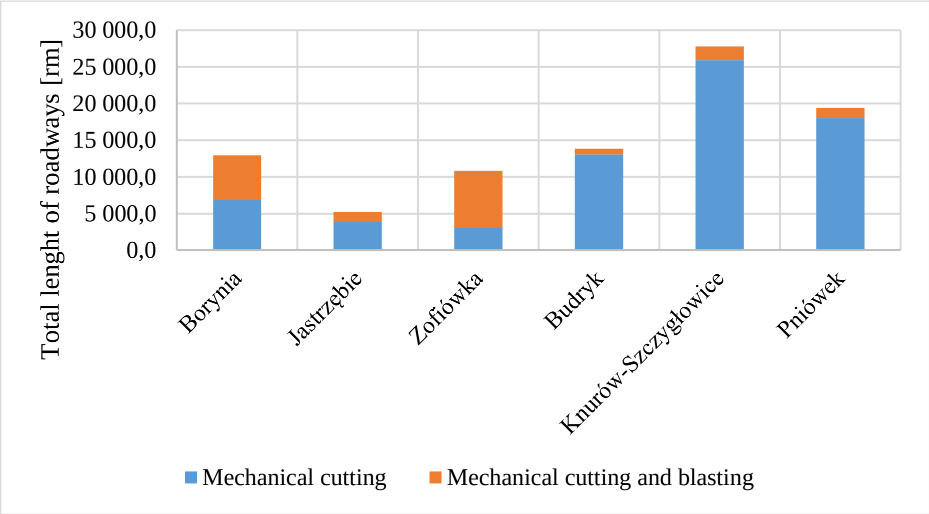

Fig 6.1. Length of roadway workings in running meters in the individual mines of the JSW S.A. in relation to driving systems over the years 2017-2018

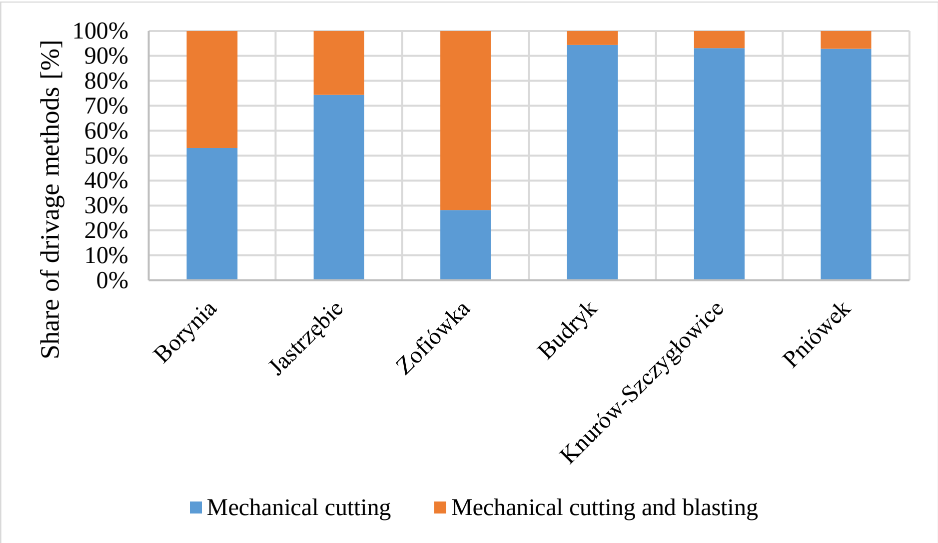

Fig. 6.2. Percentage share of roadway workings length in individual mines of the JSW S.A. over the years 2017-2018

Percentage shares of the workings drivage methods in all the mines of the JSW S.A., in the period under discussion, are shown in Fig. 6.3.

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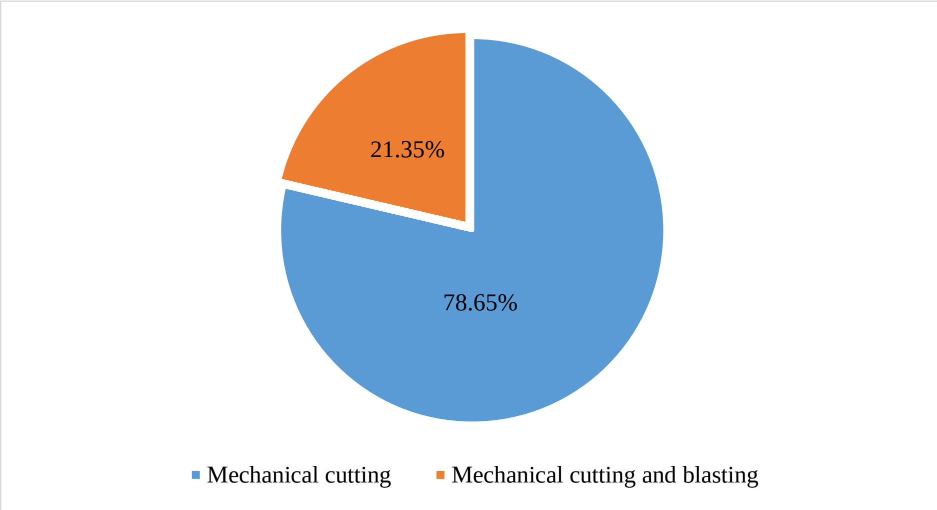

Fig. 6.3. Percentage shares of the workings drivage methods in all the mines of the JSW S.A. in the period 2017-2018

In individual mines of the JSW S.A. an advance of workings driven with use of roadheaders, during the period under discussion, was differentiated. It was comprised in the range 4.2 – 7.5 m (Fig. 6.4). The smallest twenty-four-hour advance was achieved in the Zofiówka mine, but the biggest one – in the Knurów-Szczygłowice mine. The advance, obtained during the period under discussion, can be determined as "below expectations".

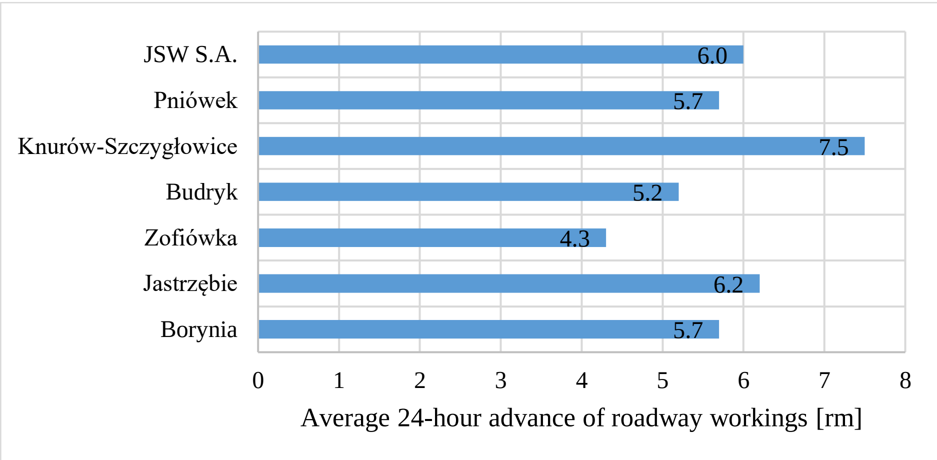

Fig. 6.4. Average twenty-four-hour advance of roadway workings in the mines of the JSW S.A. over the years 2017-2018

To increase the twenty-four-hour advance of operations, it was decided to undertake a trial of implementing independent roof bolting support in the

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roadway workings, driven with use of Bolter Miner machines. After having conducted a complex analysis of Bolter Miner machines available on the market and of technical possibilities of using the machines of this type in the mines of the JSW S.A., special attention was paid to the following machines:

– JOY 12CM30 made by Komatsu JOY,

– MB 670-1 FLP made by Sandvik.

Basic technical parameters of both machines are compared in Table 6.2.

Basic technical parameters of the JOY 12CM30 and MB 670-1 FLP machines [126-127]

Table 6.2.

Parameter	Unit	JOY 12CM30	MB 670-1 FLP
Weight	kg	96000	105000
Cutting width	mm	4800÷5600	5000÷6200
Cutting height	mm	3000÷4900	2800÷5000
Number of roof bolters	pcs.	4	4
Number of wall bolters	pcs.	2	2
Minimal roof opening before web	mm	1000	1900
Power of cutting head drive	kW	340	270
Total power	kW	610	546

Finally, the decisive factor as regards a selection of the machine was a minimal opening of the roof space. At the stage of the pilot research project, to minimize a risk connected with the work organization, it was decided to base the project activities on the JOY 12CM30 machine. The machine is presented in Fig. 6.5.

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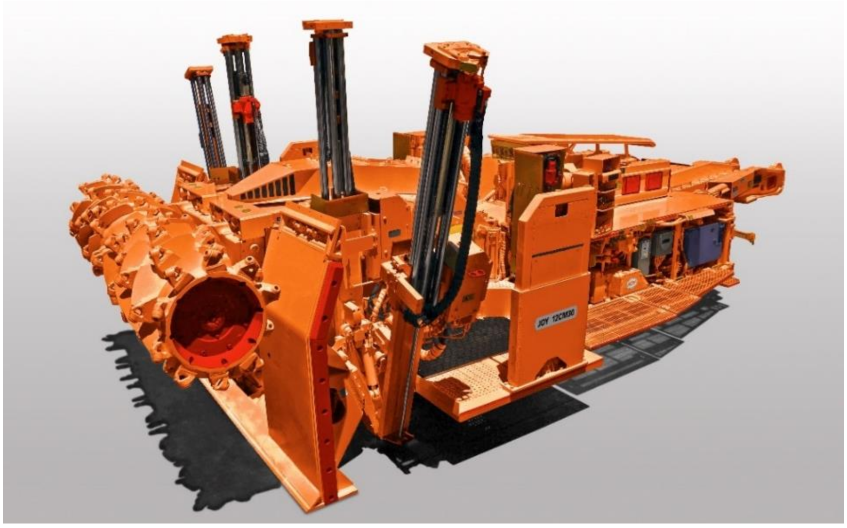

Fig 6.5. JOY 12CM30 machine

A calculation of the bolter miner machine operations was performed in the JSW S.A. mines to verify implementation effects of the independent roof bolting support. The duration of a work cycle is a function of a number and length of bolts which should be installed after each roof exposure. The following variants were subject to analyses:

– variant 1 – 8 roof bolts and 6 wall bolts of 2.6-metre length,
– variant 2 – 6 roof bolts (length: 2.6 m) and 6 wall bolts (length: 1.8 m),
– variant 3 – 6 roof bolts and 6 wall bolts of 2.6-metre length,
– variant 4 – 8 roof bolts (length: 2.6 m) and 6 wall bolts (length: 1.8 m).

The results of analyses are listed in Table 6.3.

Analysis of work cycle of the Bolter Miner machine

Table 6.3.

Cycle operations	Unit	Variant 1	Variant 2	Variant 3	Variant 4
Cutting of 1 rm	min	3.65	3.65	3.65	3.65
Mesh installation		4.17	4.17	4.17	4.17
Installation of roof bolts		15.87	11.90	11.90	15.87
Installation of wall bolts		19.40	13.65	15.40	17.70
Out-of-parallel termination		1.50	1.50	1.50	1.50
Installation of technological bolts (2.6 m) every 0.5 m		4.00	4.00	4.00	4.00
Cycle total time		48.58	38.87	40.62	46.88

In the case of variant 1 (14 bolts) the twenty-four-hour advance was 15-30 rm. Even assuming a conservative approach to the time use at the level of 50%, it was calculated that an advance, obtained with of use the bolter miner machine, should be bigger than ~ 98% of advance rates in the mines of the JSW S.A. Thus it has been stated that an implementation of the project “Independent Roof Bolting Support” will give the Company measurable benefits, causing a costs reduction of development activities at a simultaneous increase of advance and also a faster preparation of exploitational fronts. Simultaneously in the case of an application failure of the roof bolting technology for openings and gate workings there is a big potential of its use in the workings for an extraction of the deposit residues, which cannot be mined with a longwall system due to restrictions connected with an impact of mining activities on the terrain surface.

As part of the implementation of the project, Jastrzębska Spółka Węglowa applied to the Division of Exploitational Technology and Mining Supports of the Główny Instytut Górnictwa to conduct research work concerning possibilities of implementing independent roof bolting support in the JSW S.A. mines. In the framework of the research work an analysis of the possibilities of applying independent roof bolting technology in the scope of opening and development operations and also winning operations for extracting residual parts of the deposit was conducted. Based on the analysis it was decided to apply a Bolter Miner machine for the Bw-1n test roadway drivage at the Budryk mine.

The JOY 12CM30 machine was produced in the United States, transported to Poland by sea and then assembled in the Plant in Tychy (Fig. 6.6, 6.7 and 6.8), where it was subject to indispensable commissioning. Afterwards the machine was disassembled and transported to the underground part of the Budryk mine, to the area of the Bw-1n test roadway, where it was assembled again in the assembly chamber prepared for a realization of this task. The chamber was simultaneously the initial segment of the Bw-1n test roadway.

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Fig. 6.6. JOY 12CM30 machine in the Komatsu Plant in Tychy – cutting head

Fig. 6.7. JOY 12CM30 machine in the Komatsu Plant in Tychy – stage loader

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Fig. 6.8. JOY 12CM30 machine in the Komatsu Plant in Tychy – control system

A design of the independent roof bolting support, suitable for mining and geological conditions of the Bw-1n test roadway, was elaborated by the Central Mining Institute on the basis of the analytical-and-empirical as well as numerical modelling methods. The preliminary design was made for the working of dimensions: 5.4 x 3.4 m, nevertheless the final overall dimensions of the Bw-1n test roadway were 5.6 x 3.4 m. Besides, during the drivage a possibility of driving the working of bigger width and height was tested. The first segments of the working, under discussion, were driven with use of combined supports: standing support and roof bolting support, thus enabling a smooth start of the system. After having reached the mark of ~40 rm, a drivage of the working in independent roof bolting support was started, however an auxiliary standing support was used exclusively in the area of bends.

The contract for conducting operations in the working was concluded with the Company Przedsiębiorstwo Budowy Szybów (PBSz) S.A. (Enterprise of Shafts Construction), which established a new division for the project realization, consisting of 113 workers, including 100 physical workers and 13 supervisory persons [128]. Besides, the Company Sigma S.A., was engaged in the project, holding responsibility for a part of the equipment concerning the panel transport and the Company Hargreaves Mining was responsible for training the workers of the PBSz S.A. during the initial months of the project. Interrelations among the companies, engaged in the project of implementing independent roof bolting support, are shown in Fig. 6.9.

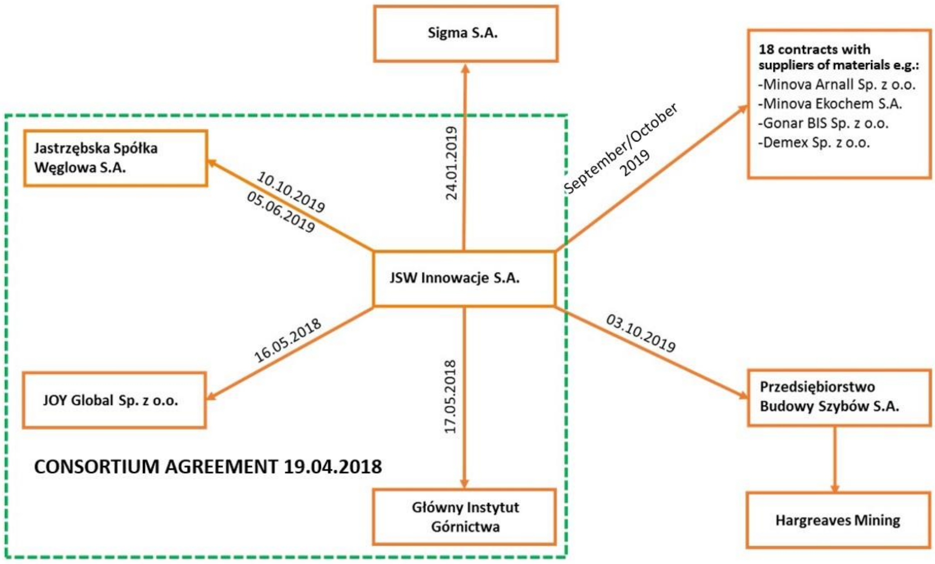

Fig.6.9. Companies engaged in the implementation project of independent roof bolting support [128]

Each of the companies, presented in Fig. 6.9, had a specific role in the project [129]:

- **JSW Innowacje S.A.** – a project coordination, scientific-and-research work, an organization of the machine delivery as well as of the face and outside the face devices according to the guidelines of the Project Team, an organization of input and exploitational materials' delivery as well as measuring instruments,
- **JSW S.A. Biuro Zarządu (Board Office)** – a project supervision,
- **JSW S.A. Budryk mine** – a supervision of the project according to operational regulations in force and the Act: Prawo Geologiczne i Górnicze (Geological and Mining Law),
- **PBSz S.A.** – drivage and conducting other operations in the working,
- **Hargreaves Mining** – training of the staff during the project initial months,
- **JOY Global sp. z. o. o.** – a machine delivery, a supervision of the mechanical part,
- **SIGMA S.A.** – a delivery of machines and equipment for use outside the face together with the required infrastructure,
- **Główny Instytut Górnictwa** – services of experts, research and design work.

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The most important organizational activities, undertaken within the framework of preparations for the project realization, are presented on the time axis (Fig. 6.10).

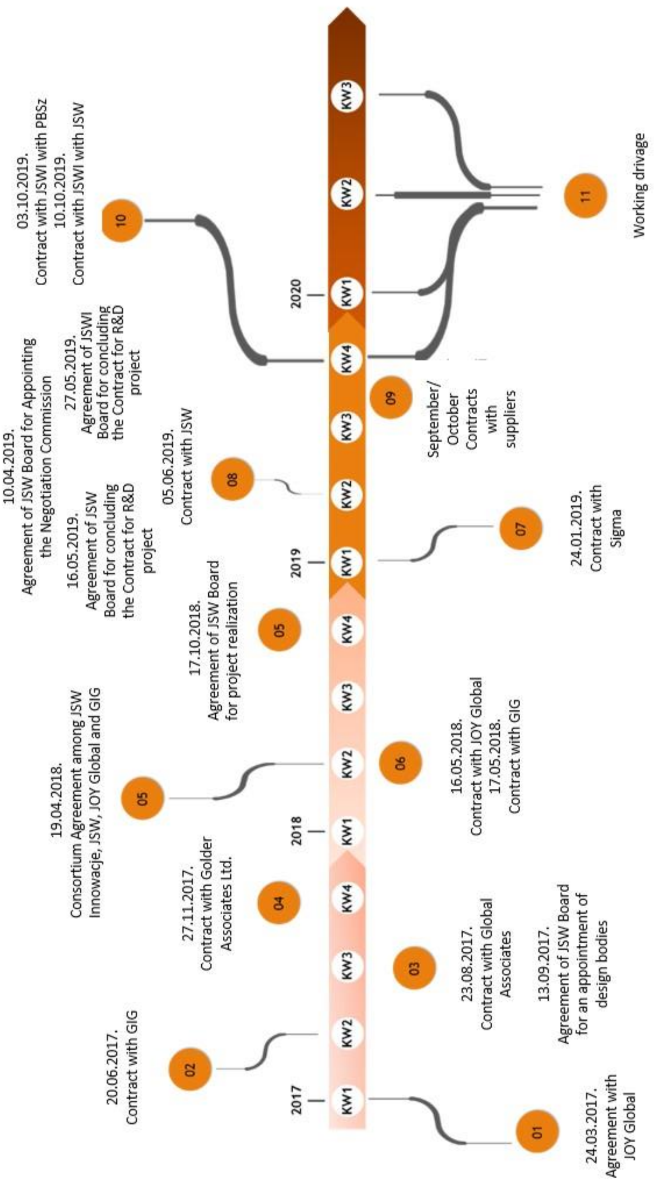

Fig.6.10. Time axis of activities conducted within the framework of preparations for the project realization of implementing independent roof bolting support [129]

6.1. Description of JOY 12CM30 machine

The JOY 12CM30 made by Komatsu Mining Corp., is a highly productive machine for use in coal seams of medium thickness. The machine is driven with use of electric energy of voltage 1000 V/50 Hz. The total power of electric motors and other equipment is 620 kW. A control of the machine is performed with a radio controller with a possibility of connecting it to the machine with a cable of 15-metre length. Nevertheless some functions of the machine can also be controlled directly from the control panel [130].

The machine can be used for cutting rock of the compressive strength which does not exceed 85 MPa, at the transverse inclination of the driven working up to ±5° and the longitudinal inclination up to ±15°. The machine is constructed from the following subassemblies [130-131]:

- cutting head,
- cutting head arm,
- cutting head supply system,
- central scraper conveyor with deflective end,
- air-and-water spraying system,
- dust controller,
- methanometric system,
- hydraulic system,
- electric equipment including control system,
- 2 drilling-and-bolting wall units,
- 4 drilling-and-bolting roof units,
- working platforms.

The main machine subassemblies are presented in Fig. 6.11. The most important machine parameters are listed in Table 6.4.

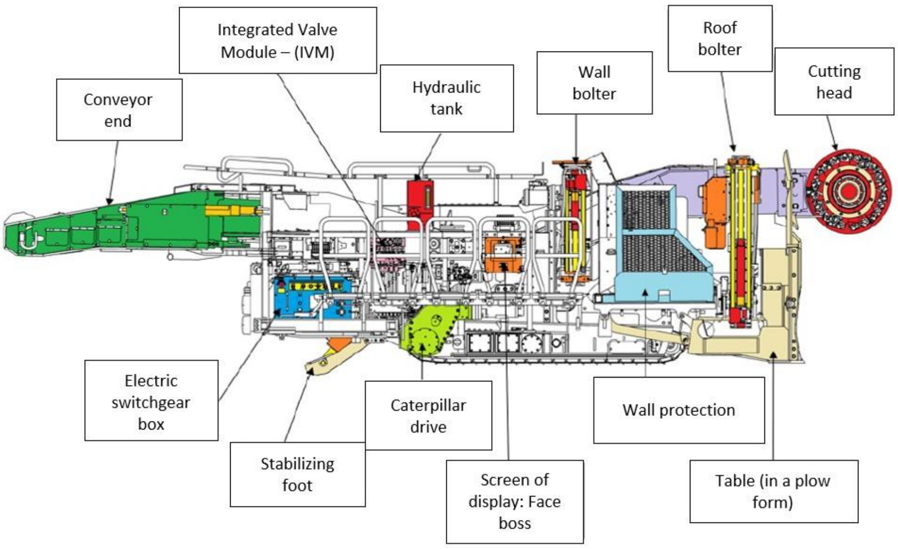

Fig. 6.11. Main subassemblies of JOY 12CM30 machine [12]

Most important technical parameters of JOY 12CM30 roadway machine

Table 6.4.

Parametr	Unit	Value
Length	mm	11 700
Width	mm	3 430
Height	mm	2 680
Cutting width	mm	4 800÷5 600
Cutting height	mm	3 000÷4 900
Minimum roof opening before web	mm	1 000
Number of roof bolters	pcs.	4
Number of wall bolters	pcs.	2
Torque of roof drill	Nm	345
Torque of wall drill	Nm	270
Supply voltage	V	1000
Drive power of cutting head	kW	340
Total power installed in the machine	kW	610
Total weight of the machine	Mg	96
Maximum compressive strength of rock under cutting	MPa	85
Maximum working longitudinal inclination	°	±15

Cutting system

The cutting head of 1120 mm dia. is supplied with two AC motors of 170 kW power, protected by couplings. The motors are installed on two sides, parallelly to the cutting head arm axis and they are connected to gearboxes with use of internal shafts. During the machine dislocation, to reduce its dimensions, it is possible to slip the cutting head. Besides, there is a possibility of widening the cutting head to increase the cutting surface. The working tools in a form of cutting picks are installed in holders of JOY J30 type with forced in bushes [130].

Transport system

The machine internal transport system is composed of a plow and the main scraper conveyor. The run-of-mine is racked to the scraper conveyor with worm wheels of the cutting head. The conveyor end can be deflected to the left or to the right to check proper loading of the run-of-mine on the haulage means. The main scraper conveyor chain is driven by the motor installed in its main frame. The width of the conveyor line pan is 760 mm [130].

Haulage system

The haulage system is installed on both sides of the machine, in the frames of caterpillars. Both caterpillars are driven by a motor of 37 kW power and a gear. Each haulage drive is equipped with the parking brake, released hydraulically, installed directly on each gear [130].

Hydraulic system

All the hydraulic functions of the machine are controlled from the remote controller, there is no possibility of a manual control of the valves. The machine is equipped with two separate hydraulic systems [130]:

- hydraulic system of the right side supplies all the functions of a given mode and the right side drills,
- hydraulic system of the left side supplies the left side drills.

Each hydraulic system is supplied with variable delivery pumps, driven by motors of 45 kW power [130].

Water system

The machine water system realizes spraying and motor’s cooling functions. The volume of water flow is controlled by flow sensors installed in the motor cooling circuits. In the case of detecting insufficient flow, the corresponding motors are switched off automatically and the corresponding message is displayed on the alarm list [130].

Control system

A control of the machine operation is realized with use of the remote control controller having a guaranteed radio range of about 100 m. The remote control system serves for controlling all the functions of the machine: start, haulage, cutting and rotation as well as for controlling the functions of drills and bolters. In the case of a break-down of the control system, the supply is switched off automatically and the machine stops. Corresponding blocking systems ensure that the sequence of a start-up operation does not cause dangerous operational conditions. For example, the spraying function must be switched on before starting the cutting head [130].

An application of remote control improves the staff operational safety to a significant extent. The operator works under the protected roof all the time and if he only observes the regulations concerning access zones, he is not subject to any hazard caused by a fall of roof rocks [130].

7. Methodology of conducting analyses related to independent roof bolting support in the mines of Jastrzębska Spółka Węglowa S.A.

Wojciech Masny[1]

The first stage of work, connected with an introduction of independent bolting support in the mines of the JSW S.A., consisted in choosing opening and development workings as well as residual panels which meet the requirements related to an application of independent roof bolting support or a Bolter Miner machine [132].

The following disqualifying criteria were distinguished [132]:

– short drivage segments,
– hazard to operations' time-scale – a long time needed for a purchase and assembly of the machine and equipment,
– a necessity of maintaining workings behind the face front in the case of increased loads e.g. in the case of the "Y" ventilation system,
– difficult geological conditions:
  - faults,
  - excessive outflows of water from the rock mass,
  - water wells,
  - effluents,
– difficult mining situation:
  - vicinity of gob or exploitation under the gob,
  - edges and residues occurring in the distance smaller than 50 m, in particular overlapping ones (an additional analysis is required),
  - passage of a driven face through the roadway workings out of operation,
  - inclination above 18-20°.

According to the regulations included in the Minister's of Energy Order from 23rd November 2016, Official Gazette No. 2017, Item 1118 [133]:

§ 125.1. An application of independent roof bolting support in mining plants extracting hard coal is permissible only in the case when:

1) roof rocks have an average weighed strength to uniaxial compression (R_c), tested for a packet of rocks 3 m in thickness, being not smaller than:
   a) 15 MPa – for layers of panel structure and measured crevice of roof rocks (RQD), being not less than 20%,

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[1] Central Mining Institute

b) 10 MPa – for layers of solid structure and measured crevice of roof rocks (RQD), being not less than 40%;

2) the rock mass is dry or non-soaking and the soakability coefficient (*r*) is not less than 0.8;
3) it is used for protecting roadway and chamber workings of the transverse section area not exceeding 30 m^2 and the working width not exceeding 7 m.

The workings and panels, for which the above data were available, were assessed additionally also from the point of view of their compliance with legal regulations.

A card with a list of basic information – Table 7.1 was filled in for each case subject to drawing up inventory.

Card A containing basic geological-and-mining information together with an assessment of possibilities of using independent roof bolting support [132]

Table 7.1.

Part	A	Card No.	Axx
Mine			
Seam			
Batch/O.G.			
Depth [m]			
Sean thickness [m]			
Assessment of possibilities of using independent roof bolting support			

In the case of a positive assessment of possibilities enabling to apply an independent roof support in Card A, the data are supplemented with detailed information in Card B – Table 7.2.

Card B – additional information – opening and development workings as well as residual panels meeting the requirements concerning an application of the Bolter Miner machine of independent roof bolting support [132]

Table 7.2.

Part	B	Card No.	Bxx
Mine			
Seam			
O.G./Batch			

Compressive strength [MPa]	Roof	
	Coal	
	Floor	
Core divisibility index *RQD* [%]		
Soakability coefficient *r*		
Geological profile		
Natural hazard	Bumps	
	Water	
	Methane	
	Coal dust Explosion	
Seams situated above/below (vertical distances)		
Inclination		
Tectonic dislocations		
Length of workings in independent roof bolting support [m]		
Is there an opening-up?		
Mining technology		

7.1. Opening and development workings meeting the requirements in the scope of applying independent roof bolting support

In the result of made inventory of workings, in which it was possible to apply independent roof bolting support and of an analysis of the time scale of opening and development operations, it was decided to apply the machine of Bolter Miner type and independent roof bolting support in Seam 401 at the Budryk mine. The conditions in the panel are listed in Table 7.3 [132].

Card A7 – opening and development workings meeting the requirements for an application of Bolter Miner machine and of independent roof bolting support [132]

Table 7.3.

Part	A	Card No.	A7
Mine	Budryk		
Seam	401		
Batch/O.G.	Ornontowice I		

Depth [m]	890-960
Seam thickness [m]	0.90-1.80
Assessment of possibilities of applying independent roof bolting support	Opened regular part of Seam 401. According to the present state of knowledge no significant tectonic dislocations are predicted. A possibility of optional shaping of this part of the seam development both from the point of view of longwall mining as well as roadway workings.

The Budryk mine did not have a sufficient recognition of the Seam 401, in the area of the suggested use of the machine of Bolter Miner type and of independent roof bolting support. In agreement with the Główny Instytut Górnictwa it was decided to conduct an exact recognition of mining-and-geological conditions in the area performing additional penetrometric tests [312].

The tests were conducted in two measuring bases [132]:

– in the Bw test incline – Gp17,

– in the B-1 test incline – Gp18.

The measuring base consisted of [132]:

– a vertical hole in the roof,

– a skew hole in the roof,

– a hole in the floor,

– a hole in the wall.

In the measuring bases the tests and measurements were conducted for [132]:

– a determination of the compressive strength of roof, floor rocks and of coal,

– a determination of fracturing zone range around the working,

– taking a core and a determination of lithology as well as of the crevice index *RQD*,

– taking samples from the core to determine the soakability coefficient *r*.

Tests of compressive strength were conducted with use of a hydraulic apertured penetrometer. The results of the measurement test, conducted with the penetrometer in the GP17 hole in the B-1 test incline, are presented in Table 7.4 [132].

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List of results obtained due to using hydraulic apertured penetrometer in the Gp17 hole in the B-1 test incline [132]

Table 7.4

Location	**Hole [m]**	**Compressive strength R_c [MPa]**	**Tensile strength R_r [MPa]**	**Crevice index RQD_p [%]**
Roof	up to 3.0	45.66	2.93	70
	0.0-8.0	42.52	2.73	61
Wall	0.0-3.0	13.82	0.89	-
Floor	0.0-2.2	25.00	1.60	54

Based on testing rock samples taken from the core, the soakability coefficient *r* was determined for all the three samples after 24, 48 and 72 hours; the samples did not change either their form or consistence, so it could be stated that the r coefficient was 1.0 [132].

The results of all the measurements taken in the Bw test incline, aimed at an assessment of possibilities of applying an independent roof bolting support, are presented in Table 7.5.

Summary of test results conducted in the Bw test incline together with an assessment of possibilities of using independent roof bolting support [132]

Table 7.5.

Parameter		**Value**	**Assessment**
R_c roof [MPa]	0-3 m	45.7	positive
	0-8 m	42.5	positive
R_c coal [MPa]		13.8	positive
R_c floor [MPa]		25.0	positive
Soakability [-]		1.0	positive
RQD [%]	0-3 m	43	positive
	0-8 m	38	positive

Based on the results of tests conducted in the Bw test incline, it can be stated that the parameters under analysis are in accordance with the regulations and they do not exclude an application of independent roof bolting support [132-133].

The results of tests, conducted with use of a hydraulic apertured penetrometer in the Gp18 hole in the B-1 test incline, are listed in Table 7.6.

 https://doi.org/10.32056/KOMAG/Monograph2021.2

List of results obtained with use of a hydraulic apertured penetrometer in the Gp18 hole in the B-1 test incline [132]

Tale 7.6.

Location	**Hole [m]**	**Compressive strength R_c [MPa]**	**Tensile strength R_r [MPa]**	**Crevice index RQD_p [%]**
Roof	up to 3.0	33.03	2.12	53
	0.0-7.7	42.25	2.71	61
Wall	0.0-2.9	8.97	0.58	-
Floor	0.0-1.5	10.82	0.69	57

Based on rock samples, taken from the core, the soakability coefficient *r* was determined. For all the three samples after 24, 48 and 72 hours the samples did not change either form or consistence, so it can be stated that the r coefficient was 1.0 [132].

The results of all the tests, performed in the Gp18 hole in the B-1 test incline, aimed at an assessment of possibilities of using independent roof bolting support, are presented in Table 7.7.

Summary of test results obtained in the B-1 test incline together with an assessment of possibilities of applying independent roof bolting support [132]

Table 7.7.

Parameter		**Value**	**Assessment**
R_c roof [MPa]	0-3 m	33.0	positive
	0-7.7 m	42.3	positive
R_c coal [MPa]		9.0	positive
R_c floor [MPa]		10.8	positive
Soakability [-]		1.0	positive
RQD [%]	0-3 m	30	positive
	0-8 m	35	positive

Based on the obtained results of tests, conducted in the B-1 test incline, it can be stated that the parameters under analysis are in accordance with the regulations and they do not exclude an application of independent roof bolting support [132].

In relation to an explicitly positive assessment of the Seam 401 parameters, it was decided to apply a machine of the Bolter Miner type and independent roof bolting support for driving the Bw-1n test roadway in the Seam 401 at the Budryk mine.

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8. Mining and geological conditions in the area of the Bw-1n test roadway in the Seam 401

Paweł Kamiński [1,2], Wojciech Masny[3], Rafał Pierszalik[3]

The Bw-1n test roadway is situated in the north-eastern part of Batch B of the Seam 401 O.G. "Ornontowice I". The working was started in the place of haulage B Roadway crossing with the B-1 test incline in the Seam 401. The working drivage consisted of the following phases (figure 8.1) [129, 134]:

- **Phase I** – an execution of an assembly chamber of the Bolter Miner machine of 30-metre length and of a straight segment of 200-metre length in the north-eastern direction.
- **Phase II** – an execution of the bend No.1 at the angle of 135°, changing the roadway driving direction from north-eastern to eastern one.
- **Phase III** – an execution of the straight segment of 780-metre length in the eastern direction (after having reached the mark 1044.1 rm, the drivage was stopped due to technical and technological reasons and an execution of the roadway junction was started on the mark of 994 rm),
- **Phase IV** – an execution of the junction (bend No. 2) at the angle 146°, changing the roadway drivage direction from eastern to south-eastern one.

The panel under discussion was recognized due to the B-1 test incline and the Bw-1n test incline. Besides, in the area of designed mining operations the following test holes: G16(2006), G25(2006), G19(2006), G.Ch7(1959) – were drilled from the surface. It is predicted that the seam thickness, in the panel under discussion, is 0.90 ÷1.80 m (in the hole G.Ch.7 the seam together with 1.0-metre of shale clay dirt band has the thickness of 3.0 m) [129].

In general, the Seam 401 subsides towards the southern-east at the angle of 5÷9°. At the present state of the panel recognition, it is not possible to exclude an occurrence of inclinations of bigger values at the length of the designed working [129].

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Fig. 8.1. Bw-1 test roadway drivage [129, 134]

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8.1. Roof and floor conditions

Roof and floor conditions, based on the data from the holes: G16(2006) and G25(2006), are listed in Tables 8.1 and 8.2.

Roof and floor conditions in the panel of the Bw-1n test roadway based on the data from the G16(2006) borehole [129]

Table 8.1.

G16(2006) borehole		
Location	Rock	Thickness [m]
Roof	coal (Seam 364/2)	1.60
	shale clay	1.5
	arenaceous shale	5.5
	shale clay	3.0
	medium and thick-grained sandstone	4.7
	shale clay	0.65
	coal (Seam 364/3)	0.95
	shale clay with interlayers of arenaceous shale	10.80
	coal (Seam 364/4)	0.9
	shale clay	1.7
	fine-grained sandstone with interlayers of arenaceous shale	1.3
	shale clay	4.0
	fine-grained sandstone with interlayers of arenaceous shale	0.70
	shale clay	2.55
Seam 401		
Floor	shale clay with interlayers of arenaceous shale	6.35
	fine-grained sandstone	13.3
	shale clay	0.5
	coal	0.5
	shale clay	5.6
	coal (Seam 402)	3.7

Roof and floor conditions in the panel of the Bw-1n test roadway based on the data from the G25(2006) borehole [129]

Table 8.2.

G25(2006) borehole		
Location	Rock	Thickness [m]
Roof	coal (364/2)	1.45
	shale clay	3.35
	shale clay with interlayers of arenaceous shale	1.7
	arenaceous shale with interlayers of fine-grained sandstone	2.4
	sandstone	8.9
	coal (Seam 364/3)	0.3
	shale clay	3.4
	arenaceous shale with interlayers of fine-grained sandstone	3.0
	fine-grained sandstone	2.8
	shale clay	0.9
	arenaceous shale	1.1
	coal (Seam 364/4)	0.95
	shale clay	7.85
Seam 401		
Floor	shale clay	3.85
	arenaceous shale	0.6
	sandstone	13.4
	shale clay	12.05
	coal (Seam 402)	4.5

During the Phase I of the working drivage at the distance of 29.8 rm of the Bw-1n test roadway, additional tests of rocks compressive strength with use of the hydraulic apertured penetrometer were conducted. Test results are listed in Table 8.3 [129].

Results of penetrometric tests in the Gp8(2019) borehole at the distance of 29.8 rm of the Bw-1n test roadway [126]

Table 8.3.

Location	**Rock**	R_c **[MPa]**
Roof	shale clay with interlayers of siderite	58.0
	sanded shale clay	53.7
	shale clay interlayed by arenaceous shale	62.7
Wall	coal	17.9
Floor	shale clay interlayed by coal and carbonaceous shale	41.7

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During the following stages of drivage, the penetrometric tests of the rocks' compressive strength were conducted currently. Tests results are listed in Table 8.4 and they are presented in Fig. 8.2.

Penetrometric test results in the boreholes Gp38(2019)/1-14 at the marks of: 80.8; 165.0; 299.0; 375.6 and 1003.7 rm of the Bw-1n test roadway [134]

Table 8.4.

	Mark [rm]				
	80.8	165.0	299.0	375.6	1003.7
Rock	R_c [MPa]				
Roof rocks	38.33	38.58	33.07	39.87	32.06
Rocks of immediate roof	37.79	36.39	28.74	38.27	36.91
Coal	7.21	7.48	7.33	-	16.18
Floor rocks	12.23	9.01	-	20.03	-

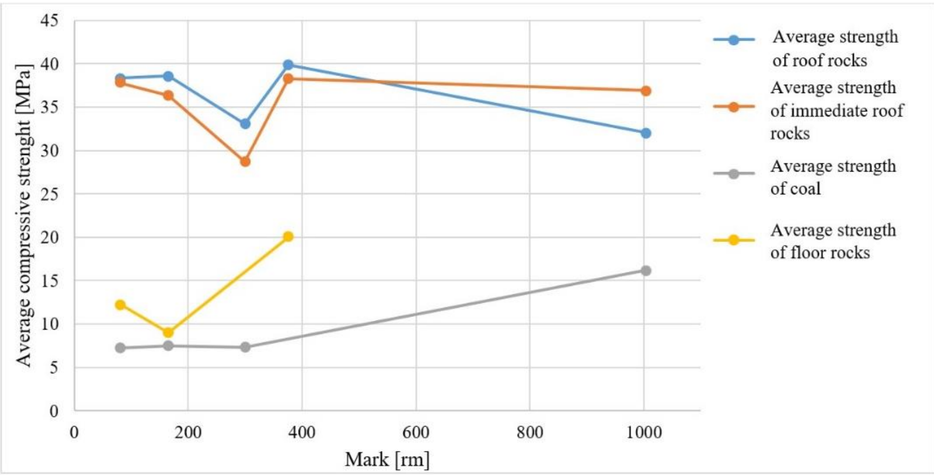

Fig. 8.2. Changeability graph of penetrometric values of the rocks compressive strength at the length of the Bw-1n test roadway [134]

Besides, based on samples taken from the Gp38(2019)/12 borehole core at the mark of 1003.7 rm the soakability coefficient of roof rocks *r* was determined. Test results are listed in Table 8.5.

Results of testing soakability of rocks, according to the GiG method, conducted on the samples taken from the Gp38(2019)/12 borehole on the mark 1003.7 rm of the Bw-1n test roadway [134]

Table 8.5.

No. of sample	Location, lithology	Time [h]		
		24	48	72
		Soakability coefficient *r* [-]		
1	1.34-1.41 m; sanded claystone	1.0	1.0	1.0
2	3.32-3.39 m; sanded claystone	1.0	1.0	1.0
3	6.05-6.11 m; sanded claystone	1.0	1.0	1.0
4	5.33-5.38 m; sanded claystone	1.0	1.0	1.0
5	8.95-9.00 m; sanded claystone	1.0	1.0	1.0

Values of the soakability coefficient *r* of roof rocks stayed at an identical level in relation to those determined before starting the project. Due to that it was possible to continue driving the working in independent roof bolting support.

8.2. Hydrogeological conditions

Due to a significant depth of conducted operations, at the designing stage an occurrence of substantial water intrusion in the rock mass was not predicted. Nevertheless, it was assumed that refluxes, effluents and even possible outflows of water might occur in the areas of sandstone and tectonical dislocations [129].

During the drivage (to the working length of 1044.1 rm) inflows of water from the roof boreholes with installed stratification meters, in the amount 0.05÷0.25 l/min, were observed. Besides, there were outflows of water from the floor in the amount of about 0.2 l/min. All the observed outflows had a decreasing trend until their complete decay. Laboratory tests of taken samples showed that the inflows came from the relict water in the layers of sandstone deposited above the working roof. Due to a local range and small values of inflows the discussed outflows and refluxes did not cause any hazard to an operation of the mining plant [134].

In the Phase IV of driving, the Bw-1n test roadway will be conducted in the distance of about 98 m south of the water reservoir from the Makoszowy mine of volume 213 000 m^3, situated in Seam 364/1. Despite of that the rock mass at the segment, under consideration, should not indicate a bigger flooding, however an occurrence of minor refluxes and effluents of water, coming from the mentioned reservoir and from sandstone deposited in the roof strata, may happen [134].

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8.3. Susceptibility to sparking

In the cross-section of the designed Bw-1n test roadway a possibility of shale clay, arenaceous shale and sandstone presence over and below the seam is predicted. These rocks are characterized by differentiated susceptibility (from non-susceptible rocks to the rocks highly susceptible) to sparking, initiating methane ignition based on the classification of rocks as regards a degree of hazard caused by mechanical sparks during mechanical cutting, presented in Table 1 of Enclosure No. 3 to the Order of Minister of Energy from 23rd November 2016 [129].

8.4. Exploitation of seams deposited above and below

In the distance of about 170 m from the Bw-1n test roadway the Bw-4 ventilating roadway is situated. It was driven in the Seam 358/1 in 2005. In the distance between 24÷31 rm from the designed roadway junction (Bend No. 2) on the mark 994 rm the Bw-1n test roadway will be driven below the exploitational edge of the Seam 358/1 of 2.2÷2.4 m in thickness, and then it will be driven in a small distance from it (2÷20 m). The Seam 358/1 was mined over the years 2006÷2007 with the Bw-4 longwall. A roadway drivage near the exploitational edge may have an impact on the rock mass stress occurring locally [134].

8.5. Tectonical dislocations

Based on the present state of the panel recognition, no tectonical dislocations are predicted. However, it is not possible to exclude faults of minor throw. In the distance of about 35 m in the north-western direction from the place of starting the first stage of applying independent roof bolting support, there is a fault of the throw 7.1 m. In the place where possible small faults occur, a decrease of physicomechanical parameters of rocks and an appearance of effusions, refluxes and effluents of water from sandstone strata in the Seam 401 roof are predicted [129].

8.6. Geological recognition

In the place of starting the Bw-1n test roadway drivage, the total thickness of the Seam 401 was 2.62 m and it consisted of the following strata [135]:

- a stratum of coal of 1.06-metre thickness,
- a dirt band of 0.95-metre thickness,
- a stratum of coal of 0.61-metre thickness.

The thickness of the mentioned dirt band at the length of the working, being driven, varied from 0.40 to 3.20 m, increasing gradually during a drivage of the Bw-1n test roadway in the eastern direction. The dirt band consisted of shale and coal, of shale clay with traces of coal and coal lamina. The thickness of the coal stratum, deposited under the dirt band, varied from 0.40 to 1.00 m. Due to the working drivage method and gradually increasing dirt band thickness, from the mark of about 656 rm, the coal stratum was deposited below the working floor and its thickness was determined with use of testing boreholes [135].

An inclination of strata, including the coal seam, along the roadway, varied from 0° to 7°, locally to 12°. From the working mark, about 1028 rm to the face, i.e. to mark 1044.10 rm an inclination increased to about 15° which had an impact on a deterioration of roof conditions [135].

A changeability of lithology in the face cross-section on the segment of the Bw-1n test roadway is shown in Fig. 8.3-8.9.

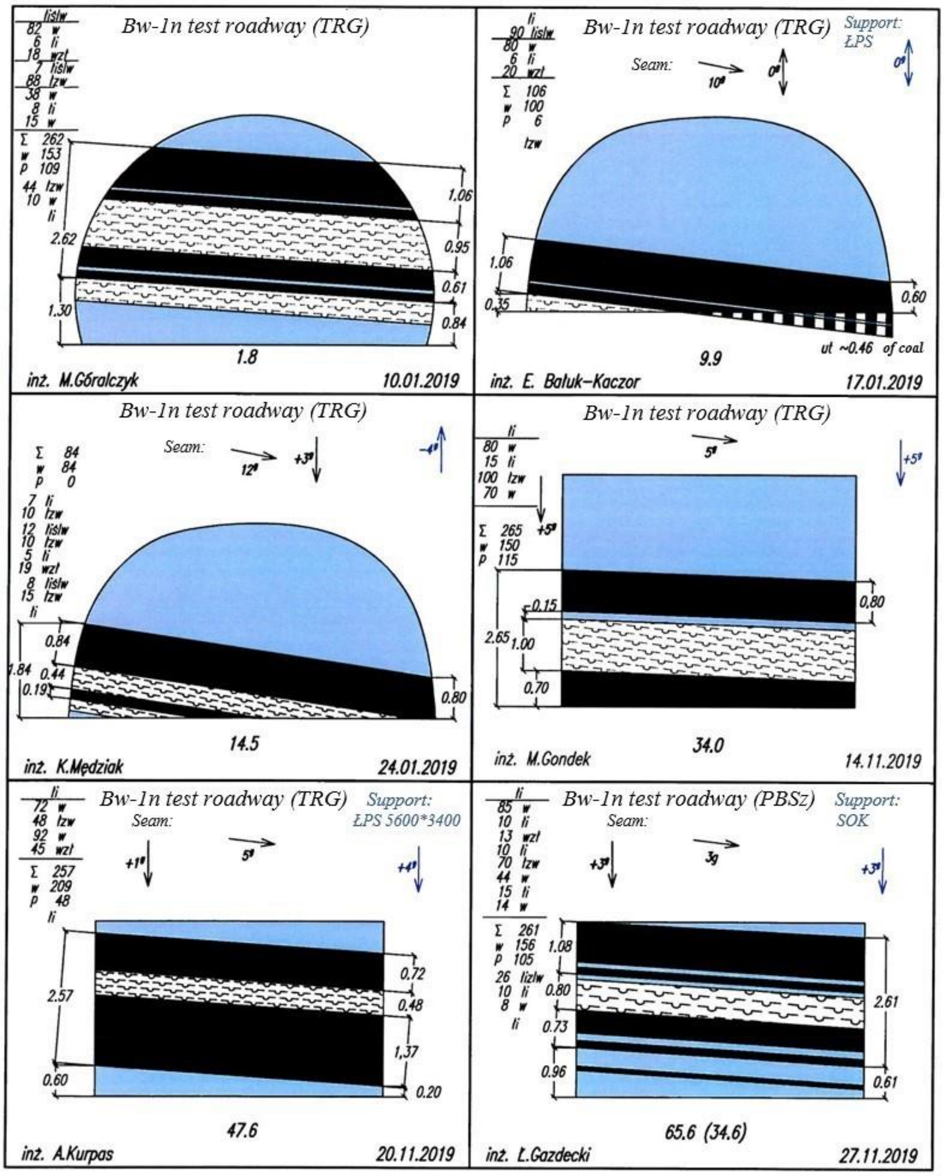

Fig. 8.3. Geological cross-sections of the face 10.01.2019 – 27.11.2019 [135]

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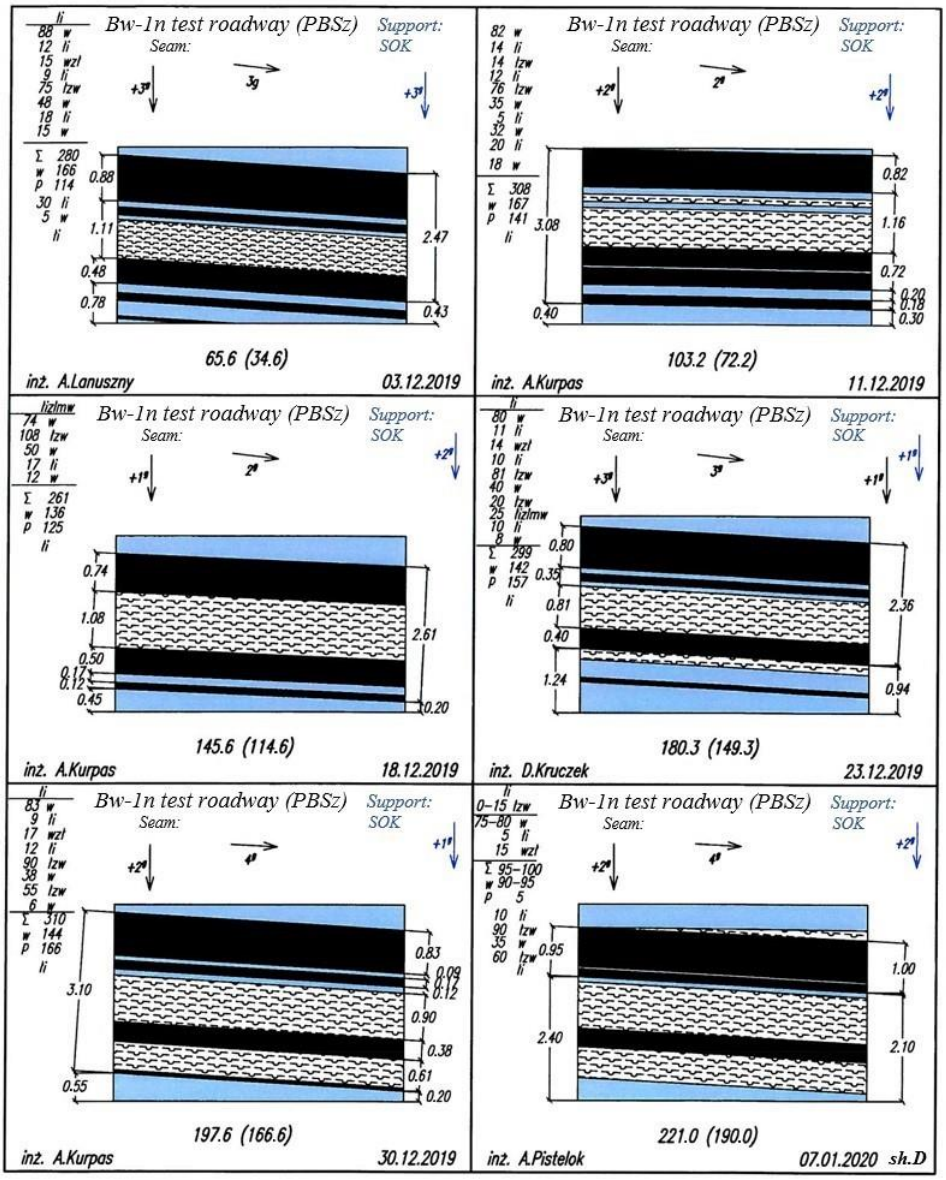

Fig. 8.4. Geological cross-section of the face 3.12.2019 – 7.01.2020 [135]

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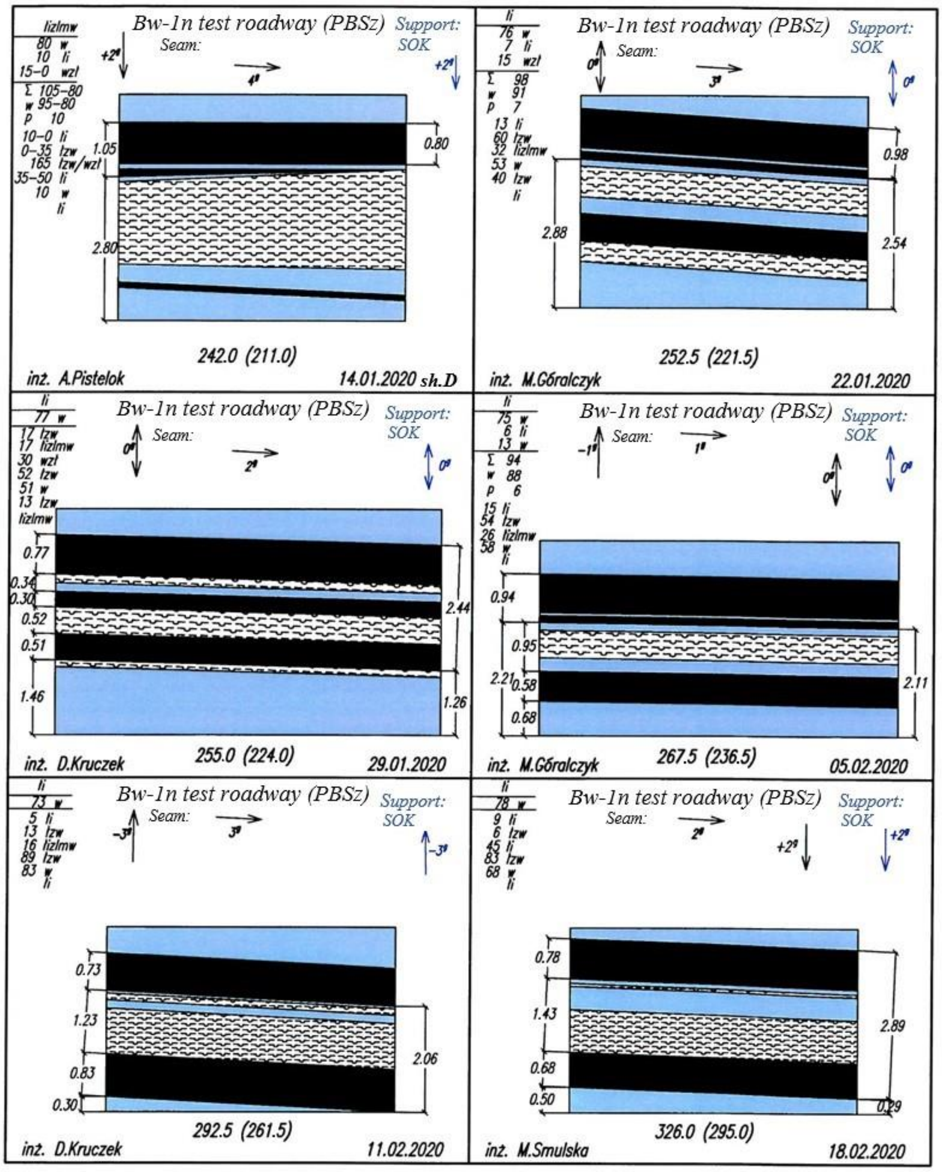

Fig. 8.5. Geological cross-sections of the face 14.01.2020 – 18.02.2020 [135]

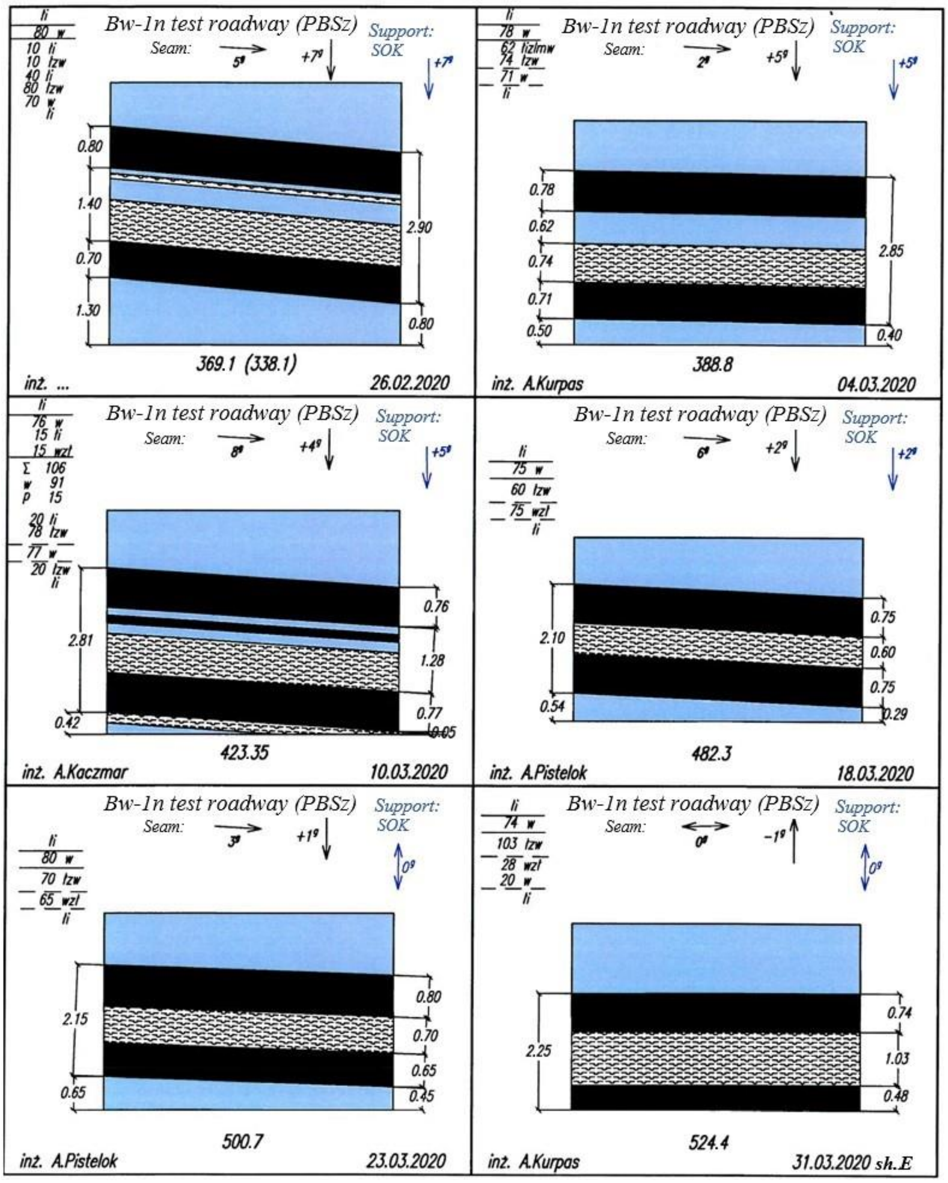

Fig. 8.6. Geological cross-sections of the face 26.02.2020 – 31.03.2020 [135]

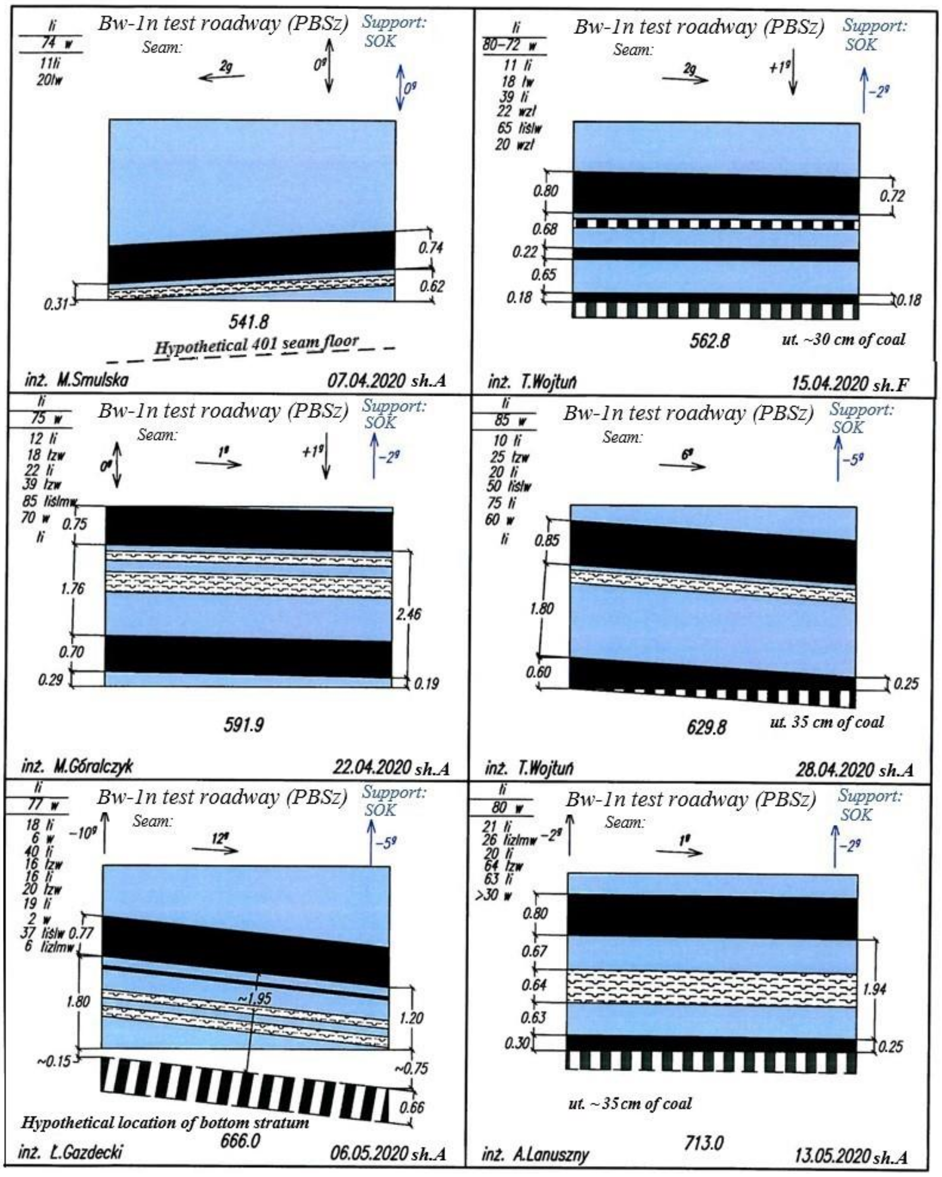

Fig. 8.7. Geological cross-sections of the face 7.04.2020 – 13.05.2020 [135]

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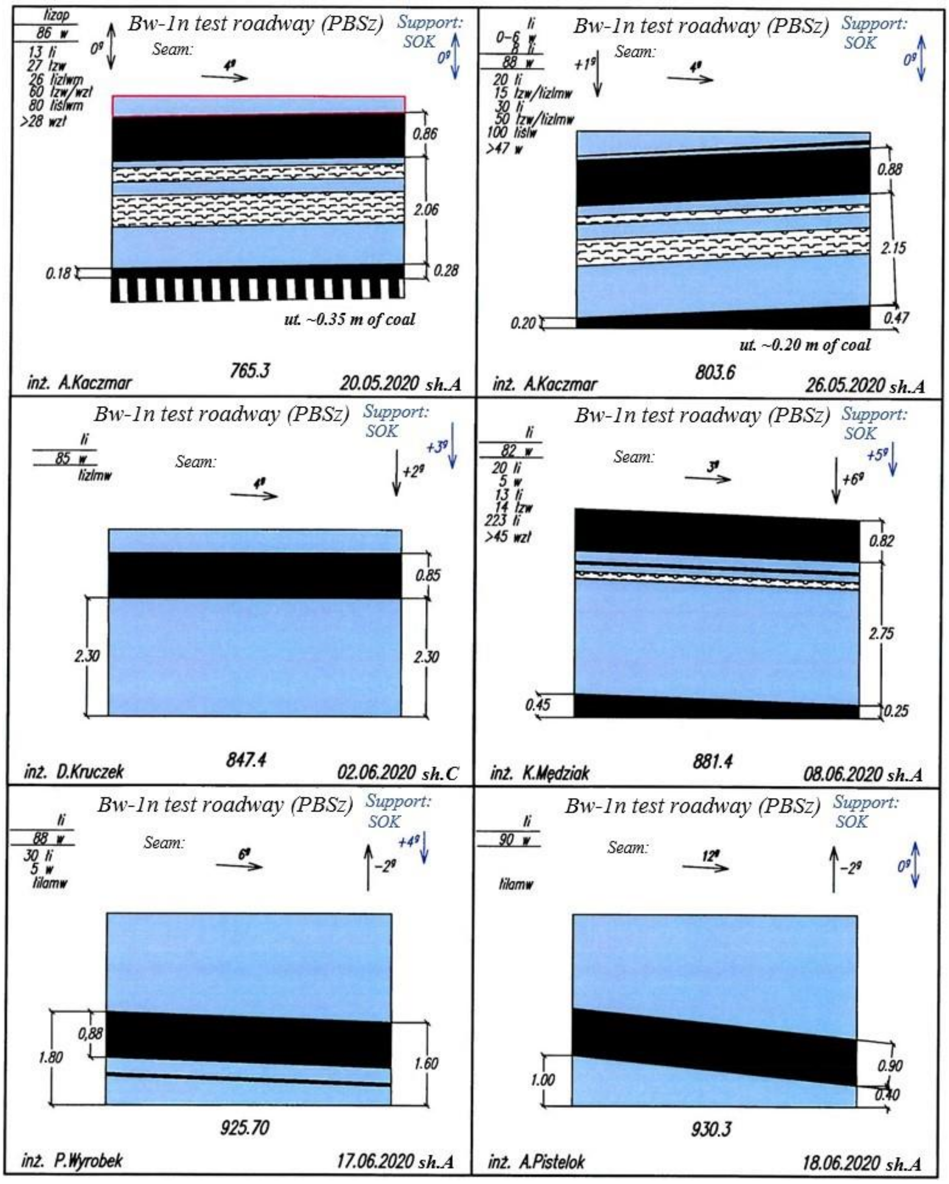

Fig. 8.8. Geological cross-sections of the face 20.05.2020 – 18.06.2020 [135]

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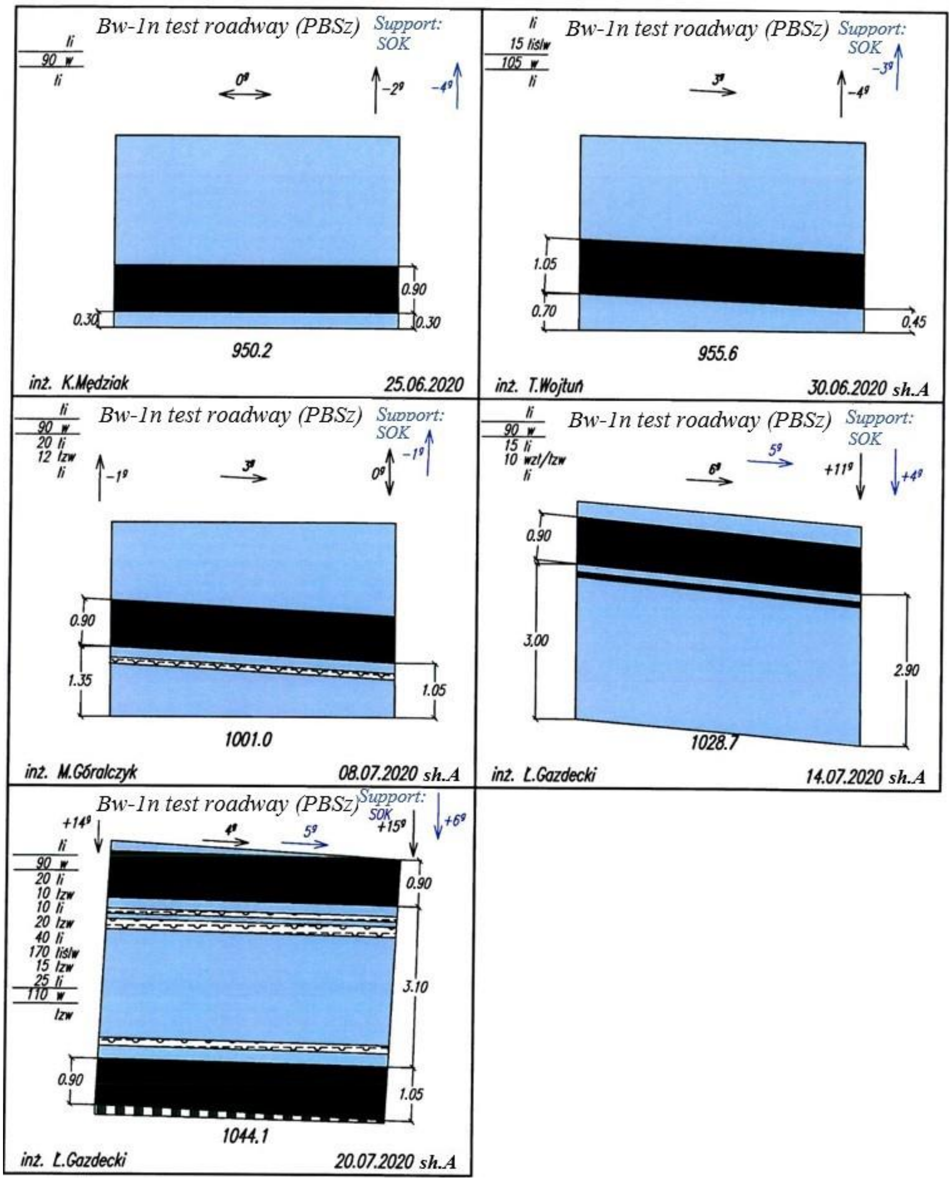

Fig. 8.9. Geological cross-sections of the face 25.06.2020-20.07.2020 [135]

After having driven the Bw-1n test roadway to the mark of about 700 rm, the working face was situated nearly at the height of the G19(2006) testing borehole, located 48 m south of the roadway. Based on an observation of the lithological structure of the face, it was stated that the structure of Seam 401 and of surrounding rocks was significantly different from the structure of the coal stratum, so far identified as Seam 401 in the section of the mentioned borehole.

In relation to a more exact recognition of the deposit, due to the Bw-1n test roadway, a correlation of stratification of Seams 401 and 402 was verified together with an analysis of an impact of these circumstances on the method of the deposit development. Besides, drilling of additional boreholes to test the way of coal strata location, below the floor of the working under drivage and correlating their profiles with the G25(2006) and G16(2006) [135] boreholes, was started with an advance of the Bw-1n test roadway face.

Test results of recognition due to boreholes are listed in Table 8.6.

Results of a geological recognition due to G7(2020)2-5 boreholes [135]

Table 8.6.

Name of borehole	G7(2020)2	G7(2020)3	G7(2020)4	G7(2020)5
Working mark [rm]	458.8	697.2	850.6	1008.4
Distance between Seams 401 and 402 [m]	8.00	4.30	3.50	2.30
Seam 402 thickness [m]	1.55	1.30	0.90	1.15

A repeated analysis of the geological situation in the panel, where coal seams 401 and 402 are deposited, enabled to state that the seams under discussion together with the surrounding strata are characterized by a significant structural changeability and inhomogeneity of stratification, which are difficult to be predicted. According to the recognition dated 24th July 2020, it was stated that in the panel, under description, probably there was a zone where the seams 401 and 402 intergrowed and their total thickness varied from 3.70 to 4.50 m. Towards the south-east of the probable intergrow line, combined coal strata occur exclusively as Seam 402, but Seam 401 is composed of independently occurring coal stratum of about 0.5 m thickness, about 5.60 m above the roof of Seam 402 [135].

Taking into consideration the latest geological conditions a new course of the Bw-1n test roadway was designed which enabled an optimum use of deposit resources due to an elongation of designed exploitational longwalls: Bw-1, Bw-2 and Bw-3 in Seam 402 [135].

Advantages resulting from changing a driving projection of the working and opening Seam 402 in the panel, under designing, include [135]:

– an elongation of Bw longwalls, Batch "B" in Seam 402 to the length of 1500 ÷ 1800 m,
– an elimination of necessity of conducting additional operations connected with a withdrawal of supports and reinforcement of the following longwalls,

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- testing a possibility of implementing independent roof bolting support technology in Seam 402,
- a possibility of testing the stability of gate working driven with use of independent roof bolting support,
- a possibility of testing the stability of opening working at the moment of finishing the longwall operation,
- an earlier start of development working drivage, enabling to begin an operation of the Bw longwalls in the “B” Batch, in Seam 402.

The present and designed course of the Bw-1n test roadway is presented in Fig. 8.10.

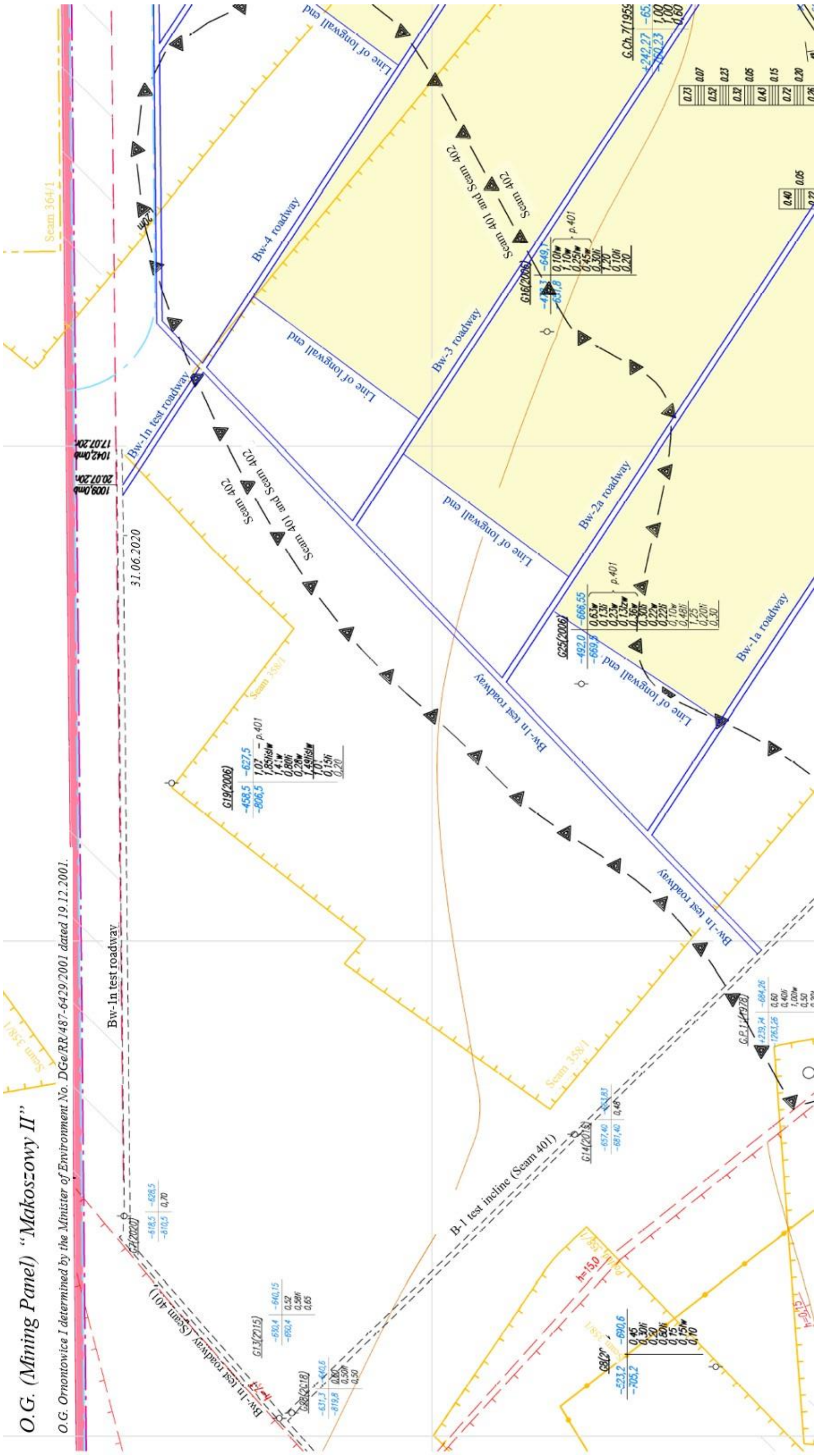

Fig. 8.10. Present (black line) and designed (blue line) course of the Bw-1n test roadway [135]

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9. Natural hazards in the area of Bw-1n test roadway in seam 401

Piotr Dawidziuk[1], Dariusz Wejman[1]

Natural hazards occurring during a drivage of the Bw-1n test roadway in Seam 401 are listed in Table 9.1.

Characteristics of natural hazards [134]

Table 9.1.

Hazard	Category	Description
water	I degree of hazard	The Bw-1n test roadway in Seam 401 was classified to the I degree of hazard.
coal dust explosion	Class B of hazard	Total Seam 401 together with the workings to the north from the "Barbara" throw in the limits of the O.G. (Mining Panel) "Ornontowice" was classified to Class B.
fire	II class of spontaneous ignition	Based on tests conducted in the B-3 test roadway in Seam 401 it was proven that: ▪ spontaneous ignition index $Sz^a_{(237)} = 68^0 C/min$, ▪ spontaneous ignition index $Sz^a_{(190)} = 17^0 C/min$, ▪ index of oxidation activation energy $A = 57\,\frac{kJ}{mol}$, ▪ predicted period of fire incubation $T = 69\ days$. Based on test results a small susceptibility of coal to spontaneous ignition was stated and Seam 401 was classified to the II class of spontaneous ignition.
methane	IV category of hazard	The panel of the Bw-1n test roadway drivage encompasses a part of Seam 401 classified to the IV category of methane hazard.
bumps	none	Seam 401 in the panel of the Bw-1n test roadway drivage is not susceptible to bumps.
radiation	none	Based on the prediction no radiation hazard in Seam 401 was stated.
outbursts of gases and rocks	none	Seam 401 in the panel of the Bw-1n test roadway drivage is not subject to outbursts of gases and rocks.
climatic	none	Primary temperature of the rock mass in the panel, where the Bw-1n test roadway drivage was started, is 36.2°C. Based on a prediction of climatic conditions it has been stated that the temperature at the workplaces will not exceed 28°C, when the cooling device of 300 kW power is in operation and thus no climatic hazard is predicted.

[1] Enterprise of Shafts Construction JSC

9.1. Principles of natural hazards control

Water hazard

In the case of workings, driven in the I degree of water hazard, no special restrictions are required. During an execution of operations it is necessary to conduct a current observation of the rock mass water intrusion according to the decisions of the mining geologist. When an increase of the water intrusion is detected, the mining supervisory officers are obliged to inform the surveying-and-geological department and to undertake actions aimed at an elimination of water collection in the face. Besides, in the case of driving the working in a weakly recognized part of the deposit, the face advance should be preceded by protection holes of the length not smaller than 4.0 m plus the web depth [134].

Water from cooling the machines and equipment as well as water from local flood waters of the Bw-1n test roadway is drained by the dewatering pipeline Ø150 mm in the direction of the OS 100 pump assembly, installed in the C-5 Drift and then pumped through the Ø150 mm pipeline in the direction of water roadways, in the area of the central shafts on the level 1050 [134].

Coal dust explosion hazard

The following preventive measures against a generation and a propagation of the coal dust explosion are used while driving the working [134]:

– Spraying through spraying nozzles installed by the manufacturer on the JOY 12CM30 machine and on all the discharge areas of the run-of-mine haulage conveyors. An appropriate amount and pressure of water in the anti-fire pipeline should be supplied for a given type of nozzles.

– A supply of a surfactant, aimed at a reduction of the surface tension, thus improving an efficiency of spraying and of dust control, to the water used for spraying.

– An application of an internal dust controlling device of the JOY 12CM30 machine and of the dust controlling device of the ODM-800 type installed in the face, in the system of suction ventilation.

The zones protecting against a propagation of coal dust explosion are maintained at the working full length through [134]:

– rock dusting to the contents of at least 80% of non-combustible solid parts or,

– washing with water until reaching the content of transient water in the mine dust, making a propagation of coal dust explosion impossible or,

– washing with water and spreading of a hygroscopic agent until reaching the content of total water in the mine dust, making a propagation of coal dust explosion impossible.

Samples of dust for laboratory analyses, oriented onto a determination of non-combustible solid parts, transient water or total water, should be taken from protecting zones. In the case of detecting lack of the required content of non-combustible solid parts, transient water or total water, a drivage of the face should be stopped and electric equipment in the face should be switched off and then a neutralization of dust should be conducted. Switching on the voltage and restarting operations will be possible after obtaining positive results of laboratory tests [134].

The working, under driving, should be protected with an explosion barrier constructed in the distance from 60 to 200 m from the face front. Besides, the collected coal dust, after eliminating its volatility should be removed periodically. An accumulation of dust under conveyor belts and also in the area of drives and of conveyors return ends [134] should be removed currently.

Fire hazard

An early detection of fires in the Bw-1n test roadway is conducted with use of manual measurements at measuring stations [134]:

- at the air inlet to Electric Booster Fan (WLE), ventilating the working,
- in the outlet air current.

Tests should be conducted twice a week. Besides in the Bw-1n test roadway, in the distance of 10-15 m from the B-1 test incline, a carbon monoxide detector of measurement range 200 ppm [134] is installed.

Anti-fire pipelines are conducted to the Bw-1n test roadway. Their main source of supply is located on the level 500 in the area of the Shaft III. A stand-by source of supply for fire pipelines is a reservoir on the level 164 in the Shaft I of 300 m^3 capacity [134].

Due to cutting with use of a machine and a haulage of the run-of-mine with use of belt conveyors, the following arrangement of the fire equipment was decided [134]:

- hydrant valves every 50 m along the belt conveyor flight and maximum 20 m from the side of air inflow at drives and return stations of the run-mine haulage conveyors,
- hydrant cabinets: one at each place of hydrant valves installation and two at two hydrants closest to the face,
- fire pipeline ended with a three-way hydrant in the distance not bigger than 50 m from the face front,
- two Automatic Fire-Extinguishing Devices are installed on the JOY 12CM30 machine and additionally one dry-chemical extinguisher – 6 kg,

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- 3 x 12 kg (or 6 x 6 kg) dry-chemical extinguishers in the area of the working face,
- 2 x 12 kg foam extinguishers and 1 x 6 kg dry-chemical extinguisher at the drives of the run-of-mine haulage conveyors installed in the working,
- 2 x 12 kg foam extinguishers – located every 200 m of each conveyor belt flight course.

The biggest hazard of fire in the working, under drivage, comes from an operation of machines and belt conveyors. A present scheme of escape routes should be hung in the place of the workers` division. Each person, present in the underground part of the mining plant, possesses an insulating equipment protecting upper respiratory tracts, approved for use in underground mines. In the Budryk mine insulating apparati with chemically combined oxygen are used [134].

Methane hazard

During the drivage of the Bw-1n test roadway the following principles of methane hazard prevention [134] were established:

- To ensure an indispensable amount of air in the working face a current check-up of the technical condition of the pumping air duct should be conducted and leakages should be eliminated currently.
- The cutter and the machine operator are obliged to check a condition of cutting picks before starting a cutting operation. Besides, the machine operator and the fitter are obliged to check the spraying system.
- The end of suction air duct or the exhaust fan of the dust control equipment should be kept in the distance which does not exceed 6 m from the face front, but of the pumping air duct-in the distance which does not exceed 12 m.
- The devices for an automatic control of methane content should be kept in the working.
- The cutter or the machine operator should have a methane detector in the face for a continuous control of the methane content under the working roof, in the places of biggest emission or accumulation of methane between the face and the air inlet to the suction air duct with the alarm threshold of 1.0%.
- In the places of methane outflow or accumulation along the course of air ducts, ventilators made of ventilation cloth, jet pumps or perforated tubes, supplied with compressed air, enabling to eliminate dangerous methane concentrations, should be installed.

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- In the case of increased methane emission (below permissible values) in the face and in the zone 50 m from the face operations, connected with the working advance, the operations should be stopped till the time of increased methane emission elimination with use of additional auxiliary ventilation devices.
- Auxiliary ventilation devices should be installed in the way excluding an interference of methanometric detectors.
- In the case of an occurrence of breaches in the roof over the support, they should be filled in tightly. However, if such a necessity arises appropriate measures, eliminating next-to-the-roof methane concentration, should be applied.
- An execution of a reconstruction of auxiliary ventilation equipment in the zone 50 m from the face front and conducting operations on the compressed air pipeline are forbidden during mining the face.
- The panel of the working, under drivage, is subject to a control conducted by gasmen from the W-1 division, maintenance technicians from the EDŁ division, cutters and machine operators, mining supervision, ventilation, higher supervision of the TTG-2 department and higher supervision of ventilation.

Mechanical cutting of cohesive rock and a big susceptibility to sparking with use of a machine can be conducted under the following conditions [134]:

- Air velocity in the working must be in accordance with the technical design of the air duct ventilation.
- Methane content measured with a detector of automatic methanometry, installed under the working roof in the distance not exceeding 2 m from the face front and causing an automatic machine switching off, cannot exceed 1.0%,
- Near-the-roof methane concentrations cannot occur in the distance up to 50 m from the place of cutting cohesive rocks.
- In the presence of the supervisory person test holes, preceding the face front of the length minimum 1 m, drilled in each of cohesive rocks layers of medium and big susceptibility to sparking, should be executed. In the made holes no overpressure can occur.

Bumping hazard

The panel of the working drivage is under a continuous observation, conducted by the station of mining geophysics of the Budryk mine with use of seismological observation system “SOS”, recording tremors of energy in the order of 10^2 J. In the case of possible recording of increased seismic activity the Mine Team for Recognizing and Control of Bumping Hazards and the Mine Team for Support will establish additional conditions of driving workings [134].

Radiation hazard

A measurement of potential energy concentration α [134] will be taken for an assessment of a real radiation hazard.

Hazard of gases and rocks outbursts

In the driven working, according to the regulations in power, measurements of methane content and volatile parts content, should be taken and the methane desorption intensity index, coal cohesion index and an output of drillings at intervals not exceeding 50 m, should be determined. In the case of detecting symptoms indicating a possibility of gases and rocks outbursts occurrence, all the further operations should be stopped and geological and ventilation services [134] should be informed.

A decision about drilling test holes, preceding the face front and about their range, is taken by the Operational Manager of Mining Plant based on an opinion of the proper Mine Team. When driving with use of a machine, the length of test holes should not be smaller than 10 m [134].

Climatic hazard

Based on the temperature prediction, no change of climatic conditions is expected, the conditions understood as dry air temperature above 28°C on the fixed work places and on representative workplaces, at the cooling device in operation, having the rated cooling power of 300 kW, installed in the air current flowing to the fan ventilating the working under drivage [134].

A need of installing cooling devices and also their number in the area of driving Bw-1n test roadway depends on in-situ climatic conditions in the working being driven [134].

To determine the climatic conditions in the working under drivage, the workers of the ventilation department take measurements of microclimatic parameters of mine air at the frequency at least once a month at work-places, where no climatic hazards occur or if they are classified as belonging to the I degree of the climatic hazard. In the case of classifying the work-places to the II degree of the climatic hazard, the measurements of the microclimate parameters should be taken at least once a week [134].

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10. Design principles of independent roof bolting support

Marcin Mieszczak[1], *Andrzej Walentek*[2]

A design of independent roof bolting support for the Bw-1n test roadway in Seam 401 was elaborated by the Central Mining Institute. The designing process was conducted on the basis of two methods [136]:

- analytical-empirical,
- numerical modelling.

The objective of the analytical-empirical method consisted in a determination of the working load, of the zone of fractures of roof rocks and coal walls and also a development of the bolting scheme, encompassing spacing and length of bolts. The objective of the numerical method was a verification of correctness of the conducted calculations as regards a collaboration of bolts with the rock mass [136].

The applied design methods take into consideration [136]:

- a vertical strength inhomogeneity and natural separation of roof rocks determined in situ,
- an impact of exploitational edge, pillars, gob, and of adjacent workings on the stress state in the rock mass,
- a range of fractures of wall rocks at an assessment of a real span of exposed roof and a deflection of the extreme bolts from the plumb-line,
- a maximum range of roof rocks breaking in the vertical cross-section of the roof as the basis for a selection of an appropriate length of bolts,
- an impact of the working maintenance time and an impact of moisture on the strength parameters of roof rocks.

10.1. Analytical-empirical method of designing roof bolting support

The design of independent roof bolting support requires a recognition of the following factors which can have an impact on the working stability [136]:

- physico-mechanical properties of roof and wall rocks,
- a stress state of the rock mass and a range of the zone of fractures in the direct vicinity of the working,

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– an exploitational past operations,
– tectonic dislocations,
– a possibility of dynamic phenomena occurrence.

In Fig. 10.1 an algorithm of activities during the roof bolting support designing with use of analytical-empirical method is presented.

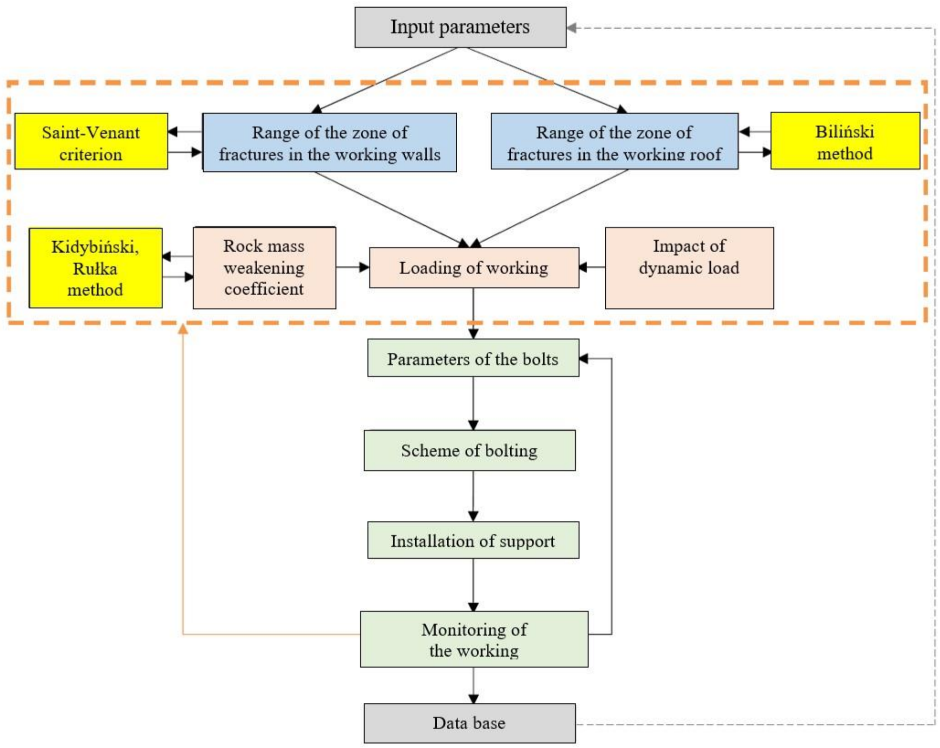

Fig. 10.1. Algorithm of activities during the roof bolting support designing with use of analytical-empirical method [136]

According to the algorithm of activities, presented in Fig. 10.1, the input parameters, including the following mining-and-geological conditions, should be subject to an analysis first of all [136]:

- W_w – height of the working in the breakout, m,
- S_w – width of the working in the breakout, m,
- H – depth of the working location, m,
- R_c – average compressive strength of roof rocks, MPa,
- R_{co} – average compressive strength of wall rocks, MPa,

- r – coefficient of rocks moisture impact, -,
- p – coefficient of the working maintenance time, -,
- γ_o – average weight by volume of wall rocks, MN/m^3,
- γ_s – average weight by volume of roof rocks, MN/m^3,
- RQD – splitting of drilling core (slotting index), %,
- k_α – coefficient of transverse inclination impact of rock strata, -,
- k_β – coefficient of the working longitudinal inclination impact, -,
- k_s – coefficient of the adjacent working impact, -,
- k_e – coefficient of edges impact, -,
- k_u – coefficient of fault impact, -,
- q_{dyn} – unit dynamic load, MPa.

The computational process starts from a determination of the range of the zone of fractures in the roof according to the Biliński method and of the range of the zone of fractures in the walls, based on the Saint-Venant strength criterion [136].

Afterwards, the load of the working is determined, taking into consideration the following factors [136]:

– an impact of energy of dynamic phenomena,

– local stress concentrations in the rocks, surrounding the working under designing, resulting from an impact of exploitational edges, vicinity of faults, gob and of other workings,

– coefficient of the rock mass weakening dependent on water contents, occurrence of natural fractures (RQD index) and predicted time of the working maintenance.

In the further order, a scheme of the working bolting is designed on the basis of the parameters of bolts (length, diameter and load-bearing capacity). For safety reasons the length of the bolts fixation in the rock mass should be bigger by 0.3 m from the predicted range of the zone of fractures. The bolting system is determined from the relationship of separated rocks to the load-carrying capacity of an individual bolt [136]. Thus, the following condition must be met [1366]:

$$Q_k = \pi \cdot \gamma_0 \cdot l_1 \cdot l_3 \cdot h_s < N_k \tag{10.1}$$

where:
Q_k – loading of an individual bolt, kN,
γ_0 – weight by volume of the rock mass situated above, kN/m^3,

l_1, l_3 – longitudinal and transverse spacing of bolts respectively, m,
h_s – range of zone of fractures, m,
N_k – real load-bearing capacity of bolts, kN.

Therefore, the design of independent roof bolting support encompasses:

- the fixation length of bolts in the rock mass,
- longitudinal and transverse spacing of bolts,
- a type of bolt together with its basic parameters,
- a method of fixation the bolt in the rock mass.

After having driven the working in the independent roof bolting support, a complex monitoring of bolts (testing of the bolt load-bearing capacity and of an accuracy of the rod insertion) and of the rock mass (roof subsidence and the working convergence) should be carried out to enable a verification and a possible correction of the design assumptions. In the case of exceeding permissible values of the roof subsidence of the working roof or not meeting the condition determined in the test of the bolt pulling out, an appropriate correction of the independent roof bolting support should be introduced immediately [136].

10.2. Designing method of bolting support with use of numerical modelling

The method of numerical modelling was used for a correctness verification of calculations, conducted with use of analytical-empirical method. The RS3 software, developed by the Rocscience Company, based on the Finite Elements Method [136] was used.

The Finite Elements Method is based on a division of a certain rock mass sector onto elements of finite dimensions. This action enables to determine relationships among the stress state, displacement and deformation. The condition of the boundary state, calculated on the basis of the Hoek-Brown criterion for the fractured rock mass, is determined from the following relationship [136]:

$$\sigma_1' = \sigma_3' + \sigma_{ci} \cdot \left(m_b \cdot \frac{\sigma_3'}{\sigma_{ci}} + s \right)^a, \tag{10.2}$$

where:
σ_1', σ_3' – effective maximum and minimum stresses at failure, MPa,
m_b – value of Hoek-Brown constant for the rock mass under consideration, -,
s, a – constants determined on the basis of the rock mass properties,
σ_{ci} – uniaxial compressive strength of the rock sample, MPa.

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The m_b parameter is determined from the following formula [23]:

$$m_b = m_i \cdot \exp\left(\frac{GSI - 100}{28 - 14 \cdot D}\right), \tag{10.3}$$

where:
m_i – the constant for undisturbed rock dependent on its type, determined on the basis of the triaxial compression test or on the basis of tables, -,
GSI – the parameter of the rock mass quality (*Geological Strength Index*), -,
D – the destruction coefficient dependent on the rock type and accepted mining method, -.

When the values of $GSI > 25$, parameters s and a are determined from the following formula [136]:

$$s = \exp\left(\frac{GSI - 100}{90 - 3 \cdot D}\right), \tag{10.4}$$

$$a = \frac{1}{2} + \frac{1}{6} \cdot \left(e^{-\frac{GSI}{15}} - e^{-\frac{20}{3}}\right). \tag{10.5}$$

When the values of $GSI < 25$ the parameter $s = 0$, and the parameter a is determined from the following formula [136]:

$$a = 0.65 - \frac{GSI}{200} \tag{10.6}$$

The most essential assumption, made in the numerical modelling process, is the appropriate selection of vertical stresses σ_z and of horizontal stresses $\sigma_x = \sigma_y$ acting on the rock mass model under consideration. In the initial phase of designing, for a calculation of a straight segment made in Phase I of the project, the stresses were determined from the following formula [136]:

$$\sigma_z = \gamma \cdot H \ [MPa], \tag{10.7}$$

where:
γ – weight by volume of the rocks situated above, MN/m^3,
H – depth, m.

$$\sigma_x = \sigma_y = \sigma_z \cdot \frac{\nu}{1 - \nu}, \tag{10.8}$$

where:
ν –Poisson’s ratio, -.

In the mining practice, it happens in many cases that values of horizontal stresses $\sigma_x = \sigma_y$ are bigger than those determined according to the formula (8). Based on the underground test results, the empirical coefficient k was determined. It defines a relationship between horizontal and vertical stresses. The k coefficient is determined from the following formula [136]:

$$k = \frac{100}{H} + 0.3 \tag{10.9}$$

In the case of designing the support of the bend No. 2, the results of tests of stresses state in the rock mass, conducted *in situ* in the panel of the working under designing, conducted by the Golder Associates (UK) Ltd., were used. These tests showed that hydrostatic state of stresses occurs in the mine area under consideration, what means that the value of vertical stresses is equal to the value of horizontal stresses [133]:

$$\sigma_x = \sigma_y = \sigma_z.$$

Besides, for the considered models the following assumptions were made [136-137]:

– the modelled rock mass is elastic-plastic and isotropic medium,

– lack of possibilities of horizontal and vertical displacements occurrence on the model vertical edges,

– lack of possibilities of vertical displacements occurrence on horizontal edges of the model.

10.3. Design of independent roof bolting support acc. to the Central Mining Institute (Główny Instytut Górnictwa)

Analytical-empirical method

Input parameters, assumed by the Central Mining Institute for initial calculations of the support loading, of the first straight segment (Phase I of the design) of the Bw-1n test roadway in the independent roof bolting support, are specified in Tables 10.1 and 10.2. The coal seam thickness of 1.8 m was assumed in the calculations, which in the case of the 3.4 m of the working height, indicates bottom rocks ripping of 0.5 m and roof rocks ripping of 1.1 m [136].

Input parameters for initial calculations of the straight segment support of the Bw-1n test roadway – mining-and-geological conditions [136]

Table 10.1.

Symbol	Parameter	Unit	Value
W_w	height in the breakout	m	3.4
S_w	width in the breakout	m	5.4
H	depth of the working location	m	880

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R_c	average compressive strength of roof rocks	MPa	36.8
R_{co}	average compressive strength of wall rocks	MPa	16.9
r	coefficient of rocks moisture impact	-	1.0
p	impact coefficient of the working maintenance time	-	1.2
γ_s	average weight by volume of roof rocks	MN/m^3	0.025
γ_o	average weight by volume of wall rocks	MN/m^3	0.014
RQD	separation of drill core (slotting index)	%	30.0
k_u	coefficient of fault impact	-	1.0
k_α	impact coefficient of the working transverse inclination	-	1.0
k_β	impact coefficient of the working longitudinal inclination	-	1.0
k_e	coefficient of edges impact	-	1.2
k_s	coefficient of adjacent working impact	-	1.0
q_{dyn}	unit dynamic load	MPa	0.023

Input parameters for initial calculations of the straight segment support of the Bw-1n test roadway – characteristics of steel bolts [136]

Table 10.2.

Symbol	Parameter	Unit	Value
R_e	yield point	MPa	>640
R_m	steel tensile strength	MPa	>770
A_s	elongation of steel	%	18
N_k	minimum load-bearing capacity of bolts	kN	260
d_r	nominal diameter of roof rods	mm	21.7
d_{ro}	nominal diameter of wall rods	mm	21.7
d_o	diameter of bolt hole in the roof and wall	mm	28.0

The results of calculations, conducted with use of analytical-empirical method, are listed in Table 10.3.

Results of initial calculations of the straight segment support of the Bw-1n test roadway [136]

Table 10.3.

Symbol	Parameter	Unit	Result
h_s	range of the zone of fractures in the roof	m	2.1
h_0	range of the zone of fractures in the wall	m	1.2
q	loading of the working	MPa	0.072
i_{ks}	number of roof bolts in the row	pcs.	6
i_{ko}	number of wall bolts in the row	pcs.	3
l	spacing between rows of bolts	m	1.0

Input parameters, accepted by the Central Mining Institute for calculating the bend support loading of the Bw-1n test roadway, in independent roof bolting support on the example of the bend No. 2, made on the 994 rm mark (Phase IV of the project design), are given in Tables 10.4-10.6. In the calculations two variants of the working overall dimensions were taken into consideration. According to the mark 1003.7 rm the thickness of the coal seam equal to 0.9 m was accepted for calculations. At the roadway height of 4.2 m, rippings of 1.95 m of roof rocks and 1.35 m of bottom rocks were required. In the case of the roadway height of 4.0 m rippings of 2.05 m of roof rocks and 1.05 m of bottom rocks were needed [137].

Input parameters for calculations of the bend support of the Bw-1n test roadway – mining-and-geological conditions [137]

Table 10.4.

Symbol	Parameter	Unit	Value	
W_w	height in the breakout	m	4.2	4.0
S_w	width in the breakout	m	5.6	6.8
H	depth of the working location	m	900	900
R_c	average compressive strength of roof rocks	MPa	36.9	36.9
R_{co}	average compressive strength of wall rocks	MPa	27.7	27.2
r	impact coefficient of rocks moisture	-	1.0	1.0
p	impact coefficient of the working maintenance time	-	1.3	1.3
γ_s	average weight by volume of roof rocks	MN/m^3	0.025	0.025
γ_o	average weight by volume of wall rocks	MN/m^3	0.014	0.025
RQD	separation of drill core (slotting index)	%	52.3	52.3
k_u	coefficient of fault impact	-	1.0	1.0
k_α	impact coefficient of the transverse inclination of rock strata	-	1.0	1.0
k_β	impact coefficient of the working longitudinal inclination	-	1.0	1.0
k_e	coefficient of edges impact	-	1.2	1.2
k_s	coefficient of adjacent working impact	-	1.0	1.0
q_{dyn}	unit dynamic load	MPa	0.023	0.023

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Input parameters for calculating the bend support of the Bw-1n test roadway – characteristics of steel bolts [137]

Table 10.5.

Symbol	Parameter	Unit	Value
R_e	yield point	MPa	>640
R_m	steel tensile strength	MPa	>770
A_s	elongation of steel	%	18
N_k	minimum load-carrying capacity of bolts	kN	260
d_r	nominal diameter of roof rods	mm	21.7
d_{ro}	nominal diameter of wall rods	mm	21.7
d_o	diameter of bolt hole in the roof and wall	mm	28.0

Input parameters for calculating the bend support of the Bw-1n test roadway - characteristics of long elastic bolts [137]

Table 10.6.

Symbol	Parameter	Unit	Value
N_k	minimum load-bearing capacity of bolts	kN	320
d_r	nominal diameter of rods	mm	18
d_o	diameter of bolt hole in the roof and wall	mm	28

The results of calculations, conducted with use of the analytical-empirical method, are presented in Table 10.7.

Results of calculations of the bend support of the Bw-1n test roadway [137]

Table 10.7.

Symbol	Parameter	Unit	Result	
			working 5.6x4.2 m	working 6.8x4.0 m
h_s	range of the zone of fractures in the roof	m	1.3	0.6
h_0	range of the zone of fractures in the wall	m	0.5	1.6
q	loading of the working	MPa	0.056	0.064
i_{ks}	number of roof bolts in a row	pcs.	6	8
i_{ko}	number of wall bolts in a row	pcs.	4	4
l	spacing between rows of bolts	m	1.0	0.8

Method of numerical modelling

To conduct an analysis of the working, under designing, drivage, a two-dimensional (for an initial first design of the straight segment support) and three-dimensional, cubicoid (for design of the bend No. 2 support) were elaborated.

They illustrated local mining-and-geological conditions and the bolting scheme. Respective values of parameters of individual rock strata were accepted on the basis of the characteristics of the panel under consideration, the tests conducted by the Central Mining Institute as well as on the basis of calculations made with use of the RocLab module. Basic parameters of rock strata, accepted for initial calculations of the support, are presented in Tables 10.8 and 10.9. A view of a two-dimensional model of the rock mass is shown in Fig. 10.2. A view of a three-dimensional model of the rock mass is shown in Fig. 10.3.

Basic parameters of rock strata in the panel of the Bw-1n test roadway accepted for initial numerical calculations of the straight segment support [136]

Table 10.8.

Type of rock	Young's modulus	Poisson's ratio	Compressive strength	Analytical parameter	
	E	***v***	R_c	m_b	***s***
	MPa	**-**	**MPa**	**-**	**-**
coal	1700	0.30	8.9	0.9632	0.0012
shale clay - roof	3100	0.25	33.0	1.5632	0.0042
sanded shale clay - roof	3300	0.25	43.0	1.5632	0.0042
arenaceous shale - roof	4900	0.22	61.0	2.2050	0.0067
sandstone - roof	5900	0.20	66.7	3.7932	0.0094
shale clay - floor	2700	0.25	10.8	1.4620	0.0041
shale clay - floor	2700	0.25	15.8	1.4620	0.0041

Basic parameters of rock strata in the panel of the Bw-1n test roadway accepted for numerical calculations of the bend No. 2 support [137]

Table 10.9.

Type of rock	Young's modulus	Poisson's ratio	Compressive strength	Analytical parameter	
	E	***v***	R_c	m_b	***s***
	MPa	**-**	**MPa**	**-**	**-**
coal	1700	0.30	16.3	0.9632	0.0012
shale clay - roof	3300	0.25	36.7	1.8623	0.0062
arenaceous shale - roof	4900	0.22	44.0	2.2050	0.0082
sandstone - roof	5900	0.20	66.7	3.7932	0.0094
shale clay - floor	2700	0.25	20.0	1.8100	0.0061
shale clay - floor	2700	0.25	41.9	1.9210	0.0062

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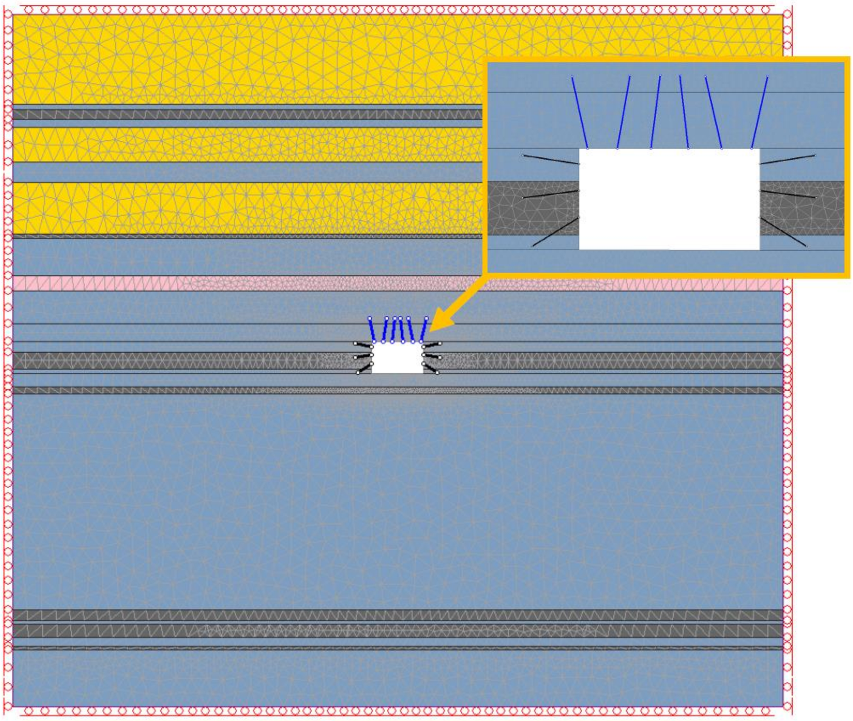

Fig. 10.2. Two-dimensional model of the rock mass acc. to GiG – support of the first straight segment [136]

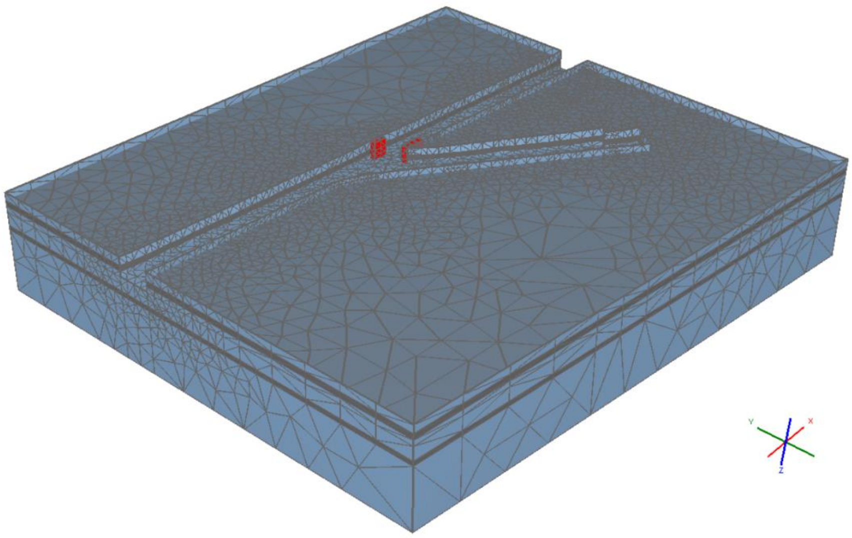

Fig. 10.3. Three-dimensional model of the rock mass acc. to GiG – support of the bend No. 2 [136]

During an elaboration of the initial support design of the Bw-1n test roadway, situated at the depth of 880 m, based on the formulae (10.7), (10.8) and (10.9), the following values of stresses in the rock mass were determined [136]:

$$\sigma_z = 19.3\ MPa,$$

$$\sigma_x = \sigma_y = 8.2\ MPa.$$

During a calculation of the bend No. 2 support of the Bw-1n test roadway, situated at the depth of 900 m, based on underground tests, an occurrence of a hydrostatic state of stresses in the rock mass was stated at the following values [137]:

$$\sigma_z = \sigma_x = \sigma_y = 21.6\ MPa$$

On the basis of the parameters, presented above, and assumptions of the state of stresses in the rock mass, it was possible to conduct calculations of the stresses distribution and of a range of the zones of fractures around the workings under designing and of values of the axial forces in bolts installed in these workings [136-137].

In the result of the numerical analysis, conducted by the Central Mining Institute, a correctness of calculations, made with use of the analytical-empirical method, was stated. Initially accepted bolting scheme, at the assumed mining-and-geological conditions, will enable a maintenance of the working stability and functionality. It is confirmed by:

- values of forces in bolts, smaller than the load-bearing capacity of the suggested bolts,
- a range of the zone of fractures, smaller than the length of suggested bolts,
- low values of the working contour dislocations.

11. Technology of the working drivage in the independent roof bolting support

Paweł Kamiński[1,2], Marcin Mieszczak[1], Stanisław Prusek[3], Sylwester Rajwa[3]

11.1. Assembly of JOY 12CM30 machine

Components of the cutting-bolting machine JOY 12CM30 were lowered underground to the Budryk mine at the beginning of October 2019. The heaviest subassemblies, including the ranging arm gear, the cutting head, the arm of the cutting head with motors, the central frame, both caterpillars, and elements of the feeder were lowered through the Shaft VI. All the other components together with charging materials were lowered through the Shaft III [130].

On 10th October 2019 an underground in situ visit was realized and the place of the project realization was approved. Before starting the machine assembly, it was indispensable to establish an appropriate sequence of connecting individual parts. It was particularly important due to the fact that the JOY 12CM30 machine was transported from the United States to Tychy in the elements of too big weight and overall dimensions, which rendered their lowering in the shaft as well as a further transportation to the assembly chamber impossible. Due to that it was necessary to conduct a disassembly of the machine components in the way, enabling their transportation, simultaneously avoiding unnecessary complications during their reassembly [130].

Before starting the assembly it was indispensable to do as follows [130]:

- to confirm a delivery completeness of elements (Fig. 11.1),
- to identify overall dimensions and weights of individual elements to ensure a correct transportation to the assembly chamber (Table 11.1),
- to check a lay-out of the machine elements in the assembly chamber from the point of view of their compliance with the assembly manual (Fig. 11.2),
- to conduct a visual inspection of the elements delivered to the assembly chamber from the point of view of possible damages.

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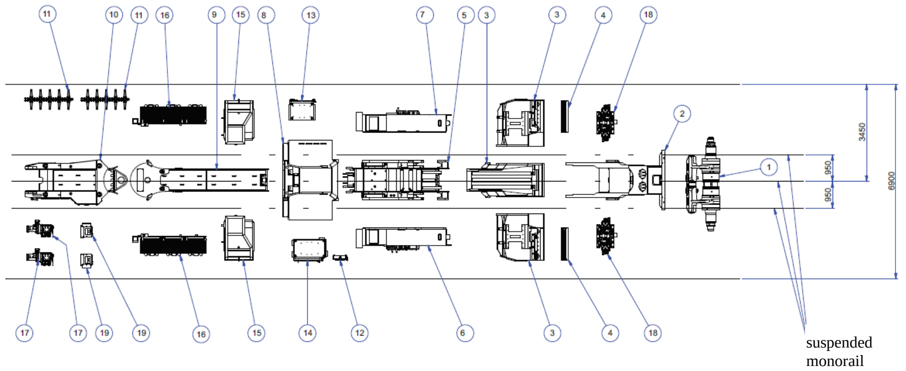

Fig. 11.1. Subassemblies of the JOY 12CM30 machine [130]

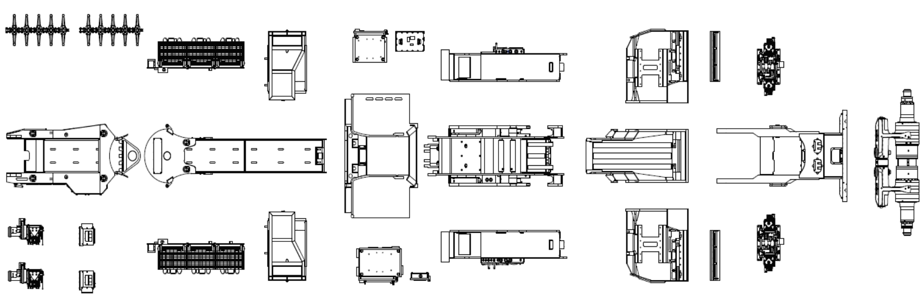

Fig. 11.2. Preparation method of subassemblies of the JOY 12CM30 machine for an assembly [130]

Specification of weights of individual subassemblies of the JOY 12CM30 machine [130]

Table 11.1.

Item	Subassembly	Weight [kg]
1	**Main subassemblies of the machine**	
1.1	Total machine	92 985
1.2	Haulage system	44 405
1.3	Conveyor assembly	9 880
1.4	Assembly of cutting arm	21 425
1.5	Assembly of cowls	14 145
2	**Cutting arm**	
2.1	Cutting arm	7 395
2.2	Arm gear and cutter bars of the cutting head	13 335
2.3	Conduits of the cutting arm	695
3	**Conveyor**	
3.1	Conveyor frame	7 985
3.2	Conveyor chain	1 405
3.3	Rack for mesh	490

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4	**Roof drilling rigs and shields**	
4.1	Roof drill	2 405
4.2	Shield	9 165
4.3	Arm, bracket	45
4.4	Step	40
5	**Haulage assembly**	
5.1	Main frame of haulage	35 250
5.2	Wall drill	1 405
5.3	Drill control	220
5.4	Oil tank	225
5.5	Conduits	2 085
5.6	Shield of conduits	105
5.7	Covers of fans	200
5.8	Railings	75
5.9	Platform for operator’s translocation	575
5.10	Operator’s wall shields	110
5.11	Operational platform	715
5.12	Ram of cutting arm lifting	580
6	**Chassis assembly**	
6.1	Chassis	7 895
6.2	Head articulated joint	870
6.3	Stabilization support	690
6.4	Operator’s safety shield	410
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6.6	Left cover of haulage frame	665
6.7	Right cover of haulage frame	1 320
6.8	Fender beams	195
6.9	Bumpers	7 290

After having finished the assembly procedure, before the first start-up of the machine, the following activities had to be done [130]:

– to check a correctness of the machine assembly, paying special attention to indispensable shields, platforms, emergency switches, tightness of the hydraulic system, a correct switching on of the feeder cable and protecting the feeder cable tension,
– to check a conformity of the voltage on the machine with the data given on the data plate,
– to check a correctness of the phases direction and of the motors rotation,
– to check a correctness of oil levels indications,
– to check refilling of lubrication points,
– to conduct a complex control of all the functions and safety devices during the start-up. In the case of detecting a possibility of the machine damage, it should be switched off and the failure should be rectified.

11.2. Operational technology of the JOY 12CM30 machine

During cutting operations with use of the JOY 12CM30 machine, the web depth should be established on the basis of the designed spacing of the working main support. In relation to that, depending on the support spacing, the web depth, executed in one machine operational cycle, is 0.6; 0.8 or 1.0 m [130].

To ensure a long-lasting and failure-free operation of the machine, the recommendations included in the operational manual, should be observed such as [130]:

– paying attention to a correct way of the machine use and appropriate operational conditions,
– the machine should be kept clean, which will enable to spot possible damages, special attention should be paid to the area of terminals of hydraulic hoses,
– during the machine standstill the system of external spraying should be switched off,
– the machine passage should be realized with the cutting assembly in the axial position, in parallel to the floor.

Each time, before starting the machine operation, the operator together with the fitter on duty and with the electrician should perform the following activities [130]:

– to check the machine as regards external damages which can impede, hinder its operation or have an impact on operational safety,
– to check, an operation of individual subassemblies of the machine, paying special attention to the components of the caterpillar chassis, of the scraper conveyor and of the intermediate conveyor chain,
– to check the condition of cutting picks and of their fixation; worn out or damaged picks should be replaced for ensuring optimum cutting conditions, the difference in the height of individual picks must be smaller than 10 mm,
– to check the condition of the hydraulic system as regards an occurrence of leakages,
– to check the oil level in the tank,
– to check all the elements of the electric equipment,
– to check an operational correctness of hydraulic and electric control systems,
– to check a permeability of spraying nozzles.

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After having performed the above mentioned activities and after having stated that in the area of the loader and of the cutting head no people are present, the operator can start the machine according to the operational manual [130]. The machine has three start-up modes [130]:

- **the radio mode** – it is the basic machine mode of operation while cutting, in this mode the machine can be operated with use of a cable or a radio signal, applying a remote control device; all the motors and hydraulic functions are switched on,
- **the local mode** – it is the mode, ensuring an alternative connection with the machine in the case of a failure of the series radio port; the connection is realized due to connecting the remote control device with the local port using a cable or due to connecting the receiver to the local port; all the motors and hydraulic functions are switched on, however the machine haulage is limited to the first speed,
- **the maintenance mode** – only the pump motor can be started in this mode, enabling to refill oil in the hydraulic tank; all the other motors and hydraulic functions are switched off.

During the machine operation the following recommendations of the BHP – occupational Health and safety should be observed [130]:

- only workers who underwent appropriate trainings and have authorizations should be approved for an independent machine operation,
- all the machine repairs should be conducted by electricians and fitters who have a sufficient knowledge of the machine construction,
- during repairs the machine should be disconnected from the supply voltage in the way hindering its accidental connection,
- an assembly and a disassembly of the machine subassemblies should be conducted exclusively in the supported part of the face,
- all the persons, working in the face, must be acquainted with the method of the machine stopping both in normal conditions as well as in emergency conditions,
- all the persons, working in the face, must be acquainted with the established system of warning signals.

In Fig. 11.3 and 11.4 access zones, in which the team of workers can be present during individual cycles of the machine operation, are presented.

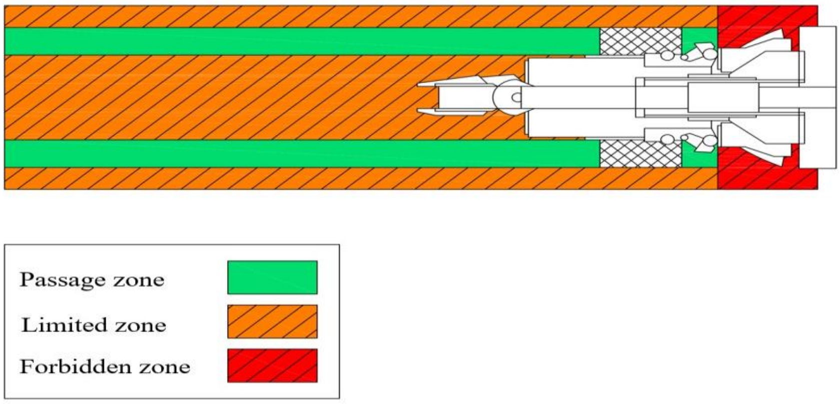

Fig. 11.3. Access zones during an operation of bolters [130]

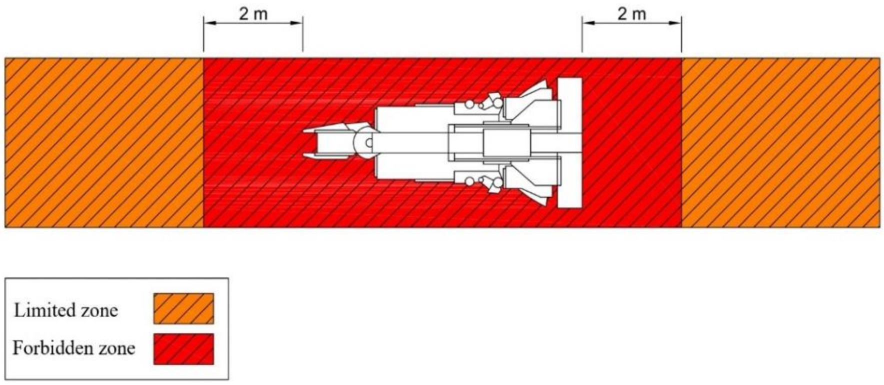

Fig. 11.4. Access zones during the machine advance [130]

11.3. Method of run-of-mine haulage

The run-of-mine haulage from the working is carried out with use of belt transport. In the Phase I of the working drivage (the first simple segment), the run-of-mine from the machine was supplied to the belt feeder of the SIGMA PDT 1000 in the suspended version and then to the belt conveyor Pioma 1200 No. 3. The run-of-mine from Bw-1n test roadway was supplied to the conveyor of the Bogda 1200 type, installed in the B transport roadway, then to the conveyor of the Pioma type No. 1 in the B-1 transport incline and further on to the main transport system towards the Shaft I [135].

The route of the SIGMA PDT 1000 conveyor in the suspended version, collecting the run-of-mine directly from the machine, was suspended to load-carrying vehicles moving on the rails of the suspended monorail. An application of this design construction enabled a continuous advance of the conveyor, following an advance of the face front and a cyclic elongation of the following means of transport [135].

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The beginning of **Phase II** of the working drivage (bend No. 1) was connected with a configuration change of the haulage devices. After having reached the minimum operational clearance gauge at the right wall by the suspended conveyor SIGMA PDT 1000, its shortening was needed at a simultaneous maximum elongation of the Pioma 1200 No. 3 conveyor. Besides, for improving the bend passage, the end of the machine feeder was deflected to the left side. After having driven several dozen meters of the working behind the bend between the suspended PDT SIGMA 1000 conveyor and the Pioma 1200 No. 3 conveyor, the PDT SIGMA 1000 conveyor in the stationary version [135], was installed.

In **Phase III** of the working drivage (straight segment No. 2) both conveyors of the SIGMA PDT 1000 type were elongated gradually. After having reached the maximum length of them, the assembly of the Pioma 1200 No. 4 belt conveyor and the MAMBA self-walking return end were started [135].

A configuration of the haulage equipment in **Phase IV** of the drivage (bend No. 2) was similar to the configuration applied in Phase II [135].

A configuration of haulage equipment, after having made the bend No. 2, is presented in Fig. 11.5.

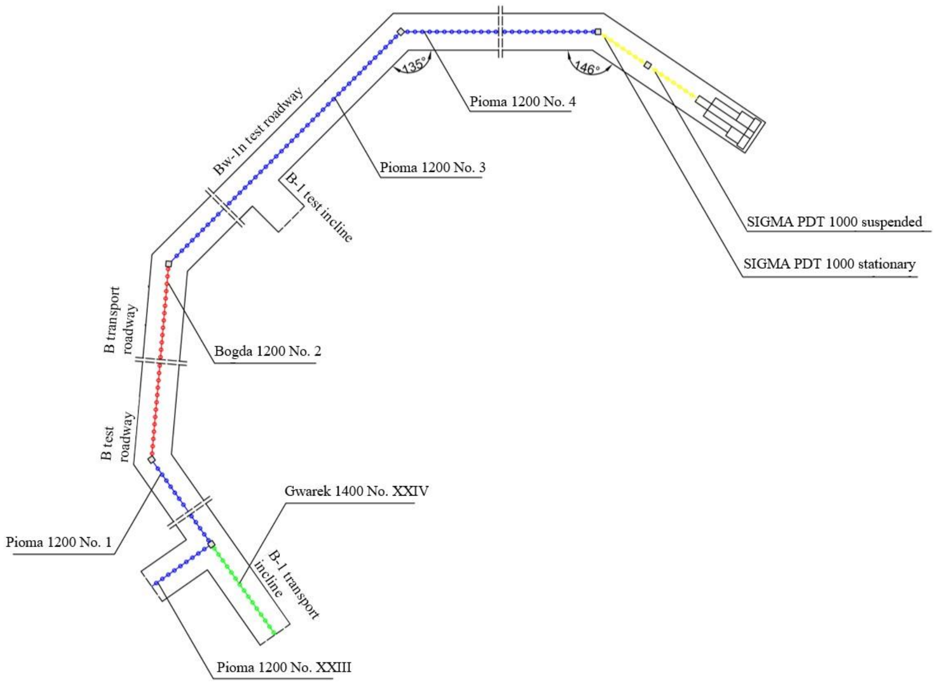

Fig. 11.5. Scheme of haulage equipment configuration after having made the bend No. 2

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During the working drivage at a longer life of face (in terms of distance), to improve the process of elongating the route of conveyors, the UPT device for pulling the return end of belt conveyors was used. This device can be also used for pulling elements of different type on the floor on condition that it is attached to a stationary fixation point. The UPT device is built of the following subassemblies [135]:

– a frame,
– a load carrying beam,
– a hydraulic hoist.

The frame is the load-carrying construction, ensuring an appropriate stiffness of the device. The other subassemblies are installed inside it. In the rear part of the frame there are holders for attaching the device to the return end. The load-carrying beam is a subassembly, on which a hydraulic hoist is installed. The hoist hook is clipped to the pin of the beam, situated in two sockets, attached to the frame with screws. The UPT device is connected with the conveyor return end with use of chain strands, attached to the holders in the rear part of the device frame to the return end frames [135].

Scheme of the UPT device construction is presented in Fig. 11.6. Pulling method of the conveyor return end by the UPT device is shown in Fig. 11.7.

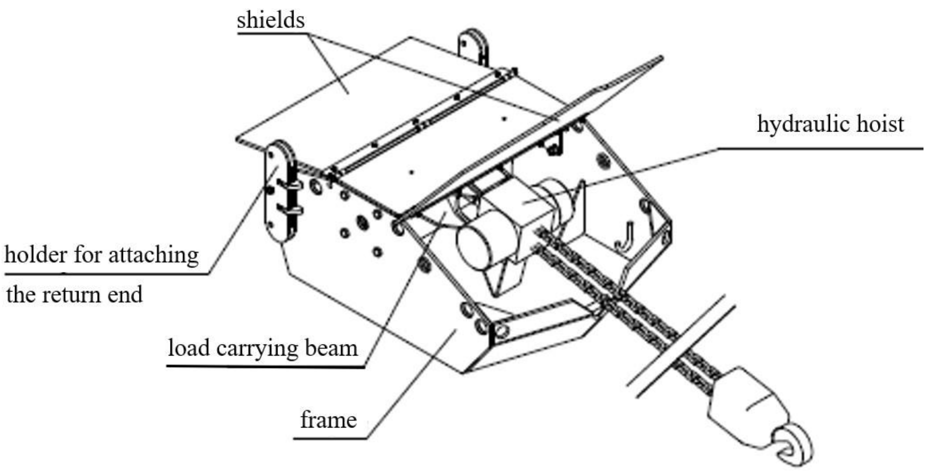

Fig. 11.6. Scheme of the UPT device construction [135]

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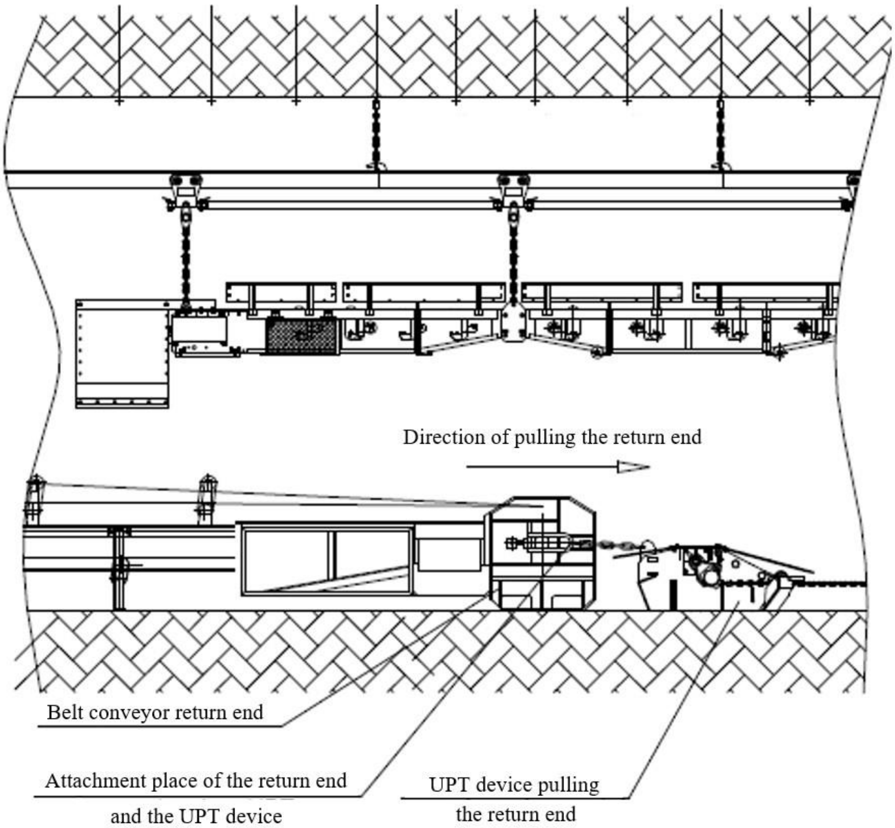

Fig. 11.7. Pulling method of the conveyor return end with use of the UPT device [135]

11.4. Installation method of the working support

A correct installation of the independent roof bolting support in the Bw-1n test roadway in Seam 401 requires an observation of the following principles [138]:

- Roof bolting should be installed by a trained team supervised by an authorized person from the group of mining officers. Before starting the activities the team of workers should get acquainted with the designed scheme of bolting, with the technology of installing the roof bolting support and with the safety principles.
- Roof bolting should be conducted with use of bolts and accessories, whose characteristics are in accordance with technical designs of the working drivage in individual phases. Bolts and accessories should have current approvals for an application in underground workings of hard coal mines.
- Transportation, storage and assembly of bolts and accessories should be in compliance with respective operational manuals of producers.

- Damaged, dirty and rusty bolts and accessories cannot be used. Special attention should be paid to the usability expiration date of glue charges.
- Drilling of bolt holes and an assembly of bolts should be carried out with use of efficient machines and equipment, having appropriate parameters and approvals for use in hard coal mines.
- The length of bolt holes should be from 50 to100 mm (for steel bolts) or 200 mm (for elastic bolts) shorter than the length of the bolt rod. After having drilled the bolt hole, it should be rinsed with water or cleaned with compressed air to remove drillings.
- The difference between the external diameter of the bolt rod and the bolt hole should be 4÷12 mm.
- Glue charges, including the quick-binding charges should be introduced into the drilled roof and wall holes. The number of glue charges should ensure full or minimum 90% filling of the hole after having introduced the bolt rod. After the time of binding of the quick-binding charge, given by the producer, the nut of the bolt should be tightened up on its thread, at previously installed lining mesh and washer. Tightening up should give the bolt an initial tension of the value not smaller than 30 kN.
- Roof bolting should be carried out directly after the working roof and wall exposition. The segment of the working in the face front should not be left unsupported during the days off work and when the break between shifts is longer than 8 hours.
- The roof should be protected with the lining mesh of the mesh eyes dimensions of 100 x 100 mm, strengthened from the side of the overlap under the bolts with an additional bar – mesh eyes 50 x 100 mm. The width of the lining mesh should correspond to the working width, and its longitudinal dimension – to the roof bolts spacing increased by 0.3 m of the overlap. For example in the case of the working width of 5.6 m and the spacing of bolts 0.8 m, the dimension of the lining mesh are 5.6 x 1.1 m.
- Walls should be protected by the lining mesh of the eye dimensions 100 x 100 mm, the lining mesh width should be by 0.4 m smaller than the working height, however its length should correspond to the spacing of the wall bolts increased by 0.2 m of the overlap. For example in the case of the working height of 4.0 m and of the bolts spacing of 0.8 m the dimensions of the lining mesh are 3.6 x 1.0 m.
- Lining meshes must be shaped and installed in the way enabling their tight adhesion to the working breakout.
- The standing support should be installed manually through lifting the canopy to the roof and an installation of the SVt or Valent props in compliance with

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the right technology. The standing support should be installed in the distance not exceeding 30 m from the face front.

– Roof bolts of the support basic scheme cannot be used for lifting and suspending heavy (above 10 kN) machines, equipment or their components or for an installation of the rails of the suspended monorail. Roof bolts of the support basic scheme can be used, however, for suspending light elements such as e.g. air ducts, pipelines, conduits and power cables.
– In the case of a necessity of applying a suspended monorail or a transportation of materials, additional bolts of the fixing length in the rock mass not smaller than 2.4 m, of the load-bearing capacity of at least 260 kN and the spacing complying with their designed maximum load, should be installed.
– In the case of sudden obstacles, occurring in the face, e.g. in a form of roof falls or other abnormalities (exceeding of critical states, sudden dynamics change of stratification increase and so on) more dense roof bolting, additional strethenings or a withdrawal of workers should be applied immediately according to the decision of the supervisory personnel. The responsible persons from the Central Mining Institute should also be informed.

Under the circumstances of the roof conditions deterioration, causing a generation of caverns and irregularities of surface in the roof, the following solutions were elaborated:

– an installation of roof bolting support in advance,
– an installation of standing support in advance.

An installation of the roof bolting support in advance should be carried out in compliance with the following principles:

– directly after opening the roof by the machine, the roof should be protected by bolting without the lining mesh,
– the bolts should be installed not in accordance with the existing bolting grid but in accordance with current needs and conditions,
– the machine cutting head should be lowered to the lowest possible position and it should be slumped-in in the way enabling the machine to get as close to the face as possible,
– due to uneven shape of a cavern an installation of lining mesh can be impossible, so the mesh should be installed as in the case of normal bolting and bolted to the roof in the places, where such a possibility exists (Fig. 11.8),
– in the places, where the lining mesh does not reach the level of the exposed roof, it should be bolted with use of additional bolts (Fig. 11.9),

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– free space should be fulfilled with timber or with a sack of *Big-bag* type, compressing it with a binder for fulfilling caverns.

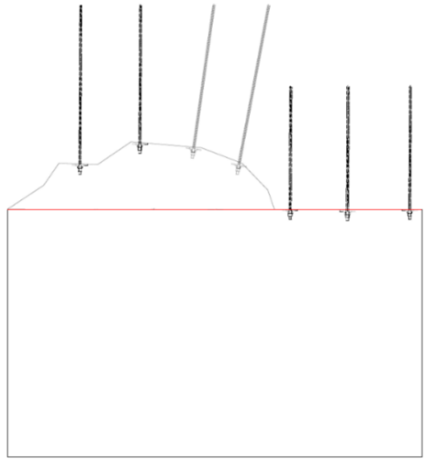

Fig. 11.8. Installation method of lining mesh in the case of caverns occurrence in the roof

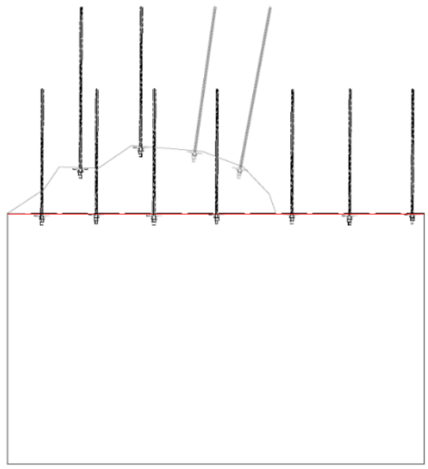

Fig. 11.9. Bolting method of lining mesh in the case of caverns occurrence in the roof

11.5. Parameters of the working support

Assembly chamber

The initial segment of the Bw-1n test roadway is the assembly chamber of the JOY 12CM30 machine, which was made earlier. Its overall dimensions [135] are as follows:

- length – 31.2 m,

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- maximum width – 7.2 m,
- maximum height – 5.3 m.

The support of the assembly chamber consists of 55 sets of steel frames made of V36 shapes, two parallel ones placed symmetrically under the roof, horseheads made of V32 shapes, a set of inter-frame sprags and 324 glue-bedded roof bolts APB (total length – 2.5 m; length of glue-bedding – 2.4 m; minimum load-bearing capacity – 260 kN) in the number of 6 units for each inter-frame field [135].

Phase I

The working driving process in Phase I was carried out in four stages of the length: 6.0; 16.0; 16.0 and 160 rm respectively, but in the fourth stage an independent roof bolting support was used for the first time. During all the four stages the working width was ~5600 mm and its height ~3400 mm. Parameters of the working support on the individual stages of Phase I are listed in Tables 11.2-11.5 [138]. The scheme of the working protection during the stages I-III is presented in Fig. 11.10.

Parameters of the working support in Stage I of Phase I of the Bw-1n test roadway driving [138]

Table 11.2.

	Stage I	
	Roof protection	Wall protection
Type of bolt	Steel glue-bedded bolt	Steel glue-bedded bolt
Total bolt length [m]	2.50	2.50
Bolt length in the rock mass [m]	2.40	2.40
Bolt diameter [mm]	21.7	21.7
Bolt load-bearing capacity [kN]	260	260
Spacing of bolts	6 bolts every 0.6 rm of the working	6 bolts every 0.6 rm of the working
Washer	Steel, square 150x150x8 mm	Steel, square 150x150x8 mm
Glue charges	Diameter 24 mm, length 1250 mm (500+750 mm)	Diameter 24 mm, length 1250 mm (500+750 mm)
Remarks	Additionally standing support was applied, made of frames of V29, straight connections supported at the walls with SVt steel props of minimum load-bearing capacity 250 kN and of 0.6 m spacing.	

Parameters of the working support in Stage II of Phase I of the Bw-1n test roadway driving [138]

Table 11.3.

	Stage II	
	Roof protection	Wall protection
Type of bolt	Steel glue-bedded bolt	Steel glue-bedded bolt
Total bolt length [m]	2.50	2.50
Bolt length in the rock mass [m]	2.40	2.40
Bolt diameter [mm]	21.7	21.7
Bolt load-bearing capacity [kN]	260	260
Spacing of bolts	6 bolts every 0.8 rm of the working	6 bolts every 0.8 rm of the working
Washer	Steel, square 150x150x8 mm	Steel, square 150x150x8 mm
Glue charges	Diameter 24 mm, length 1250 mm (500+750 mm)	Diameter 24 mm, length 1250 mm (500+750 mm)
Remarks	Additionally standing support was applied, made of frames of V29, straight connections supported at the walls with SVt steel props of minimum load-bearing capacity of 250 kN and of 0.8 m spacing.	

Parameters of the working support in Stage III of Phase I of the Bw-1n test roadway driving [138]

Table 11.4.

	Stage III	
	Roof protection	Wall protection
Type of bolt	Steel glue-bedded bolt	Steel glue-bedded bolt
Total bolt length [m]	2.50	2.50
Bolt length in the rock mass [m]	2.40	2.40
Bolt diameter [mm]	21.7	21.7
Bolt load-bearing capacity [kN]	260	260

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Spacing of bolts	6 bolts every 0.8 rm of the working	6 bolts every 0.8 rm of the working
Washer	Steel, square 150x150x8 mm	Steel, square 150x150x8 mm
Glue charges	Diameter 24 mm, length 1250 mm (500+750 mm)	Diameter 24 mm, length 1250 mm (500+750 mm)
Remarks	Additionally standing support was applied, made of frames of V29, straight connections supported at the walls with SVt steel props of minimum load-bearing capacity of 250 kN and of 1.6 m spacing.	

Parameters of the working support in Stage IV of Phase I of the Bw-1n test roadway driving [138]

Table 11.5.

	Stage IV	
	Roof protection	Wall protection
Type of bolt	Steel glue-bedded bolt	Steel glue-bedded bolt
Total bolt length [m]	2.50	2.50
Bolt length in the rock mass [m]	2.40	2.40
Bolt diameter [mm]	21.7	21.7
Bolt load-bearing capacity [kN]	260	260
Spacing of bolts	6 bolts every 0.8 rm of the working	6 bolts every 0.8 rm of the working
Washer	Steel, square 150x150x8 mm	Steel, square 150x150x8 mm
Glue charges	Diameter 24 mm, length 1250 mm (500+750 mm)	Diameter 24 mm, length 1250 mm (500+750 mm)

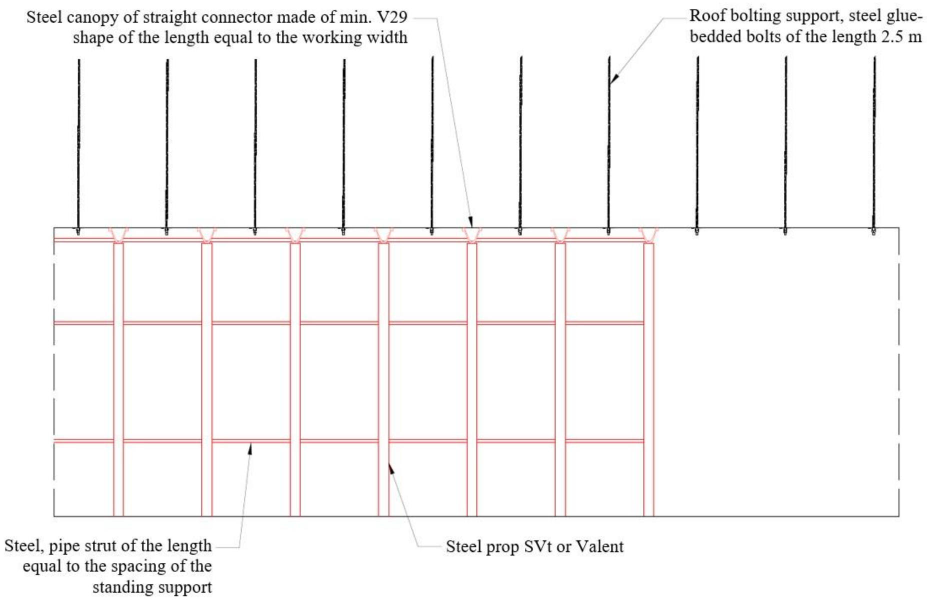

Fig. 11.10. Scheme of the working protection at Stages I-III of Phase I [138]

Phase II

Phase II of driving encompassed the bend No. 1 from 216 to 310 rm of the working. This bend changed the direction of the Bw-1n test roadway drivage from the northern-east to the eastern one. The driving process of the bend No. 1 was realized in three stages ~25.0; ~25.0 and ~45.0 rm respectively. During Stage I, at the segment of the length 10 m, a combination of roof bolting support and of standing support was applied. The parameters of the working cross-section were also diversified. A changeability of cross-section parameters in Phase II is presented in Table 11.6. Parameters of the working support, at individual stages of driving the bend No. 1, are presented in Tables 11.7-11.9. The method of installation and protection of the bend is illustrated in Fig. 11.11-11.13.

Changeability of the working overall dimensions during Phase II of driving the Bw-1n test roadway [129]

Table 11.6.

Stage	I	II	III
Width [mm]	~5600	~6800	~5600
Height [mm]	~4200	~4000	~4000

Parameters of the working support in Stage I of driving the bend No. 1 of the Bw-1n test roadway [129]

Table 11.7.

	Stage I		
	Roof protection		Wall protection
Type of bolt	Steel glue-bedded bolt	Long elastic bolt	Steel glue-bedded bolt
Total bolt length [m]	2.50	4.20	2.50
Bolt length in the rock mass [m]	2.40	4.00	2.40
Bolt diameter [mm]	21.7	18.0	21.7
Bolt load-bearing capacity [kN]	260	320	260
Spacing of bolts	6 bolts every 0.6 rm of the working	2.5 bolts every 0.6 rm of the working	8 bolts every 0.6 rm of the working
Washer	Steel, square 150x150x8 mm	Steel, square 200x200x12 mm	Steel, square 150x150x8 mm
Glue charges	Diameter 24 mm, length 1250 mm (500+750 mm)	Diameter 24 mm, length 2450 mm (450+2000 mm)	Diameter 24 mm, length 1250 mm (500+750 mm)
Remarks	At the segment of 10 m from the end of the straight segment of the roadway, additionally straight connections V29 were installed, supported with props of 0.6 m spacing and chocks, from the place of starting a roadway function in the wall under cutting, workable bolts were installed instead of steel ones of the load-bearing capacity of at least 180 kN.		

Parameters of independent roof bolting support in Stage II of driving the bend No. 1 of Bw-1n test roadway [129]

Table 11.8.

	Stage II		
	Roof protection		Wall protection
Type of bolt	Steel glue-bedded bolt	Long elastic bolt	Steel glue-bedded bolt
Total bolt length [m]	2.50	4.20	2.50
Bolt length in the rock mass [m]	2.40	4.00	2.40

Bolt diameter [mm]	21.7	18.0	21.7
Bolt load-bearing capacity [kN]	260	320	260
Spacing of bolts	8 bolts every 0.6 rm of the working	2.5 bolts every 0.6 rm of the working	8 bolts every 0.6 rm of the working
Washer	Steel, square 150x150x8 mm	Steel, square 200x200x12 mm	Steel, square 150x150x8 mm
Glue charges	Diameter 24 mm, length 1250 mm (500+750 mm)	Diameter 24 mm, length 2450 mm (450+2000 mm)	Diameter 24 mm, length 1250 mm (500+750 mm)

Parameters of independent roof bolting support in Stage III of driving the bend No. 1 of Bw-1n test roadway [129]

Table 11.9.

	Stage III	
	Roof protection	Wall protection
Type of bolt	Steel glue-bedded bolt	Steel glue-bedded bolt
Total bolt length [m]	2.50	2.50
Bolt length in the rock mass [m]	2.40	2.40
Bolt diameter [mm]	21.7	21.7
Bolt load-bearing capacity [kN]	260	260
Spacing of bolts	6 bolts every 0.6 rm of the working	8 bolts every 0.6 rm of the working
Washer	Steel, square 150x150x8 mm	Steel, square 150x150x8 mm
Glue charges	Diameter 24 mm, length 1250 mm (500+750 mm)	Diameter 24 mm, length 1250 mm (500+750 mm)

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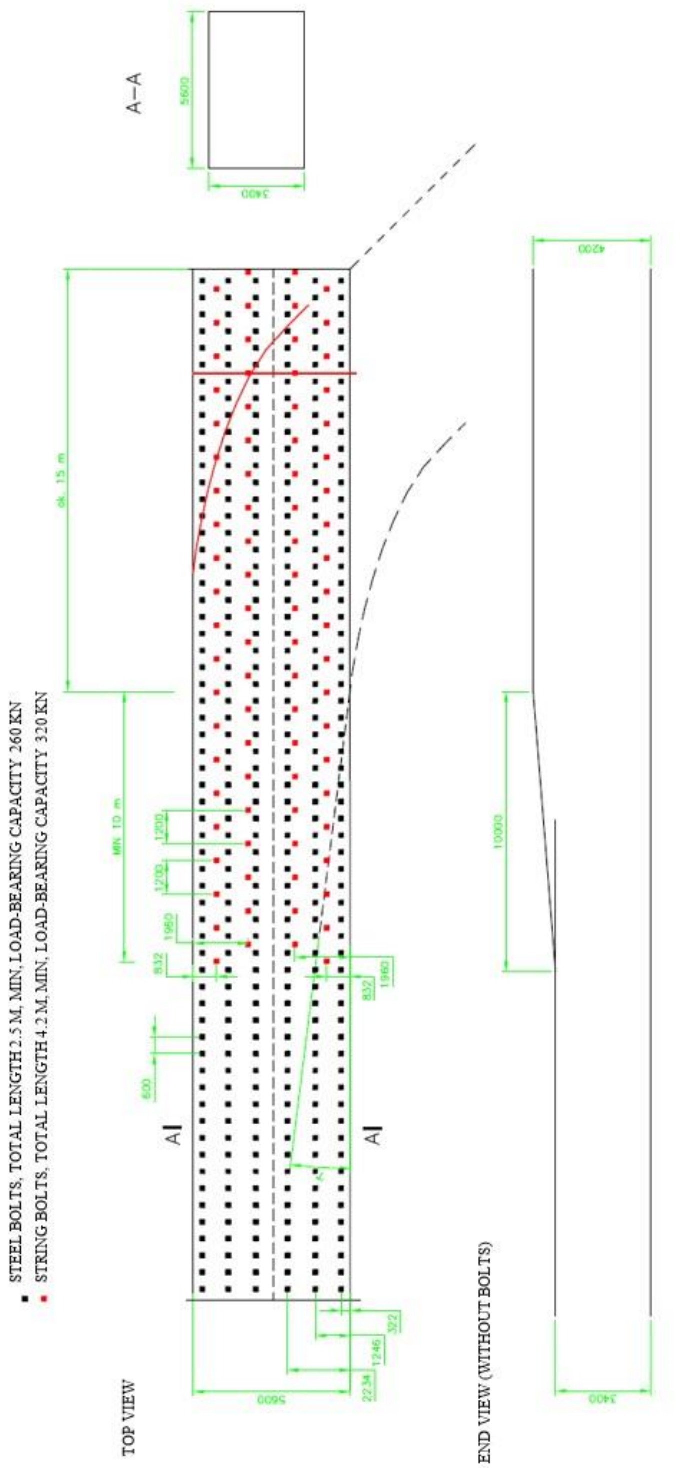

Fig. 11.11. Driving method of the bend No. 1 – Stage I at the Bw-1n test roadway [129]

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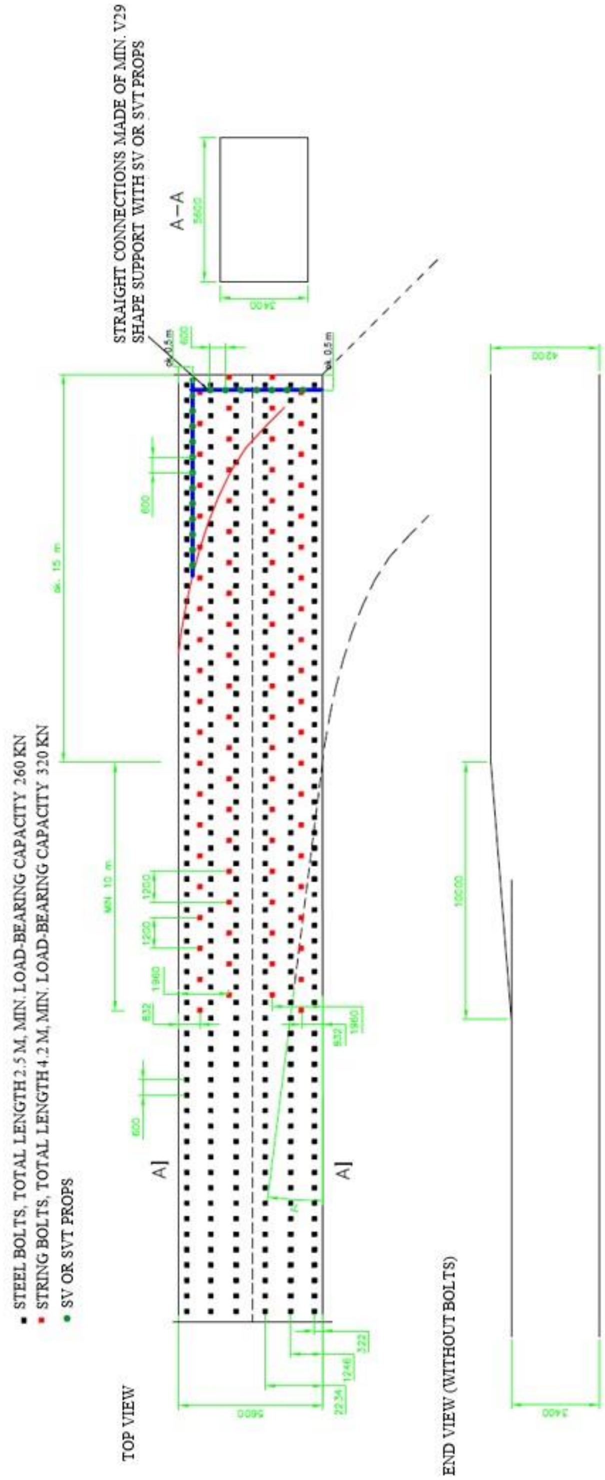

Fig. 11.12. Protection method of the bend No. 1 in the Bw-1n test roadway [129]

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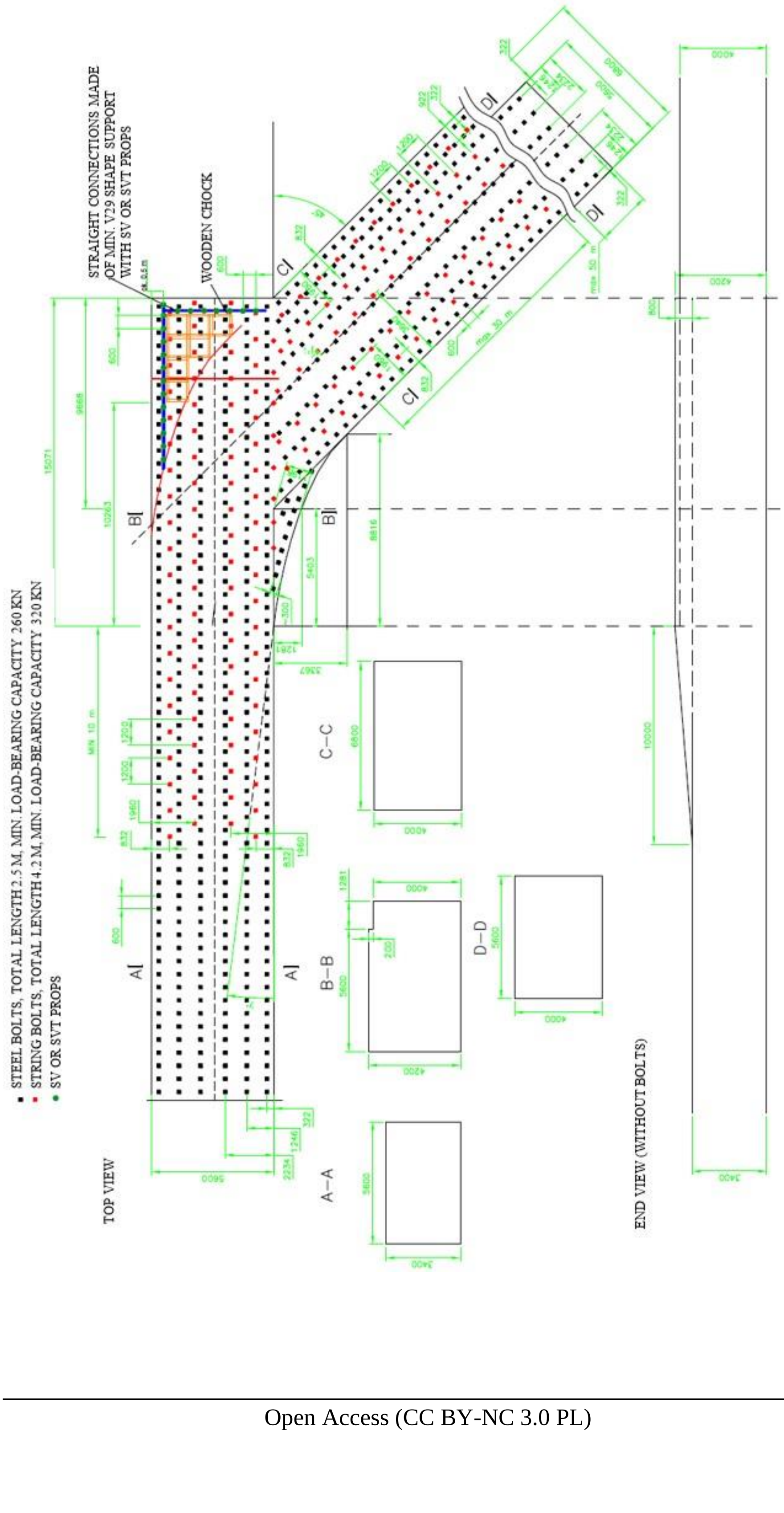

Fig. 11.13. Protection method of the bend No. 1 – Stage II in the Bw-1n test roadway [129]

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Phase III

Phase III encompassed driving of about 800 rm of the straight segment towards the eastern direction from the mark 310 rm to about 1100 rm. Finally, due to a deterioration of geological conditions, the Bw-1n test roadway, in Phase III, was driven to the mark 1044 rm. The working driving in Phase III consisted of three stages. In Stage I the working width was ~5600 mm and its height ~3400 mm. In Stage II an expansion was driven for an installation of the transformer station of the length 10 rm. At this stage the working width reached ~6800 mm, and its height – 3200÷3400 mm. In Stage III the working dimensions were back to those from Stage I [139].

Parameters of the working support at individual stages of Phase III are listed in Tables 11.10-11.12. Bolting scheme of Stage II, before an introduction of changes resulting from a deterioration of geological conditions, is presented in Fig. 11.14.

Parameters of independent roof bolting support at Stage I of Phase III of driving Bw-1n test roadway [135, 139]

Table 11.10.

	Stage I	
	Roof protection	Wall protection
Type of bolt	Steel glue-bedded bolt	Steel glue-bedded bolt
Total bolt length [m]	2.50	2.50
Bolt length in the rock mass [m]	2.40	2.40
Bolt diameter [mm]	21.7	21.7
Bolt load-bearing capacity [kN]	260	260
Spacing of bolts	6 bolts every 0.8 rm of the working	6 bolts every 0.8 rm of the working
Washer	Steel, square 150x150x8 mm	Steel, square 150x150x8 mm
Glue charges	Diameter 24 mm, length 1250 mm (500+750 mm)	Diameter 24 mm, length 1250 mm (500+750 mm)
Remarks	On 25th March 2020 a visit to the working was paid. In the result of it very high values of the roof stratification were stated. On the basis of conducted observations it was decided to change the bolting scheme in relation to the one described above:	

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	▪ in the working axis additional elastic bolts of the length 5.0 m and the spacing of 2 bolts every 0.8 rm of the working were installed, ▪ at the working walls two additional steel bolts of the length 2.40 m were installed; finally the spacing of steel glue-bedded bolts in the roof was 8 bolts every 0.8 rm of the working. A new bolting scheme was applied from ~480,0 rm of the working.

Parameters of independent roof bolting support at Stage II of Phase III of driving the Bw-1n test roadway [135, 139]

Table 11.11.

	Stage II		
	Roof protection		Wall protection
Type of bolt	Steel glue-bedded bolt	Long elastic bolt	Steel glue-bedded bolt
Total bolt length [m]	2.50	4.20	2.50
Bolt length in the rock mass [m]	2.40	4.00	2.40
Bolt diameter [mm]	21.7	18.0	21.7
Bolt load-bearing capacity [kN]	260	320	260
Spacing of bolts	8 bolts every 0.6 rm of the working	2.5 bolts every 0.6 rm of the working	6 bolts every 0.6 rm of the working
Washer	Steel, square 150x150x8 mm	Steel, square 200x200x12 mm	Steel, square 150x150x8 mm
Glue charges	Diameter 24 mm, length 1250 mm (500+750 mm)	Diameter 24 mm, length 2450 mm (450+2000 mm)	Diameter 24 mm, length 1250 mm (500+750 mm)
Remarks	Due to a deterioration of mining-and-geological conditions a decision was taken to change the bolting scheme in relation to the one described above: ▪ a change of steel bolts spacing in the roof for 8+1 bolts every 0.8 rm of the working, (an additional bolt installed from the right wall side), ▪ a change of steel bolts spacing in the wall for 6 bolts every 0.8 rm of the working, ▪ a change of long elastic bolts spacing of the length 4.20 m for 2 bolts every 0.8 rm of the working,		

	▪ an application of additional long elastic bolts of the length 5.00 m and of the load-bearing capacity of 300 kN, in the spacing of 3 bolts every 0.8 m.

Parameters of independent roof bolting support in Stage III of Phase III of driving the Bw-1n test roadway [135]

Table 11.12.

	Stage III		
	Roof protection		Wall protection
Type of bolt	Steel glue-bedded bolt	Long elastic bolt	Steel glue-bedded bolt
Total bolt length [m]	2.50	5.00	2.50
Bolt length in the rock mass [m]	2.40	4.80	2.40
Bolt diameter [mm]	21.7	18.0	21.7
Bolt load-bearing capacity [kN]	260	300	260
Spacing of bolts	6+1 bolts every 0.8 rm of the working (additional bolt installed from the left wall side)	2 bolts every 0.8 rm of the working	6 bolts every 0.8 rm of the working
Washer	Steel, square 150x150x8 mm	Steel, square 200x200x12 mm	Steel, square 150x150x8 mm
Glue charges	Diameter 24 mm, length 1250 mm (500+750 mm)	Diameter 24 mm, length 2450 mm (450+2000 mm)	Diameter 24 mm, length 1250 mm (500+750 mm)
Remarks	This bolting scheme was elaborated with regard to a change of geological conditions based on a stratification control and on tests of pulling out bolts conducted during underground visits to the scene.		

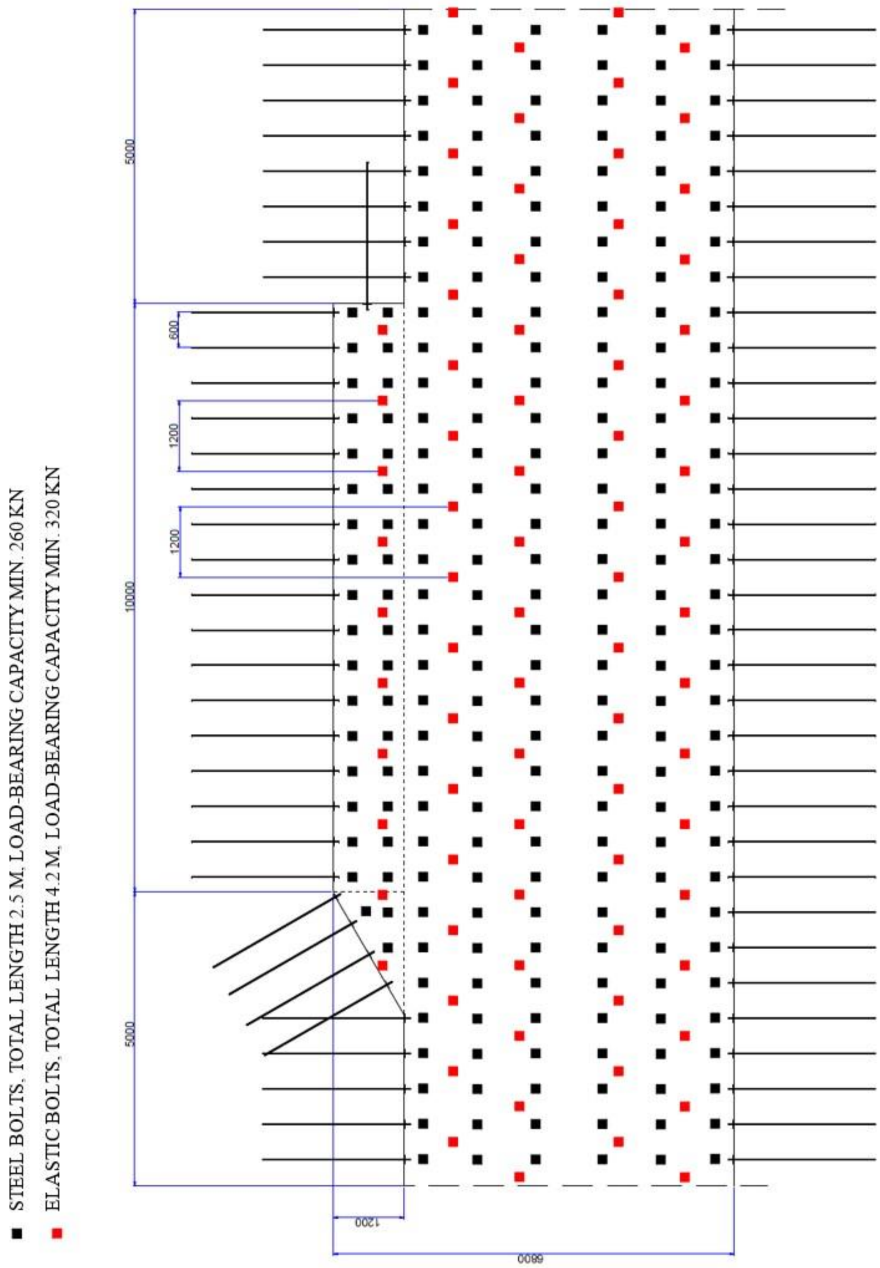

Fig. 11.14. Bolting scheme of an extension for the transformer station before an introduction of changes [135]

Phase IV

The Bw-1n test roadway was driven to the mark 1044 rm. With regard to a change of geological conditions it was decided to change a position of the bend No. 2. Based on an underground inspection on 20th July 2020, it was stated that the most optimum location of the bend, as regards geological and technological aspects, is on the mark 984-1009 rm. Finally it was decided to drive a junction on the 994 rm mark. The driven straight section, starting from the position of the planned junction was enclosed with a stopping. The process of driving the bend No. 2 was realized in stages of the lengths 30.0; 25.0 and 50.0 rm respectively. The parameters of the working cross-section were diversified. A changeability of the cross-section parameters in Phase II is presented in Table 11.13. The parameters of the working support at individual stages of the bend No. 2 driving are listed in Tables 11.14-11.16 [134, 137]. The bolting scheme of all the three stages of driving the bend is presented in Fig. 11.15-11.17. An in-situ view of the bend No. 2 is shown in Fig. 11.18 and 11.19.

Changeability of the working parameters in Phase III of driving the Bw-1n test roadway [134]

Table 11.13.

Stage	I	II	III
Width [mm]	~5600	~6800	~5600
Height [mm]	~4200	~4000	~4000

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Parameters of independent roof bolting support in Stage I of driving the bend No. 2 of the Bw-1n test roadway [134]

Table 11.14.

	Stage I			
	Roof protection			Wall protection
Type of bolt	Steel glue-bedded bolt	Long elastic bolt	Long elastic bolt	Steel glue-bedded bolt
Total bolt length [m]	2.50	4.20	5.00	2.50
Bolt length in the rock mass [m]	2.40	4.00	4.80	2.40
Bolt diameter [mm]	21.7	18.0	18.0	21.7
Bolt load-bearing capacity [kN]	260	320	320	260
Spacing of bolts	8 bolts every 1.0 rm of the working	2 bolts every 1.0 rm of the working (on the mark 984÷994 rm)	2 bolts every 1.0 rm of the working (on the mark 984÷994 rm); 5 bolts every 1.0 rm of the working (on the mark 994÷1014 rm)	8 bolts every 0.8 rm of the working
Washer	Steel, square 150x150x8 mm	Steel, square 200x200x12 mm	Steel, square 200x200x12 mm	Steel, square 150x150x8 mm
Glue charges	Diameter 24 mm, length 1250 mm (500+750 mm)	Diameter 24 mm, length 2450 mm (450+2000 mm)	Diameter 24 mm, length 2450 mm (450+2000 mm)	Diameter 24 mm, length 1250 mm (500+750 mm)
Remarks	Starting from the junction, in the wall under cutting the breaking bolts instead of the steel ones of the load-bearing capacity not less than 180 kN should be used.			

Parameters of independent roof bolting support in Stage II of driving the bend No. 2 of the Bw-1n test roadway [134]

Table 11.15.

	Stage II		
	Roof protection		Wall protection
Type of bolt	Steel glue-bedded bolt	Long elastic bolt	Steel glue-bedded bolt
Total bolt length [m]	2.50	5.00	2.50
Bolt length in the rock mass [m]	2.40	4.80	2.40
Bolt diameter [mm]	21.7	18.0	21.7
Bolt load-bearing capacity [kN]	260	320	260
Spacing of bolts	8 bolts every 0.8 rm of the working	5 bolts every 0.8 rm of the working	8 bolts every 0.8 rm of the working
Washer	Steel, square 150x150x8 mm	Steel, square 200x200x12 mm	Steel, square 150x150x8 mm
Glue charges	Diameter 24 mm, length 1250 mm (500+750 mm)	Diameter 24 mm, length 2450 mm (450+2000 mm)	Diameter 24 mm, length 1250 mm (500+750 mm)

Parameters of independent roof bolting support in Stage III of driving the bend No. 2 of the Bw-1n test roadway [134]

Table 11.16.

	Stage III		
	Roof protection		Wall protection
Type of bolt	Steel glue-bedded bolt	Long elastic bolt	Steel glue-bedded bolt
Total bolt length [m]	2.50	5.00	2.50
Bolt length in the rock mass [m]	2.40	4.80	2.40
Bolt diameter [mm]	21.7	18.0	21.7
Bolt load-bearing capacity [kN]	260	300	260
Spacing of bolts	8 bolts every 0.8 rm of the working	2 bolts every 0.8 rm of the working	8 bolts every 0.8 rm of the working
Washer	Steel, square 150x150x8 mm	Steel, square 200x200x12 mm	Steel, square 150x150x8 mm
Glue charges	Diameter 24 mm, length 1250 mm (500+750 mm)	Diameter 24 mm, length 2450 mm (450+2000 mm)	Diameter 24 mm, length 1250 mm (500+750 mm)

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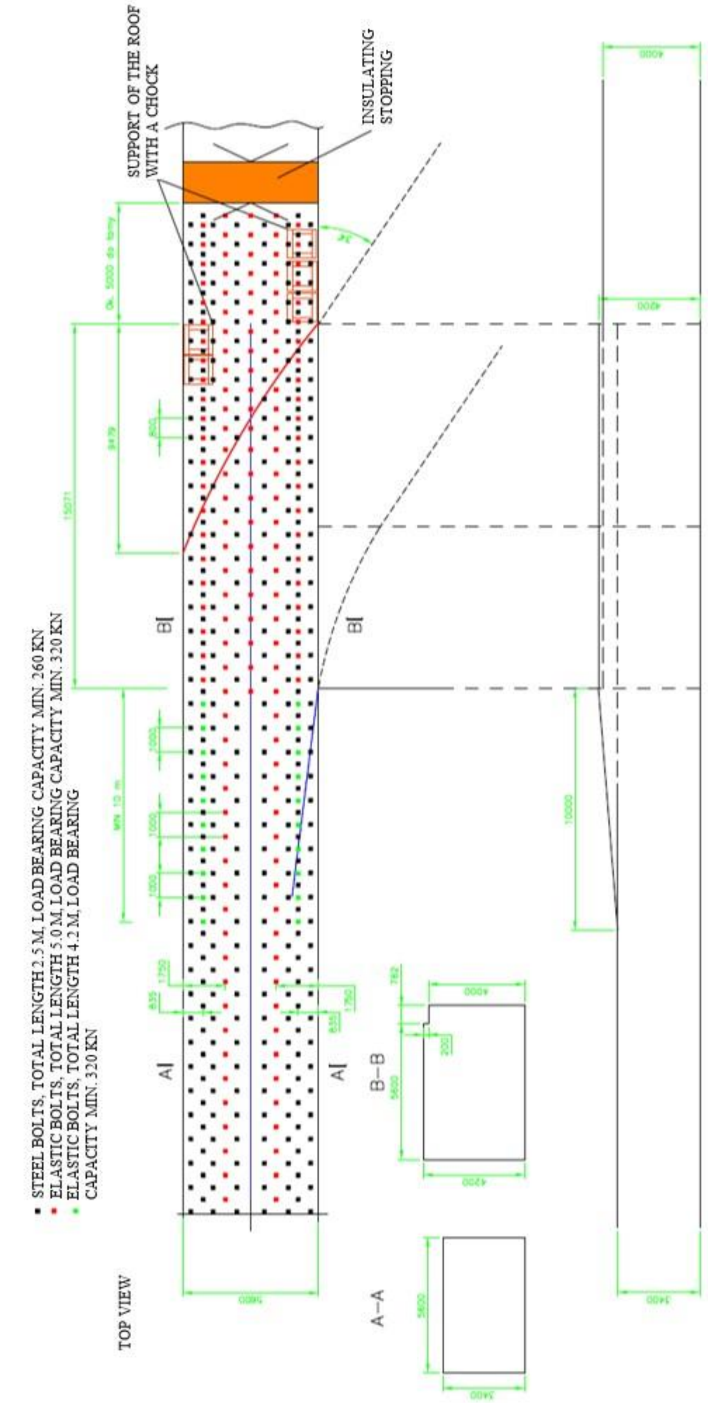

Fig. 11.15. Driving method of the bend No. 2 in the Bw-1n test roadway – Stage I [134]

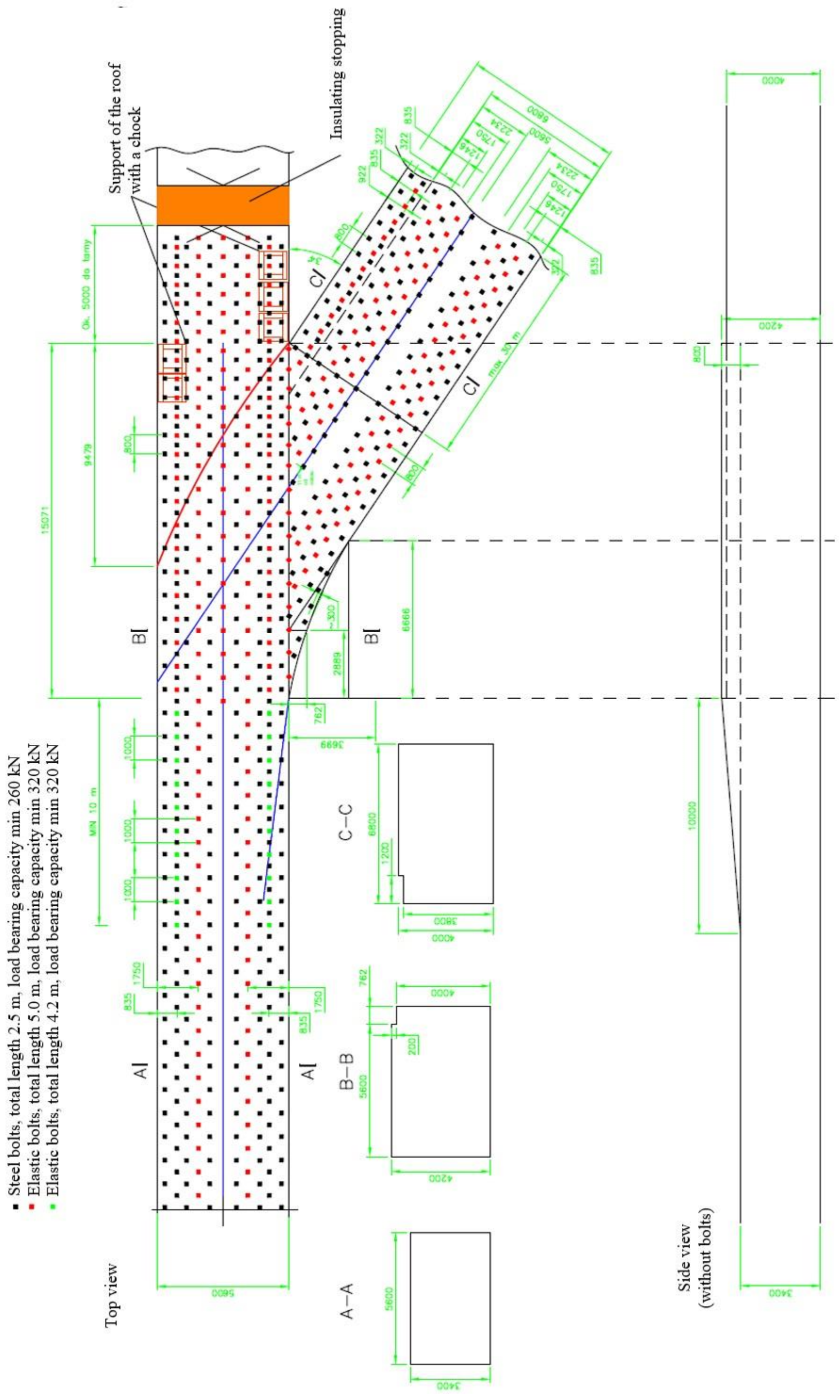

Fig. 11.16. Driving method of the bend No. 2 in the Bw-1n test roadway – Stage II [134]

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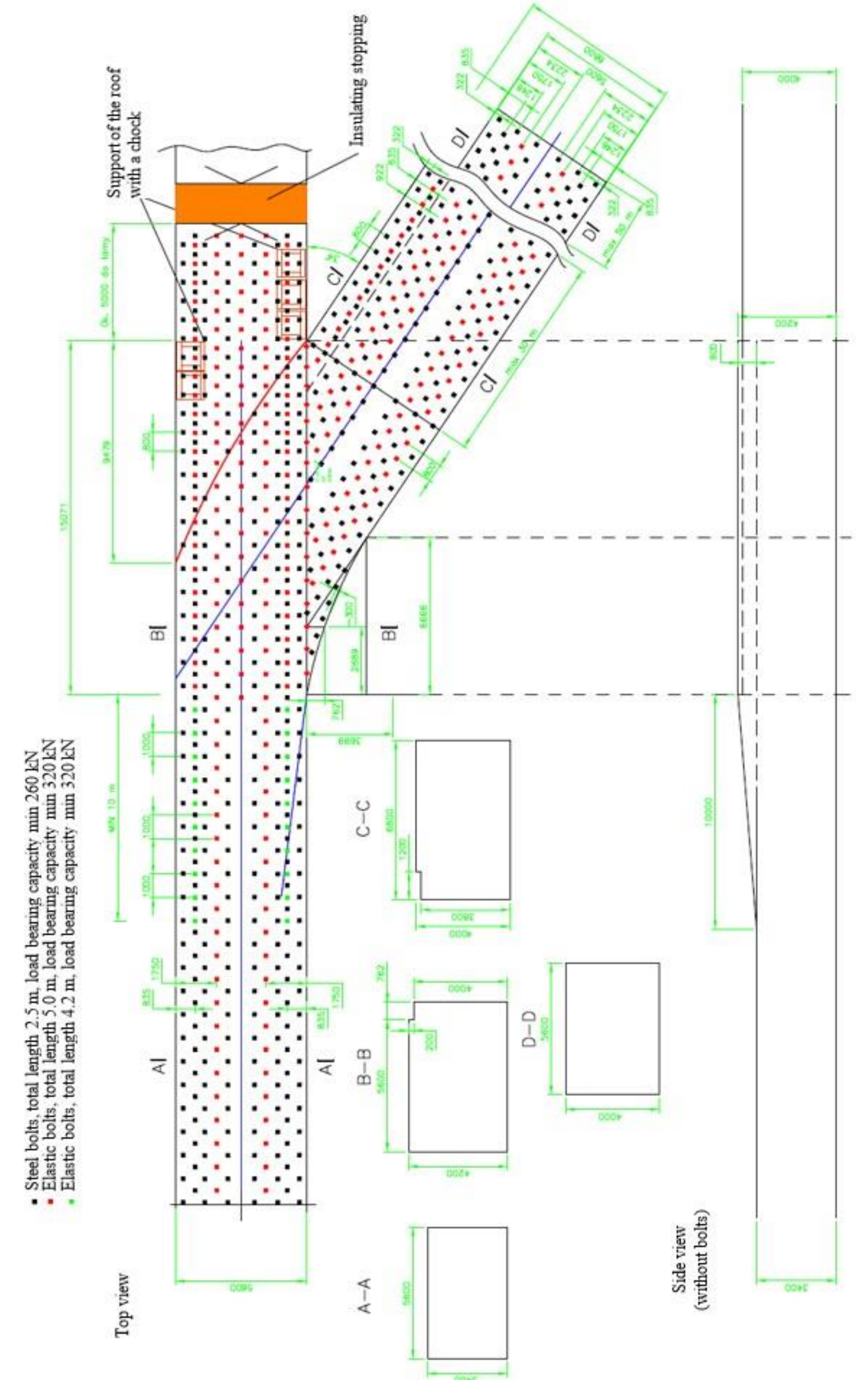

Fig 11.17. Driving method of bend No. 2 in the Bw-1n test roadway – Stage III [134]

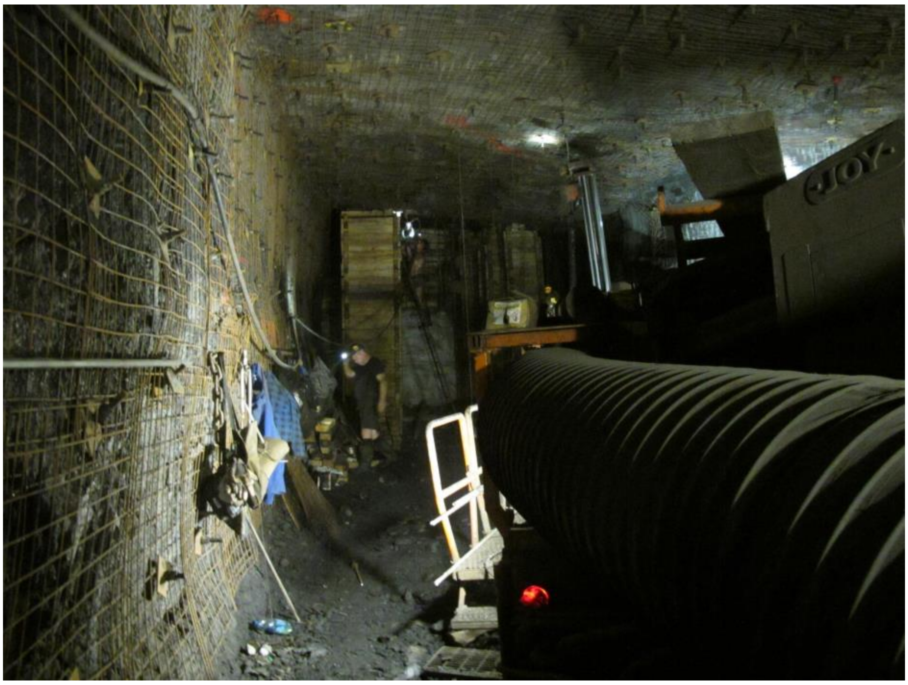

Fig. 11.18. Driving the bend No. 2 - chocks [132]

Fig. 11.19. Driving the bend No. 2 – a fragment at the right wall [132]

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An approximate number of bolts in the basic bolting scheme, falling to 1 rm of the working at individual stages of the Bw-1n roadway drivage, is presented in Table 11.17 and in Fig. 11.20 and 11.21.

Number of bolts falling to 1 rm of the working at individual stages of the Bw-1n test roadway drivage

Table 11.17.

<table>
<tr><th>Phase</th><th colspan="2">Stage</th><th>Steel bolts in the roof [pcs./1 rm]</th><th>Steel bolts in the wall [pcs./1 rm]</th><th>Long bolts 5.0 m [pcs./1 rm]</th><th>Long bolts 4.2 m [pcs./1 rm]</th></tr>
<tr><td rowspan="4">I</td><td colspan="2">I</td><td>10.00</td><td>10.00</td><td>0.00</td><td>0.00</td></tr>
<tr><td colspan="2">II</td><td>7.50</td><td>7.50</td><td>0.00</td><td>0.00</td></tr>
<tr><td colspan="2">III</td><td>7.50</td><td>7.50</td><td>0.00</td><td>0.00</td></tr>
<tr><td colspan="2">IV</td><td>7.50</td><td>7.50</td><td>0.00</td><td>0.00</td></tr>
<tr><td rowspan="3">II</td><td colspan="2">I</td><td>10.00</td><td>13.33</td><td>0.00</td><td>4.17</td></tr>
<tr><td colspan="2">II</td><td>13.33</td><td>13.33</td><td>0.00</td><td>4.17</td></tr>
<tr><td colspan="2">III</td><td>10.00</td><td>13.33</td><td>0.00</td><td>0.00</td></tr>
<tr><td rowspan="4">III</td><td rowspan="2">I</td><td>Segment I</td><td>7.50</td><td>7.50</td><td>0.00</td><td>0.00</td></tr>
<tr><td>Segment II</td><td>10.00</td><td>7.50</td><td>2.50</td><td>0.00</td></tr>
<tr><td colspan="2">II</td><td>11.25</td><td>7.50</td><td>3.75</td><td>2.50</td></tr>
<tr><td colspan="2">III</td><td>11.25</td><td>7.50</td><td>2.50</td><td>0.00</td></tr>
<tr><td rowspan="3">IV</td><td colspan="2">I</td><td>8.00</td><td>10.00</td><td>2.00</td><td>2.00</td></tr>
<tr><td colspan="2">II</td><td>10.00</td><td>10.00</td><td>6.25</td><td>0.00</td></tr>
<tr><td colspan="2">III</td><td>10.00</td><td>10.00</td><td>2.50</td><td>0.00</td></tr>
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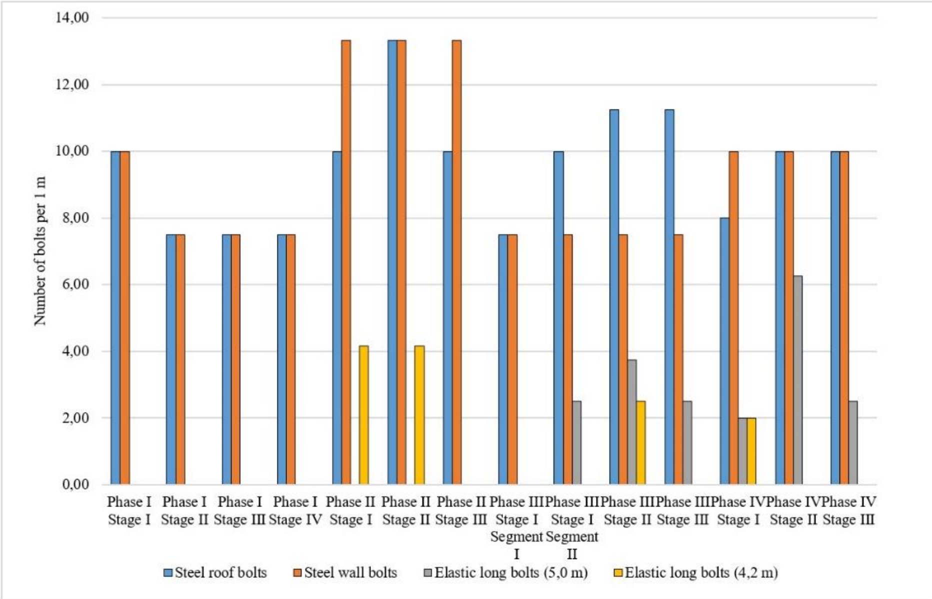

Fig. 11.20. Number of bolts falling to 1 rm of the working at individual driving stages of the Bw-1n test roadway

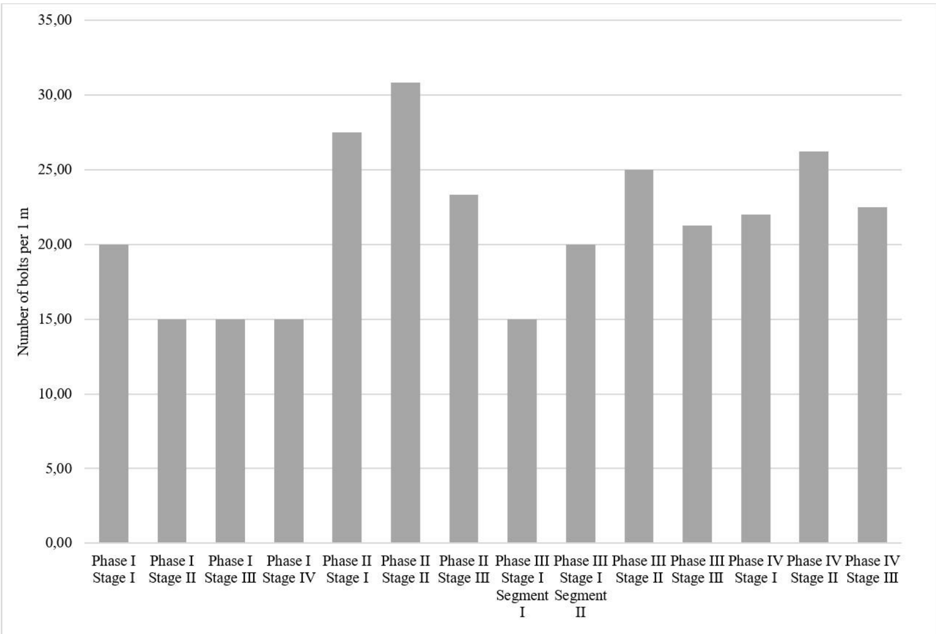

Fig. 11.21. Total number of bolts falling to 1 rm of the working at individual driving stages of the Bw-1n test roadway

11.6. The working ventilation system

The Bw-1n test roadway is ventilated with use of a separate, combined air pipes with auxiliary sucking air ducts, consisting of the following devices [135]:

- WLE-12132B/E fan,
- elastic air pipes Ø1400 mm (to the mark 675 rm),
- elastic air pipes Ø1200 mm (from the mark 675 rm),
- a bunker air pipe Ø1000 mm of the length 20 m installed at the terminal segment of the pressure air duct,
- a segment of the air duct of the length 10 m at the working face, built of spiral air pipes and equipped with a device for changing the air direction (WIR-1000P),
- an internal dust collector of the JOY 12CM30 machine.

In Table 11.18 basic technical parameters of the WLE-12132B/E fan and of the air duct Ø1400 mm are presented. In Fig. 11.22 a location of the Ø1400 mm air pipe in the working is shown. In Fig. 11.23 a location of the WIR-1000P air pipe in the working is presented.

Basic technical parameters of the WLE-12132B/E fan and of the air duct Ø1400 mm [135]

Table 11.18.

Item	Parameter	Unit	Size
Air duct Ø1400 mm			
1	Efficiency	%	90.81
2	Air velocity	m/s	0.38
3	Amount of air	m^3/s	7.20
4	Output – flow rate	m^3/s	7.93
WLE-12132/E II fan			
1	Output – flow rate	m^3/s	21.52
2	Ram effect	Pa	7 308
3	Amount of air in the face	m^3/s	19.54
4	Amount of the flowing round air	m^3/s	25.68

 https://doi.org/10.32056/KOMAG/Monograph2021.2

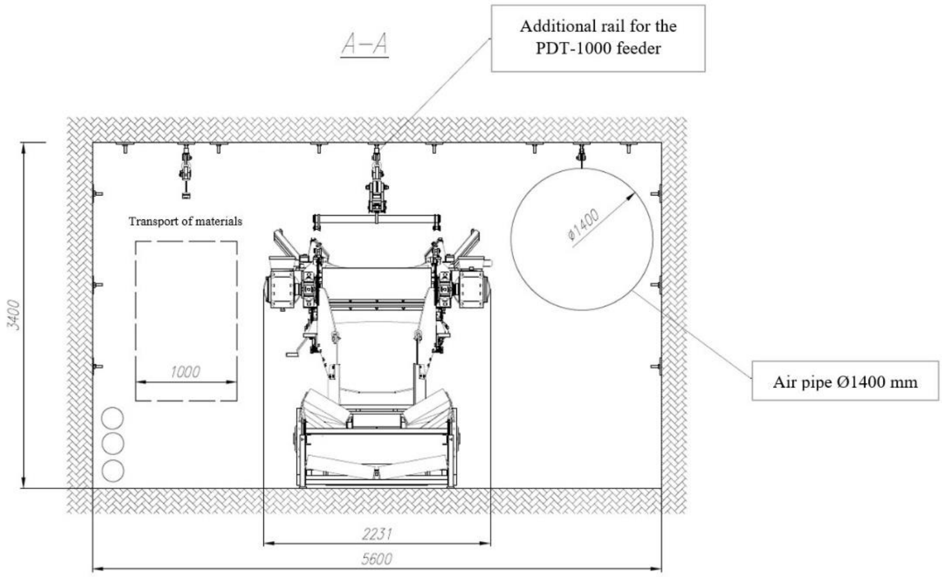

Fig. 11.22. Cross-section of the working with installed air pipe Ø1400 mm [135]

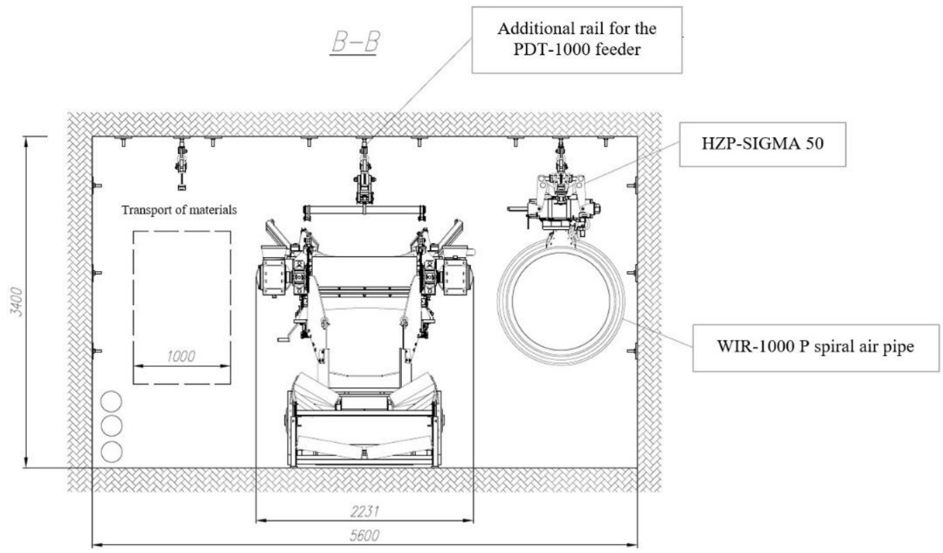

Fig. 11.23. Cross-section of the working with installed WIR-1000P air pipe [135]

The system of air supply and offtake from the Bw-1n test roadway is presented in Fig. 11.24.

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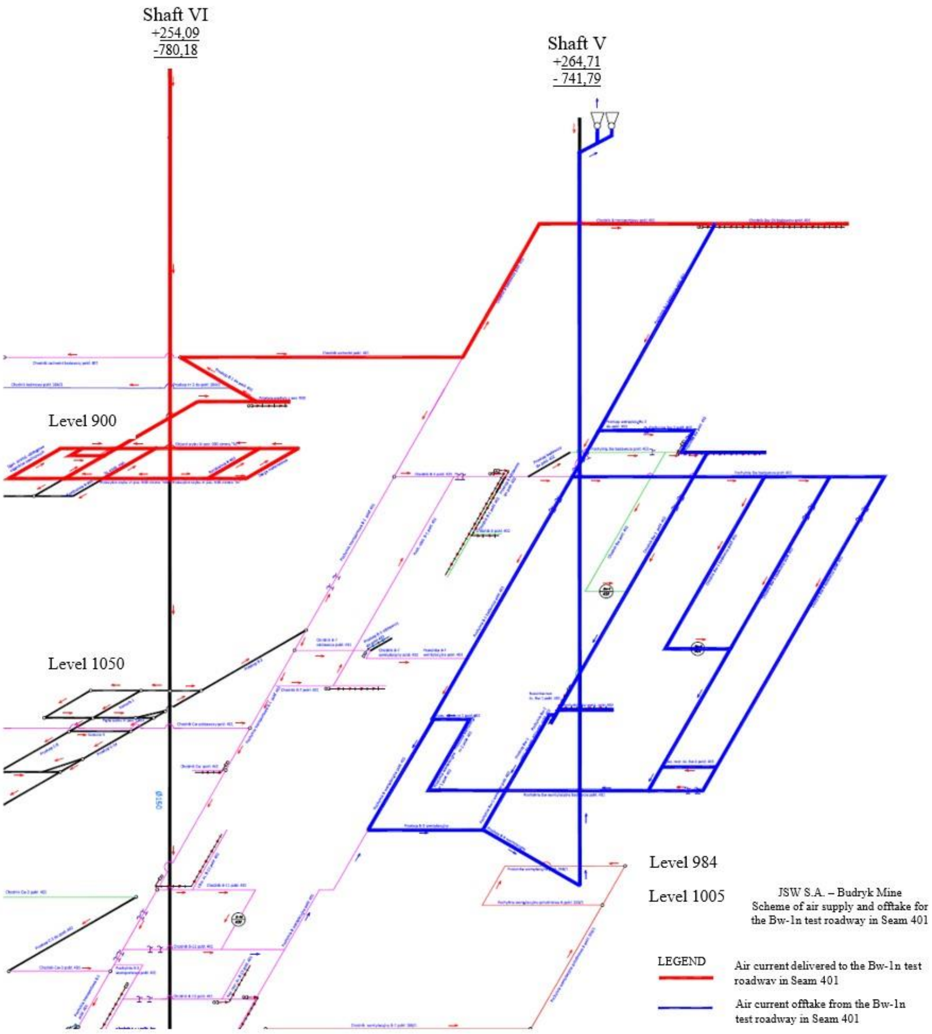

Fig. 11.24. System of air supply and offtake from the Bw-1n test roadway [135]

Air parameters in the working

Basic ventilation parameters and climatic conditions experienced during the first nine months of the project duration are listed in Table 11.19. A graph of temperature changes is presented in Fig. 11.25. A graph of air amount changes in the working is shown in Fig. 11.26.

 https://doi.org/10.32056/KOMAG/Monograph2021.2

Ventilation parameters and climatic conditions in the Bw-1n test roadway [135]

Table 11.19.

Item	1	2	3	4	5	6	7	8	9
Month	November 2019	December 2019	January 2020	February 2020	March 2020	April 2020	May 2020	June 2020	July 2020
Fan	WLE12132 B/E	WLE12132 B/E	WLE12132 B/E	WLE12132 B/E	WLE12132 B/E	WLE12132 B/E	WLE12132 B/E	WLE 12132 B/E	WLE 12132 B/E
Fan capacity	750 m^3/min	750 m^3/min	750 m^3/min	1000 m^3/min	1000 m^3/min	1100 m^3/min	1100 m^3/min	1100 m^3/min	1150 m^3/min
Amount of air in the working face	690 m^3/min	690 m^3/min	690 m^3/min	900 m^3/min	900 m^3/min	900 m^3/min	900 m^3/min	850 m^3/min	850 m^3/min
Amount of air in the working outlet	700 m^3/min	700 m^3/min	700 m^3/min	950 m^3/min	950 m^3/min	1000 m^3/min	1000 m^3/min	1000 m^3/min	1000 m^3/min
Dry temperature in the working	25.2ºC (without cooling with air cooler)	25.6ºC (without cooling with air cooler)	23.6ºC (without cooling with air cooler)	23.4ºC (without cooling with air cooler)	24.0ºC (without cooling with air cooler)	24.0ºC (without cooling with air cooler)	25.4ºC (without cooling with air cooler)	26.0ºC (without cooling with air cooler)	27.0ºC (cooling with the cooler of the air supplied to WLE)
Humid temperature in the working	22.6ºC (without cooling with air cooler)	22.4ºC (without cooling with air cooler)	20.2ºC (without cooling with air cooler)	17.6ºC (without cooling with air cooler)	18.2ºC (without cooling with air cooler)	19.6ºC (without cooling with air cooler)	22.6ºC (without cooling with air cooler)	23.4ºC (without cooling with air cooler)	24.4ºC (cooling with the cooler of the air supplied to WLE)
Cooling intensity	15 K_w	15.2 K_w	17.3 K_w	21.2 K_w	20.5 K_w	18.9 K_w	15.5 K_w	14.7 K_w	14.3 K_w
Rock mass temperature	36.2 ºC	36.2 ºC	36.2 ºC	36.5 ºC	36.5 ºC	36.5 ºC	36.1 ºC	36.3 ºC	36.3 ºC

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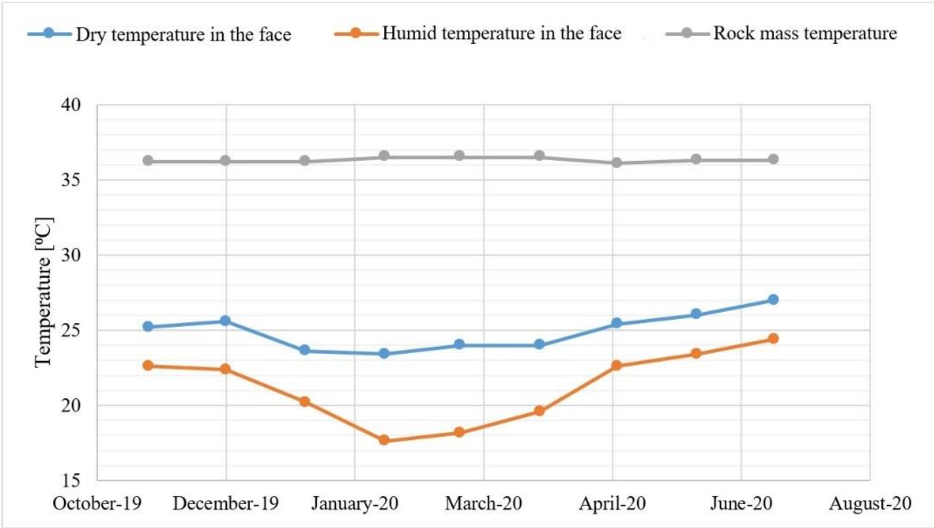

Fig. 11.25. Graph of the temperature changeability in the Bw-1n test roadway [135]

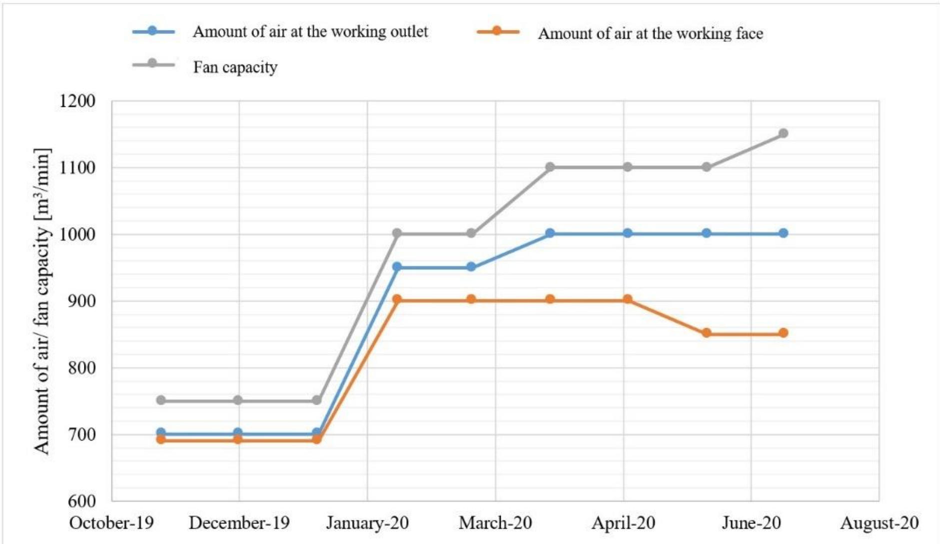

Fig. 11.26. Changeability graph of the air amount and of the fan capacity in the Bw-1n test roadway [135]

As it is presented in Fig. 11.25, starting from February (4th month of the project duration) a constant increase of the air dry and humid temperatures in the working was observed at simultaneous minor variations of the rock mass temperature. Due to that in July (10th month of the project duration) it was decided to start cooling the air supplied to the working. Besides, as it can be seen from Fig. 11.26 an increasing trend of the fan capacity and of the air amount in the working can be observed.

Device for changing the air direction

The spiral air pipe is used for controlling the air flow direction in the working and for diluting methane concentrations close to the roof. Two states of the spiral air pipe can be distinguished [135]:

– Combined ventilation is switched off – the spiral air pipe realizes a typical forced ventilation, directing the basic part of the air axially towards the roadway working front. The air flows freely through the segments of the spiral air pipe.

– Combined ventilation is switched on – the air supplied to the drive assembly gets rotated in it, and changing the flow direction it flows out through the slot in the segments of the spiral air pipe. Aerodynamically shaped walls of the slot direct the air stream tangentially to the roadway working wall towards the roof. The air stream flowing around the roof removes possible methane concentrations. The air close to the roof can reach a velocity up to 1.50 m/s. There is no gust of air onto the working front.

Cooling system of air

Due to the temperature increase of the air flowing to the fan, since July (10th month of the project realization) it has been decided to apply additional air cooling in the working under driving. The MK-300 cooling device was started. Its rated cooling power is 300 kW. It was installed in the B roadway, in the air current supplied to the fan ventilating the Bw-1n test roadway. Besides, it was decided that in a further panel of the working under driving, a cooling device would be installed in the distance of 200 m from the face front in so called side-working with an additional fan forcing the air flow through the cooler [135].

Control system of the working ventilation

The Bw-1n test roadway, during its drivage, was equipped with an automatic control system of the working ventilation and with methanometric protections [135]. A specification of used sensors is presented in Table 11.20.

Specification of sensors for controlling the working ventilation [135]

Table 11.20.

Item	Type of sensor	Threshold	Place of installation
1	CH_4 sensor	2.0%	MK
2	CH_4 sensor	1.0%	Face
3	CH_4 sensor	1.0%	Zone of air duct gearing
4	CH_4 sensor	1.0%	Outlet from the external dust control installation

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5	CH_4 sensor	1.0%	Outlet from the internal dust control installation
6	CH_4 sensor	1.0%	At cutting rocks of high and medium susceptibility to sparking
7	CH_4 sensor	0.5%	Inlet to the air pipe fan
8	CH_4 sensor	1.5%	Over each transformer station
9	CH_4 sensor	2.0%	Outlet from the working
10	CO sensor	-	Outlet from the working
11	Sensor of WLE operation	-	-

 https://doi.org/10.32056/KOMAG/Monograph2021.2

12. Control and monitoring of independent roof bolting support

Wojciech Masny[1], Dariusz Wejman[2]

12.1. Control of bolts, accessories and of bolting equipment as well as a correctness control of bolts installation

- a visual control of the working at reaching the face,
- a current control of the roof separations,
- a supervision of observing the working determined dimensions during the cutting process,
- ripping of rocks after cutting the web,
- spacing of bolt holes according to the bolting scheme,
- ensuring correct lengths, diameters, inclinations and cleanness of bolt holes,
- a correct installation of bolts in holes,
- a control of expiration dates and of binding time of glue charges,
- a control of types, diameters and lengths of used bolts,
- an elimination of damaged bolts,
- a control of correct filling of bolt holes with glue charges,
- a correct installation of the roof and walls lining.

The scope of a correctness control of the bolts installation includes:

- a correctness of the bolts insertion,
- an adhesion of washers to the roof,
- spacing of bolts according to the bolting scheme,
- a load-bearing capacity of bolts.

The measurement of the bolts load-bearing capacity is conducted with use of a hydraulic puller, and testing consists in applying the pulling force to a bolt at a simultaneous measurement of the load and deformation [136].

12.2. Control principles of the working stability

A stability control of the working, driven in the independent roof bolting support, should be conducted up to date along the working by the supervisory

[1] Central Mining Institute
[2] Enterprise of Shafts Construction JSC

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personnel and periodically in selected places, according to the decisions of the Mining Department Manager [139].

The stability control of the working includes a current control and a periodical control.

A current control, according to the guidelines of the Central Mining Institute, encompasses an observation of two-level indicators of low and high separation on which safety thresholds and hazards are marked [136]:

- indicators of low separation: installed at the depth exceeding 0.3 m above the bolting range of roof rocks, close to the working axis, at intervals not bigger than 30 m, the permissible value of the low separation is 50 mm, an emergency state is signalled in red and it shows that the separation of 2% bolting range is exceeded,
- indicators of high separation: installed at the depth exceeding 0.3 m above a double bolting range of roof rocks, but not smaller than 4.5 m, near the workings axis, at intervals not bigger than 30 m, the permissible value of the high separation is 81 mm, an emergency state is signalled in red and it shows that the separation of 1.5% working width is exceeded.

Additionally, after having reached 50% of maximum separation values and at a significant change of their increase dynamics, an expert should be notified. The described critical values concern a situation, when the working is surrounded, at both sides, with the rock mass and it is not subject to other impacts. The first stand of current control should be built in the distance not bigger than 5 m from the place of starting the working drivage in independent roof bolting support and the following indicators should be installed at the distances respective to the mining situation [135].

In the framework of the periodical control, periodical control stations were installed [135]:

- I station of periodical control was installed on 19th November 2019 at the mark 14 rm. Its equipment consists of:
  - ➢ 6 bolts with instrumentation in one row of a location, spacing and length in accordance with the basic support,
  - ➢ 3 separation meters, 3-level, installed at the depths: 2.1; 5.0; 6.0 m,
  - ➢ a hand displacement meter probe in the working axis.

A scheme of the measuring station is presented in Fig. 12.1 and 12.2.

 https://doi.org/10.32056/KOMAG/Monograph2021.2

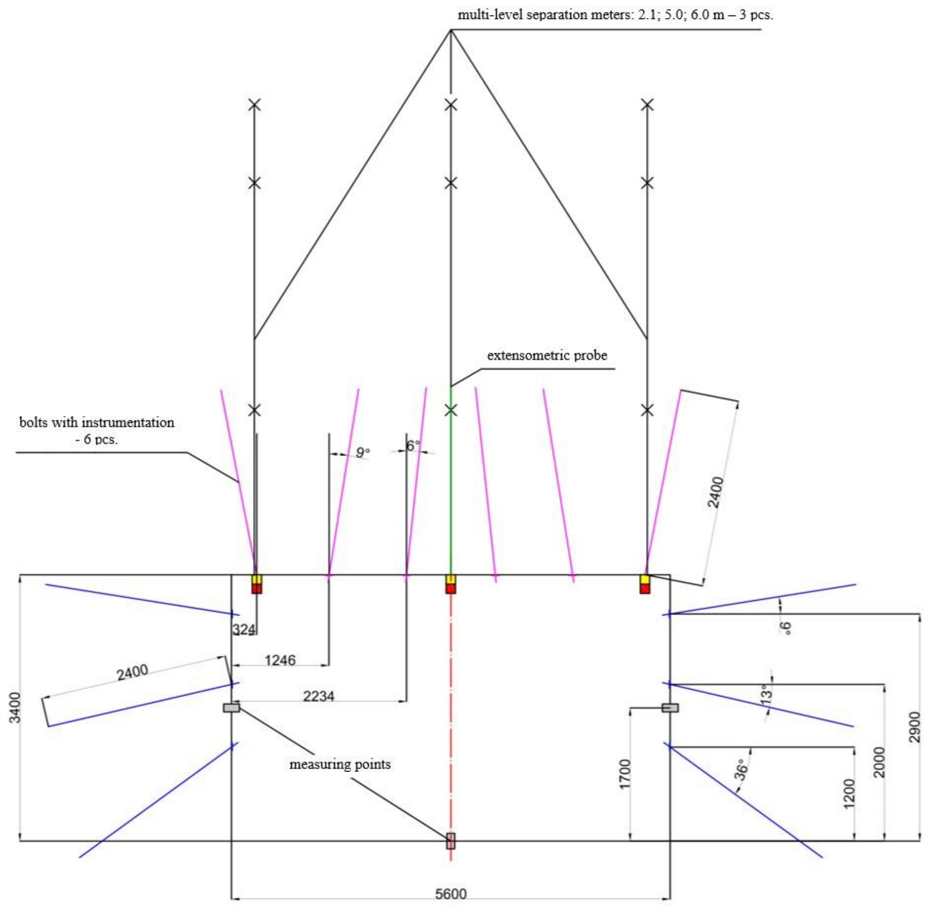

Fig. 12.1. Scheme of a periodical control station – station No. 1 – vertical section

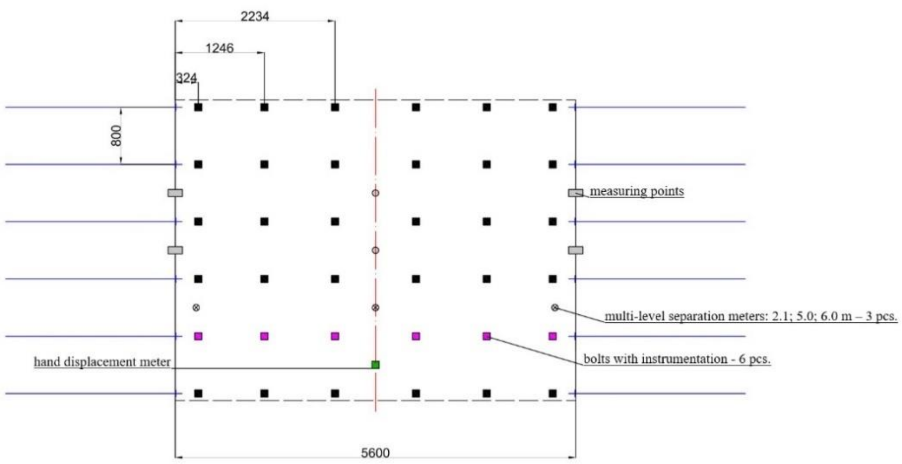

Fig. 12.2. Scheme of a periodical control station – station No. 1 – top view

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– II station of periodical control was installed on 6th December 2019 at the mark 58.6 rm and its instrumentation consists of [135]:
  - ➢ 6 bolts with instrumentation in a row of a location, spacing and length in accordance with the basic support,
  - ➢ 3 separation meters, 3-level, installed at the depths: 2.1; 5.0; 6.0 m,
  - ➢ 2 separation meters mutually supplementary, whose principle of operation is similar to a hand displacement meter, installed at the depths: the first one – 1.0; 3.0; 5.0 m and the other one 2.0; 4.0; 6.0 m called mechanical hand displacement meters.

A scheme of the measuring station is shown in Fig. 12.3 and 12.4. In Fig. 12.5 a measurement scope of the separation meters, described as mechanical hand displacement meters, is presented.

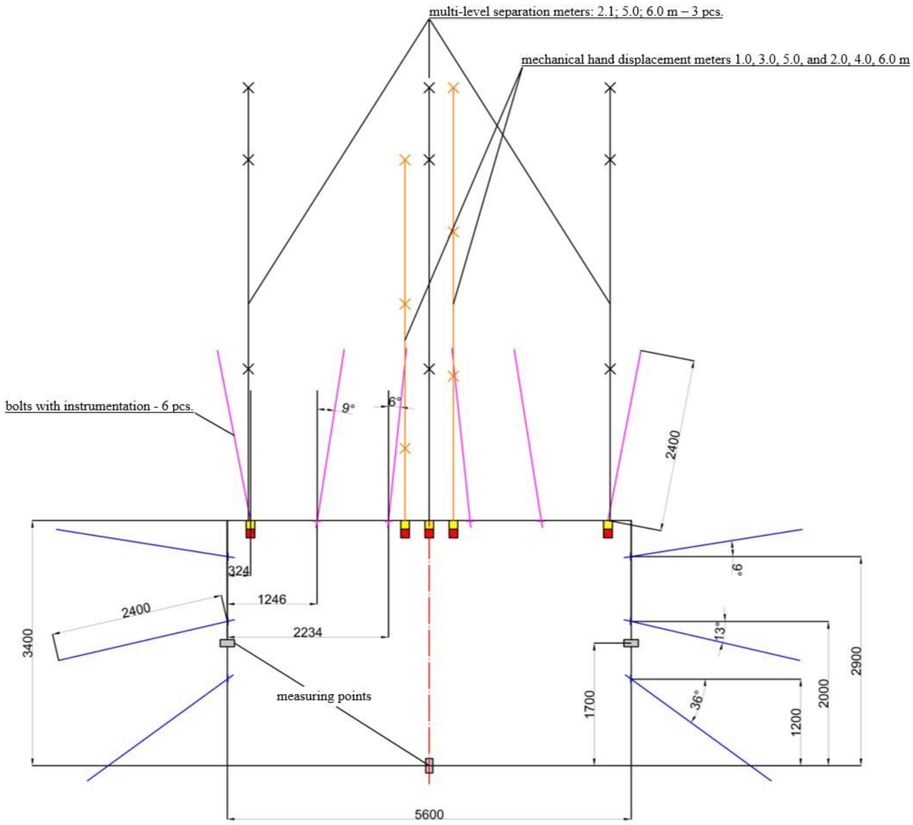

Fig. 12.3. Scheme of periodical control station – station No. 2- vertical section

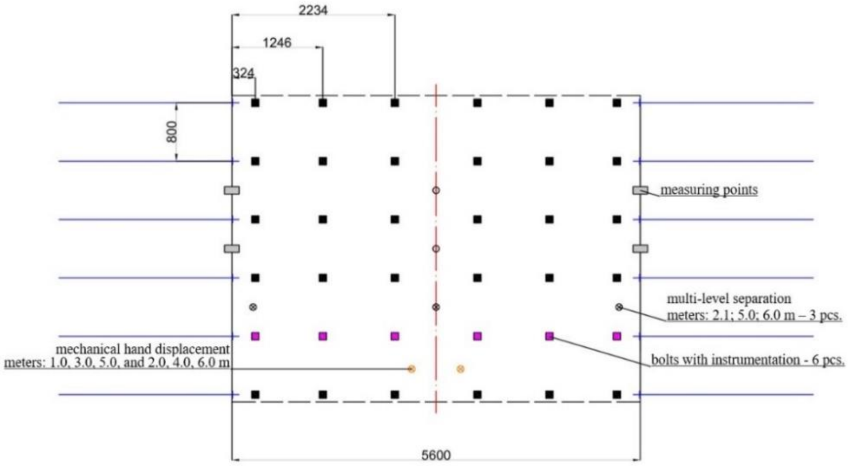

Fig. 12.4. Scheme of periodical control station – station No. 2- top view

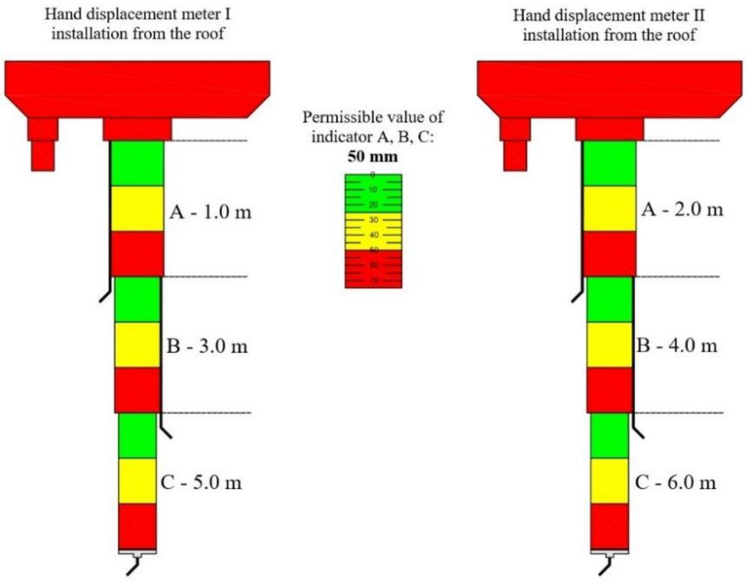

Fig. 12.5. Scheme of the mechanical hand displacement meter

- III additional periodical control station was installed on 27th December 2019 on the mark 161 rm to check measuring bolts, installed directly after an exposition of the roof, loading them with axial forces. Its equipment consists of [135]:
  - 6 bolts with instrumentation in a row at a location, spacing and length in accordance with the basic support,

- 2 separation meters, 3-level, installed at the depths: 2.1; 5.0; 6.0 m,
- 2 separation meters mutually supplementary, whose principle of operation is similar to a hand displacement meter, installed at the depths: the first one – 1.0; 2.0; 2.5 m and other one – 3.0; 5.0; 6.0 m called a mechanical hand displacement meter.

A scheme of the measuring station is presented in Fig. 12.6 and 12.7.

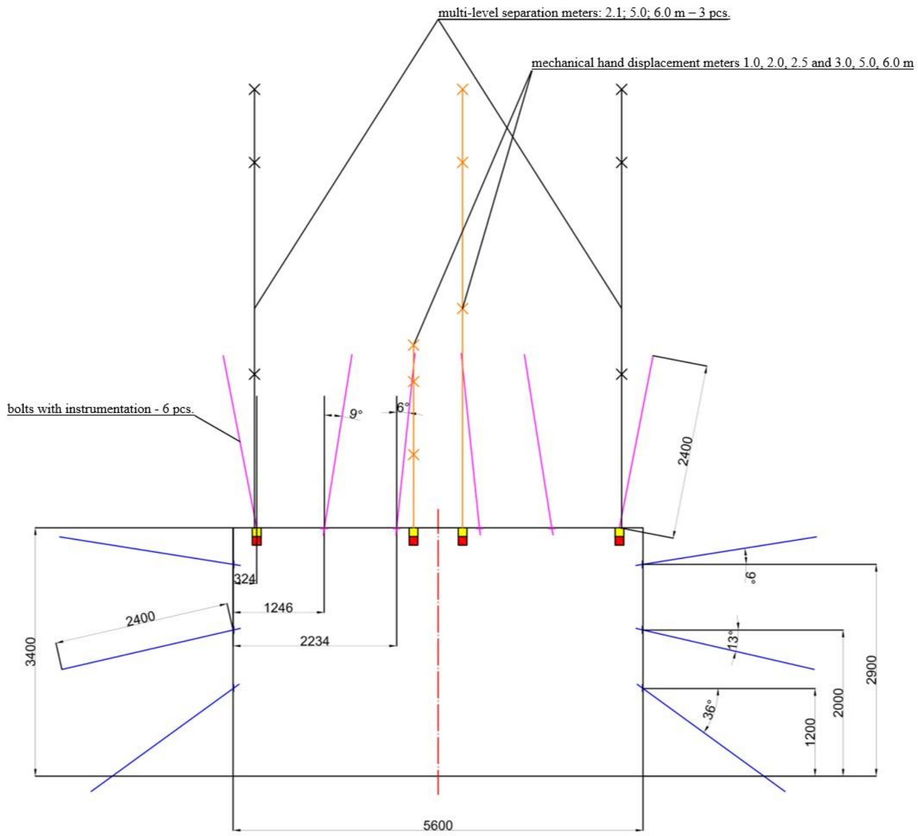

Fig. 12.6. Scheme of a periodical control station – station No. 3 – vertical section

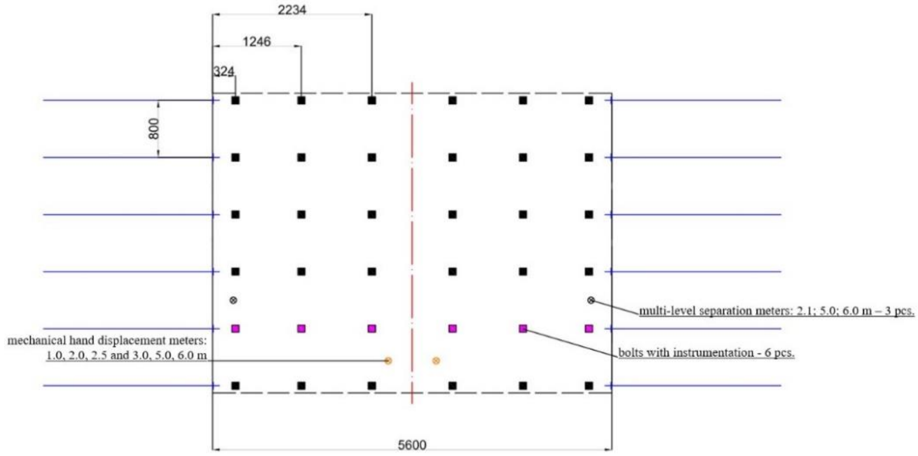

Fig. 12.7. Scheme of a periodical control station – station No. 3 - top view

– IV station of periodical control station was installed on 16th April 2020 at the mark 564.6 rm and its equipment consists of [135]:

- ➢ 6 bolts with instrumentation in a row at a location, spacing and length in accordance with the basic support,
- ➢ 4 bolts with instrumentation in a row at a location, spacing and length in accordance with the changed bolting network,
- ➢ 3 bolts with instrumentation in the left and right walls at a location, spacing and length in accordance with the basic support,
- ➢ 2 separation meters 3 - level, installed at the depths: 2.1; 5.0; 6.0 m,
- ➢ 2 separation meters mutually supplementary, whose principle of operation is similar to a hand displacement meter, installed at the depths: the first one – 1.0; 3.0; 5.0 m and other one – 2.0; 4.0; 6.0 m, called mechanical hand displacement meters.

A scheme of the measuring station is shown in Fig. 12.8 and 12.9.

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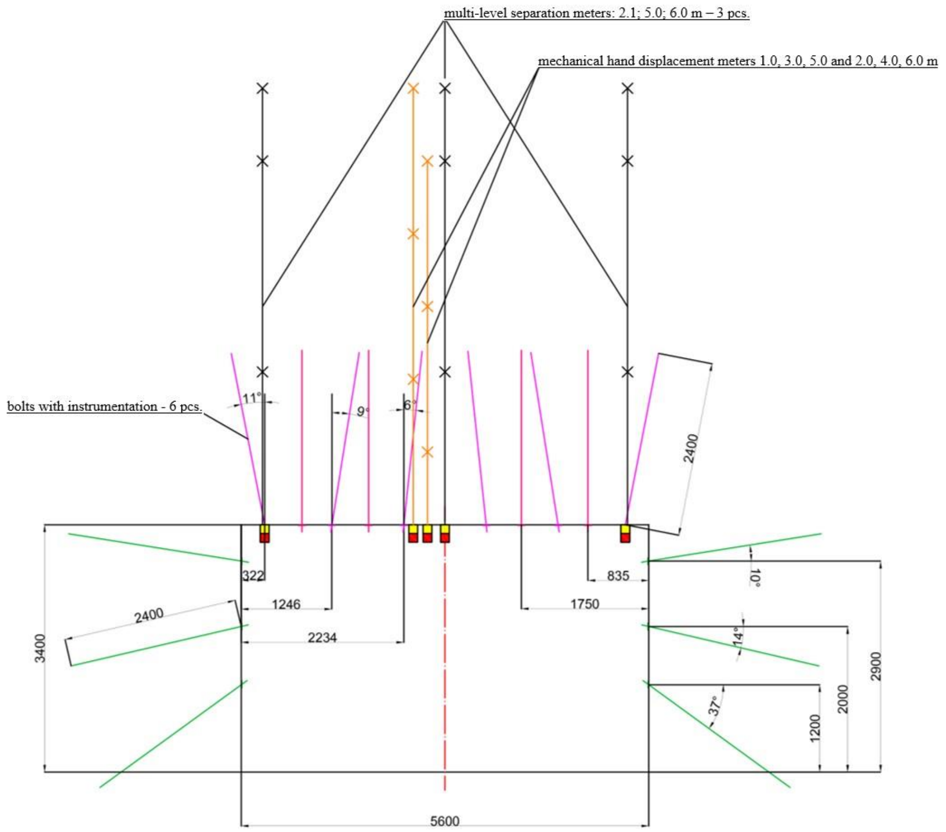

Fig. 12.8. Scheme of a periodical control station – station No. 4 – vertical section

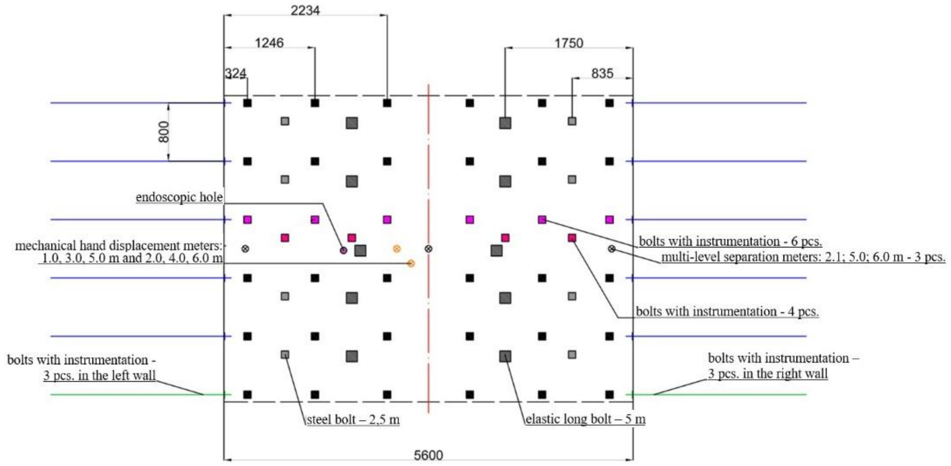

Fig. 12.9. Scheme of periodical control station – station No. 4 – top view

- V station of periodical control was installed on 1st July 2020 at the mark 967 rm. Its equipment includes [135]:

- 6 bolts with instrumentation in a row of a location, spacing and length in accordance with the basic support,
- 4 bolts with instrumentation in a row of a location, spacing and length in accordance with the changed bolting network,
- 2 separation meters 3 - level, installed at the depths: 2.1; 5.0; 6.0 m,
- 2 separation meters mutually supplementary, whose principle of operation is similar to a hand displacement meter, installed at the depths: the first one – 1.0; 3.0; 5.0 m and the other one – 2.0; 4.0; 6.0 m, called mechanical hand displacement meters.

A scheme of the measuring station is shown in Fig. 12.10 and 12.11.

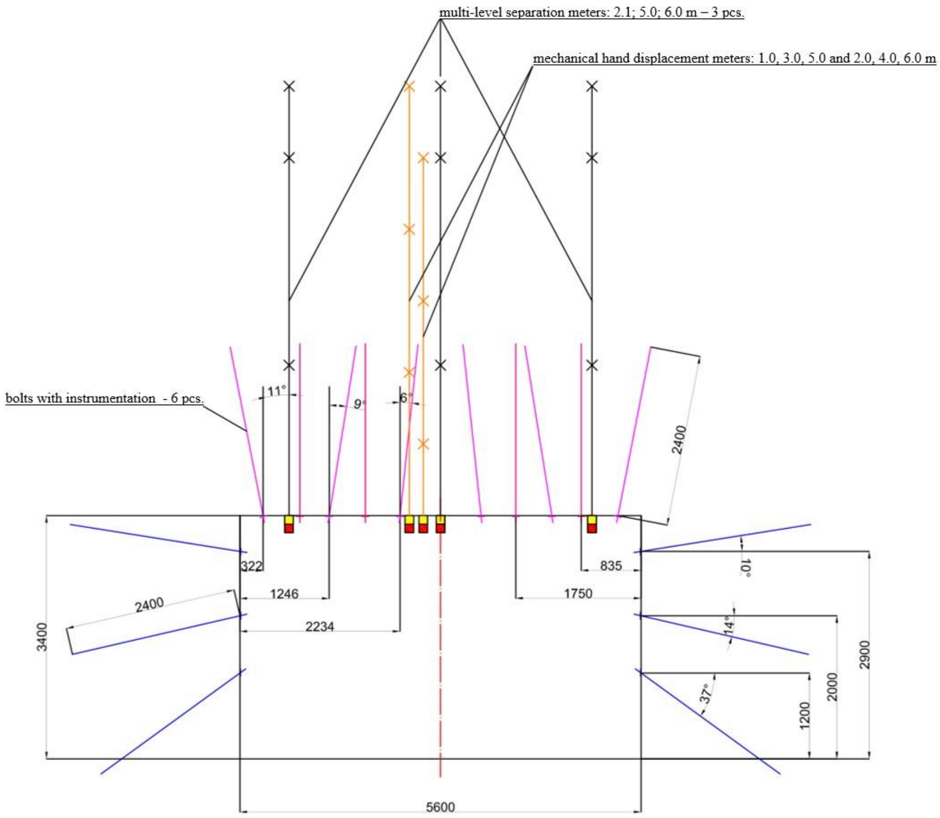

Fig. 12.10. Scheme of the periodical control station – station No. 5 – vertical section

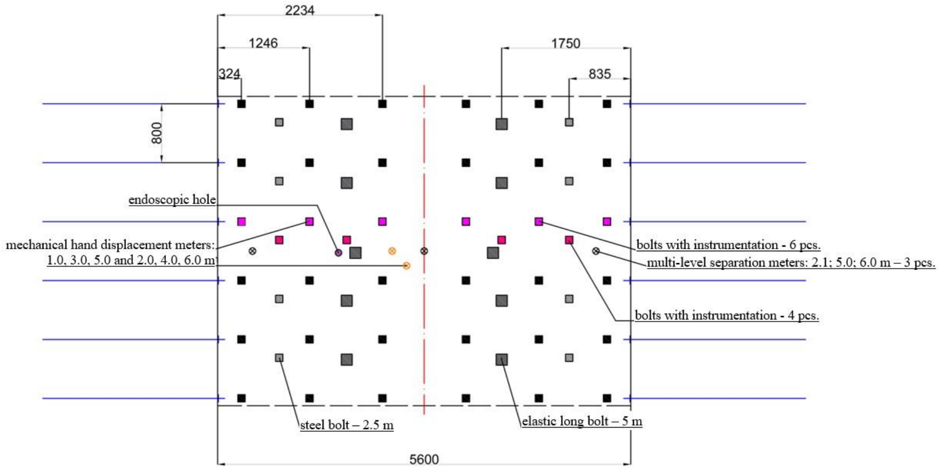

Fig. 12.11. Scheme of the periodical control station – station No. 5 - top view

A convergence of the workings is measured at the stations of the convergence measurement. There are two measuring points at each station. They enable to assess the working convergence in the horizontal and vertical directions. The stations of convergence measurements were installed as follows [135]:

– the first convergence measuring station at 14.80 and 15.55 m, the station scheme is presented in Fig. 12.12,
– the second convergence measuring station at 58.20 and 59.00 m, the station scheme is presented in Fig. 12.13,
– the third convergence measuring station at 413.10 and 414.00 m, the station scheme is presented in Fig. 12.14,
– the fourth convergence measuring station at 503.10 and 503.80 m, the station scheme is presented in Fig. 12.15,
– the fifth convergence measuring station at 563.1 and 563.8 m, the station scheme is presented in Fig. 12.16.

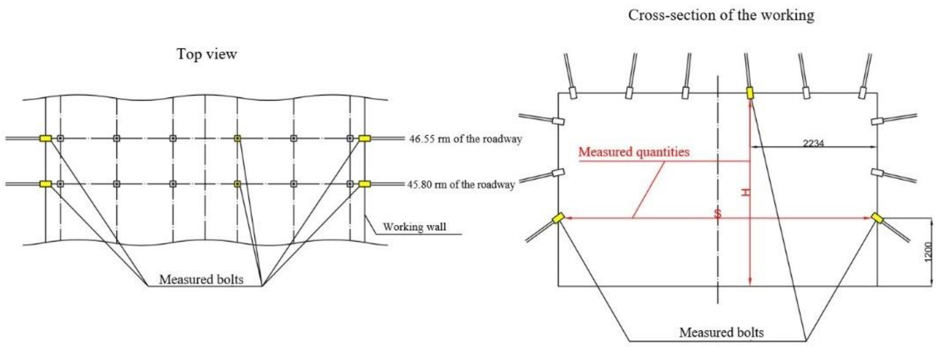

Fig. 12.12. Scheme of the working convergence measurement – station No. 1

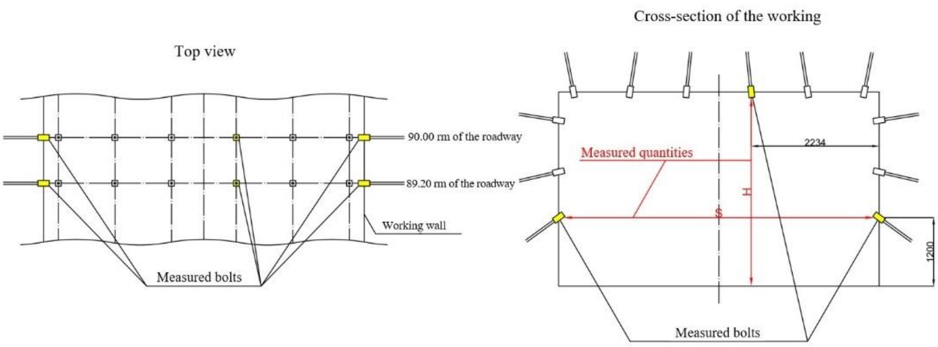

Fig. 12.13. Scheme of the working convergence measurement – station No. 2

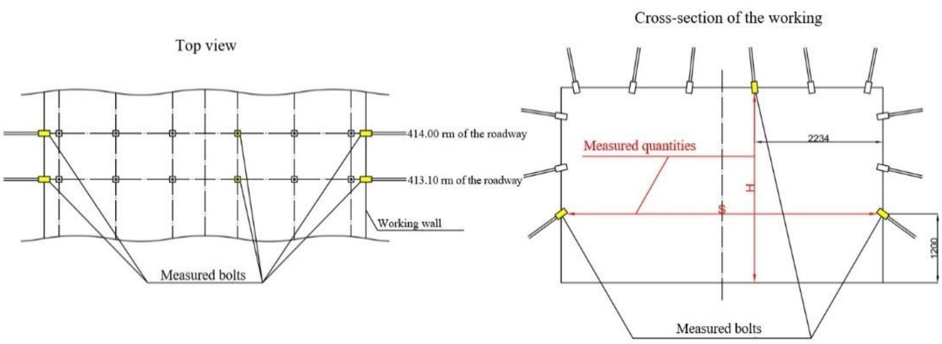

Fig. 12.14. Scheme of the working convergence measurement – station No. 3

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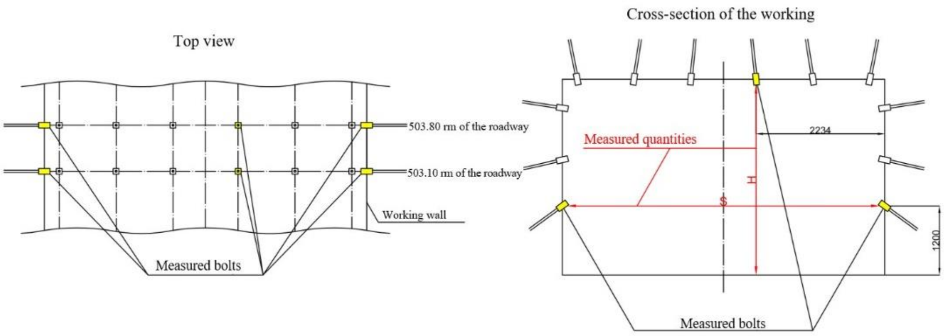

Fig. 12.15. Scheme of the working convergence measurement – station No. 4

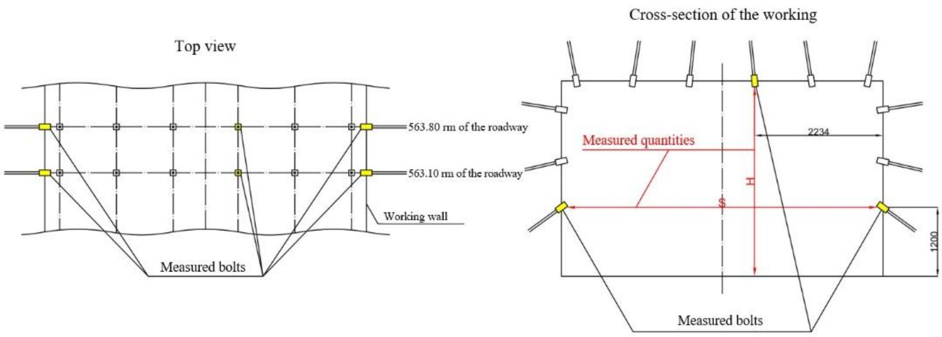

Fig. 12.16. Scheme of the working convergence measurement – station No. 5

A recommended frequency of conducting the current control of low and high separation for Phase I of the project [138] is as follows:

– a visual control together with a record – an initial phase of drivage, i.e. 3 first separation meters, counting from the face – once at each working shift; once every 24 hours on days off,
– a visual control together with a record – all the other separation meters – once a week,
– a visual control – all the other separation meters – once every working shift; once every 24 hours during the days off,
– a visual control with a record – after having finished the working drivage – once a week.

A recommended frequency of conducting periodical control at the periodical control stations and of measuring the convergence in Phase I of the project [138] is as follows:

– during 2 weeks since the stand installation:

  - a visual control together with a record at the periodical control station – once a day on working days,
  - a visual control with a record at convergence measurement stations – once a week,
- after 2 weeks since the stand installation:
  - a visual control with a record at the periodical control stations and at the convergence measurement stations – once a month.

A frequency of measurements may be subject to a change in relation to the working behaviour.

The recommended frequency of current control of low and high separation indicators for Phase II of the project (bend No. 1) [129]:

- a visual control with a record – during work in the crossing zone – once during each working shift, once every 24 hours on days off,
- a visual control with a record – after having finished the work in the crossing zone – once a week,
- a visual control with a record – after having finished the working drivage – once a month,
- a visual control – after having finished the working drivage – once a week.

A frequency of measurements may be subject to a change in relation to the working behaviour.

The recommended frequency of current control of low and high separation indicators for Phase III of the project [139]:

- a visual control with a record – an initial phase of drivage, i.e. 3 first separation meters counting from the face – once at each working shift, once every 24 hours on days off,
- a visual control with a record – all the other separation meters – once a week,
- a visual control – all the other separation meters – once at each working shift, once every 24 hours on days off,
- a visual control with a record – after having finished the working drivage – once a month,
- a visual control – after having finished the working drivage – once a week.

The recommended frequency of a periodical control at the periodical control and convergence measurement stations for Phase III of the project [139]:

- during 2 weeks since the stand installation:

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- ➢ a visual control with a record at the periodical control station – once a day on working days,
- ➢ a visual control with a record at the convergence measurement station – once a week,

– after 2 weeks since the stand installation:

- ➢ a visual control with a record at the periodical control station – once a month,
- ➢ a visual control with a record at the convergence measurement station – once every 3 weeks.

A frequency of measurements may be subject to a change in relation to the working behaviour.

The recommended frequency of conducting the current control of low and high separation for Phase IV of the project (bend No. 2) [134]:

- a visual control with a record – during the work in the crossing zone – once at each working shift, once every 2 days on days off,
- a visual control with a record – after having finished the work in the crossing zone – once a week,
- a visual control with a record – after having finished the working drivage – once a month,
- a visual control – after having finished the working drivage – once a week.

A frequency of measurements may be subject to a change in relation to the working behaviour.

12.3. Control results of the working stability

Control of the roof separations – separation meters

In Fig. 12.17-17.21 the sizes of roof separations, measured at the individual stations of periodical control with the measurement date, the distance of the station from the face front and the separation level are presented. The biggest increases of separations are observed shortly after an installation of separation meters, which is connected with a small distance of the measuring point from the face front. As the face front distance from the measuring point increases, a stabilization of separations can be observed. In particular this phenomenon can be noticed on the graphs for the stations 1-4. It confirms a correctness of the support operation, which binds the rock strata and prevents a propagation of incontinuities in the rock mass.

At the periodical control station No. 1, after increasing the face front distance to about 25 m from the installation place of separation meters, an increase of separations at the left and right walls happened. Nevertheless, the sizes of the separations did not exceed critical values, so it was not necessary to undertake any additional activities. The values of the roof separations, measured in the working axis, were unchanged during the whole period of measurements.

At the periodical control station No. 2 the maximum observed value of the roof separations was only 5 mm, which confirmed an operational efficiency of independent roof bolting support and a correctly chosen bolting scheme.

The measurements of the roof separations at the periodical control station No. 3 were conducted at the right and left walls exclusively. The maximum observed separation was 5 mm, which confirmed an operational efficiency of independent roof bolting support and a correctly chosen bolting scheme.

At the periodical control station No. 4 a constant increase of roof separations at the left wall was observed initially. The separations stabilized finally at the value of 23 mm, when the face front reached the distance of 400 m from the installation place of separation meters. In connection with a significant outnumber of separations, measured at the left wall in relation to the separations measured in the working axis and at the right wall, it was decided to change the bolting scheme and to install an additional bolt at the working left wall.

At the periodical control station No. 5 the biggest measured values of the roof separations occurred at the left wall, reaching 15 mm. Besides, a constant increase of separations in the working axis was observed. It was connected with a small distance of the measuring point from the face front and with disadvantageous geological conditions.

In Fig. 12.22 values of separations in the measuring points of current control are presented. During the operations in the 6th month of the project realization (March 2020), the permissible values of separations were exceeded, thus indicating a generation of fissures in the roof. Due to that on 25th March 2020 an inspecting visit was conducted in the working, based on which a change of the bolting scheme in Phase III of the project was decided. Based on roof tests, conducted with use of the introscopic camera, it was shown that an increase of separations in the roof was caused by an occurrence of coal laminae in the roof. After having undertaken protective measures, a stabilization of separations was observed. During the following stages of operations, no alarming movements of the rock strata, resulting in separations, occurred. At present the size of fissures in the roof stays at a safe level.

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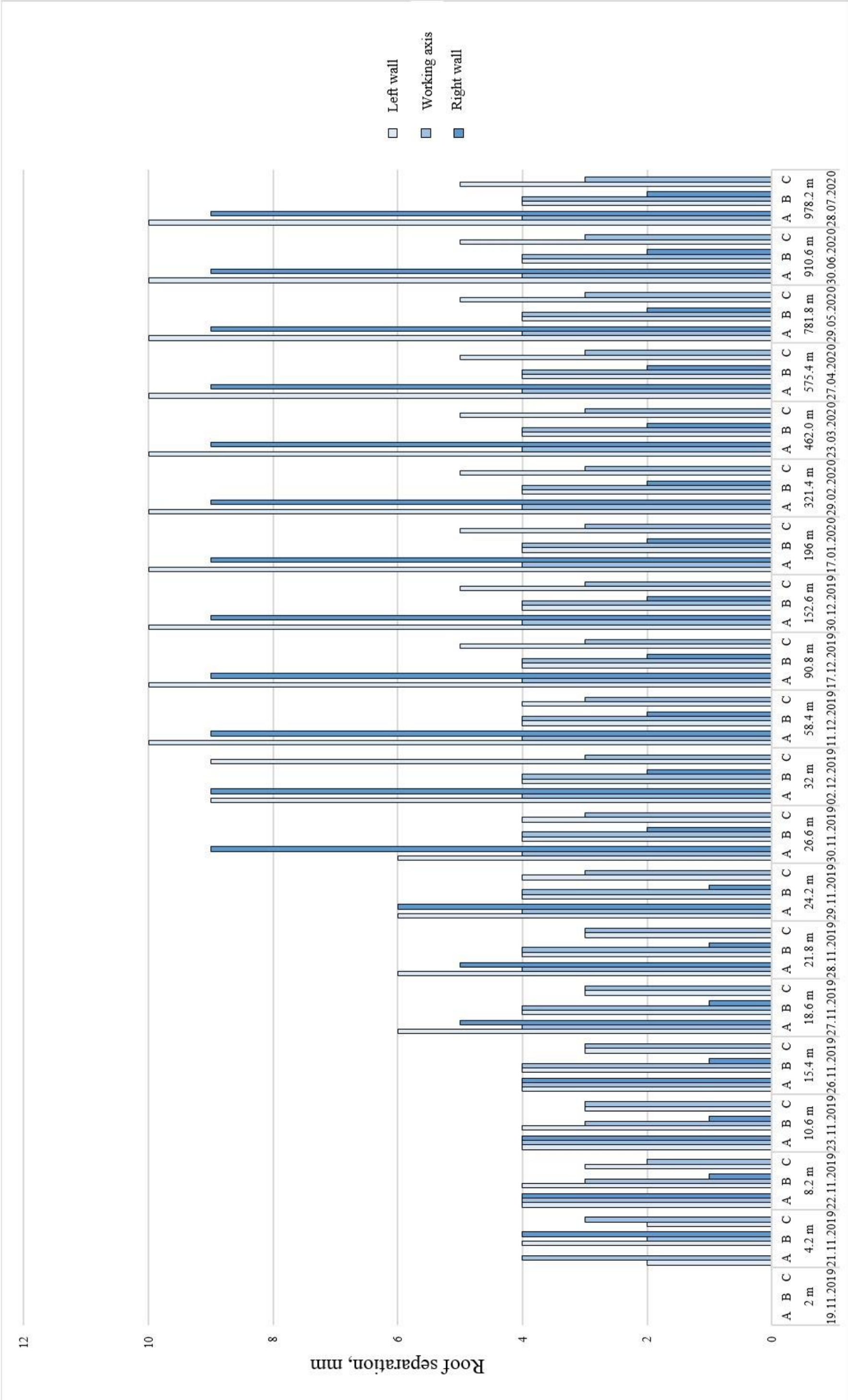

Fig. 12.17. Sizes of separations in the working roof at the periodical control station No. 1 [135]

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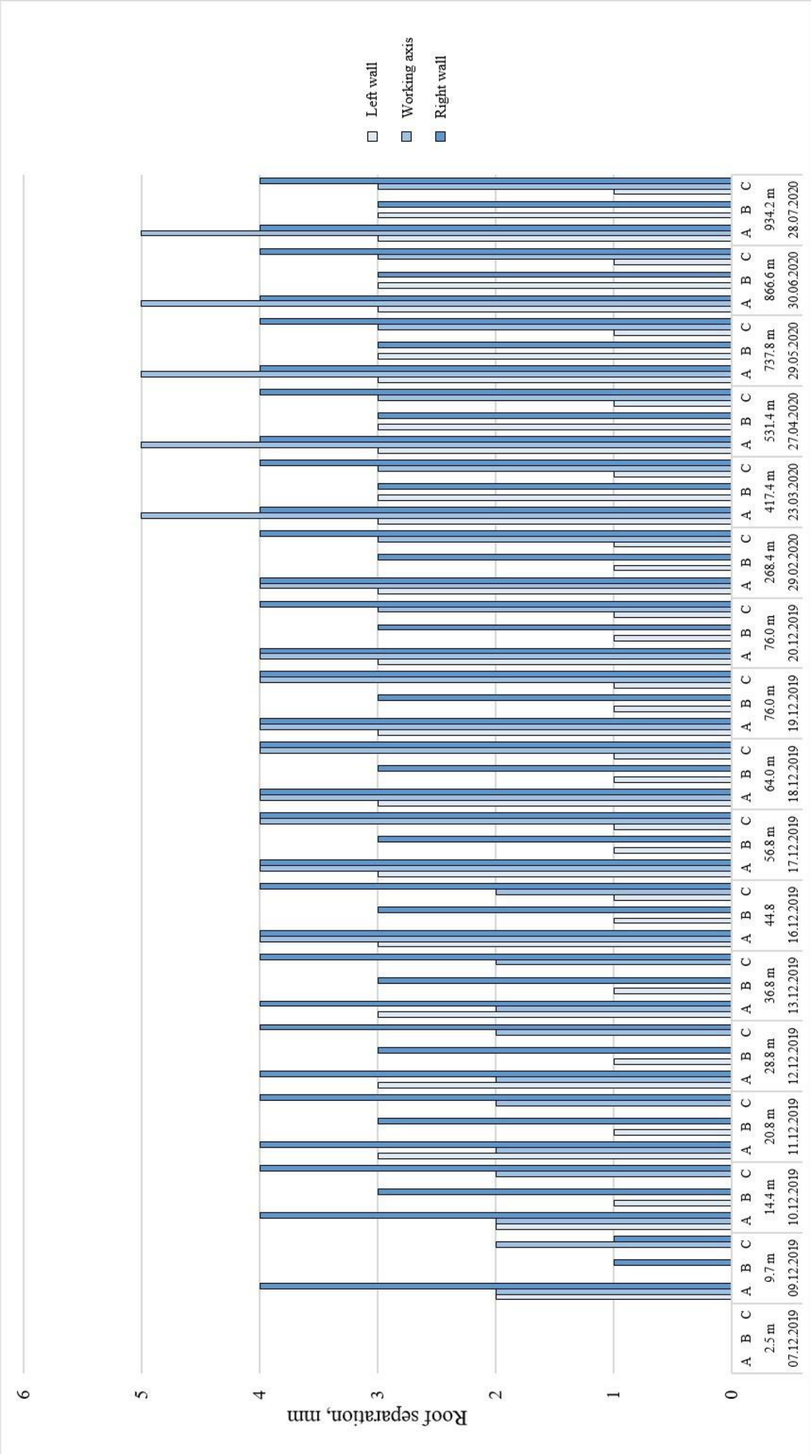

Fig. 12.18. Sizes of separations in the working roof at the periodical control station No. 2 [135]

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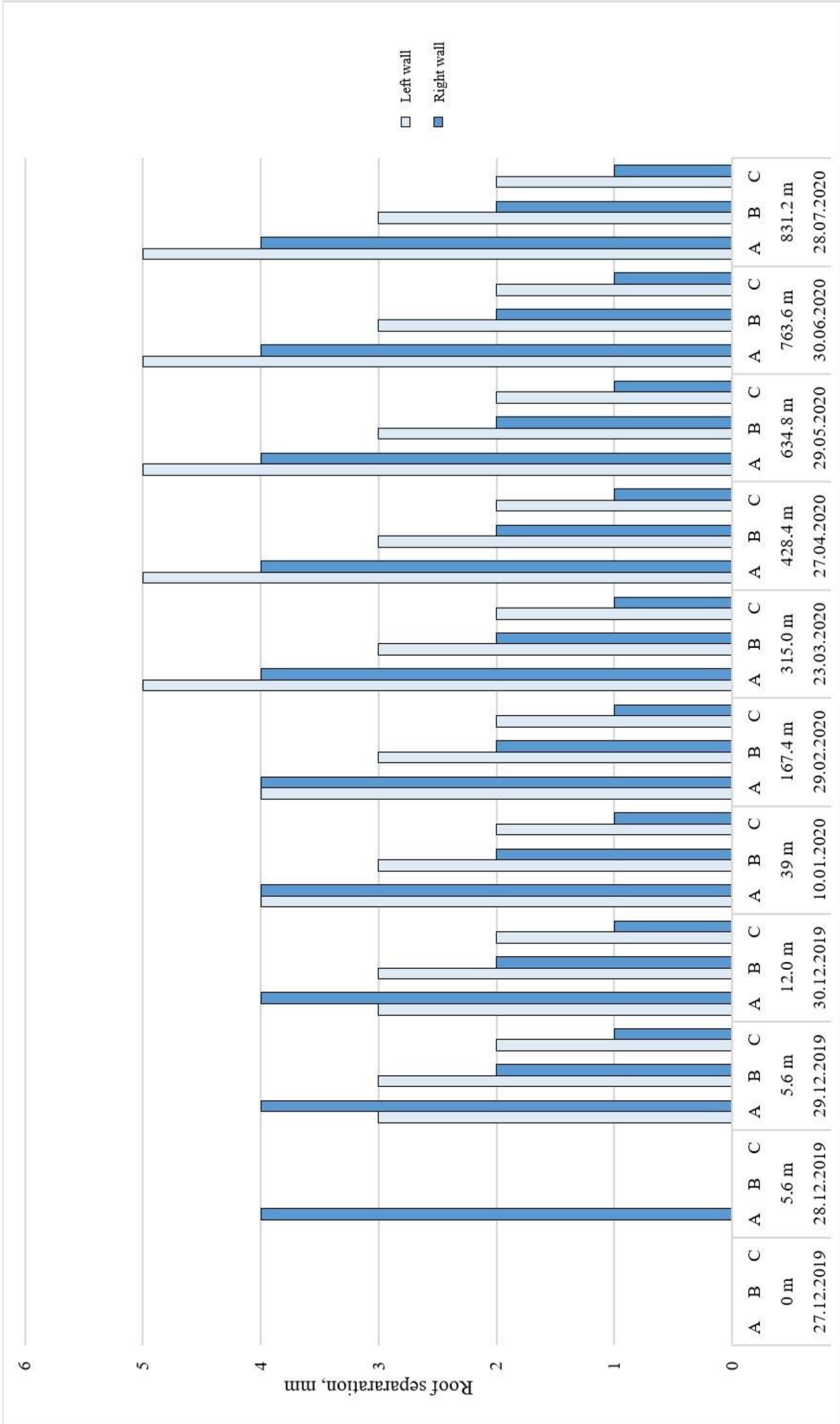

Fig. 12.19. Sizes of separations in the working roof at the periodical control station No. 3 [135]

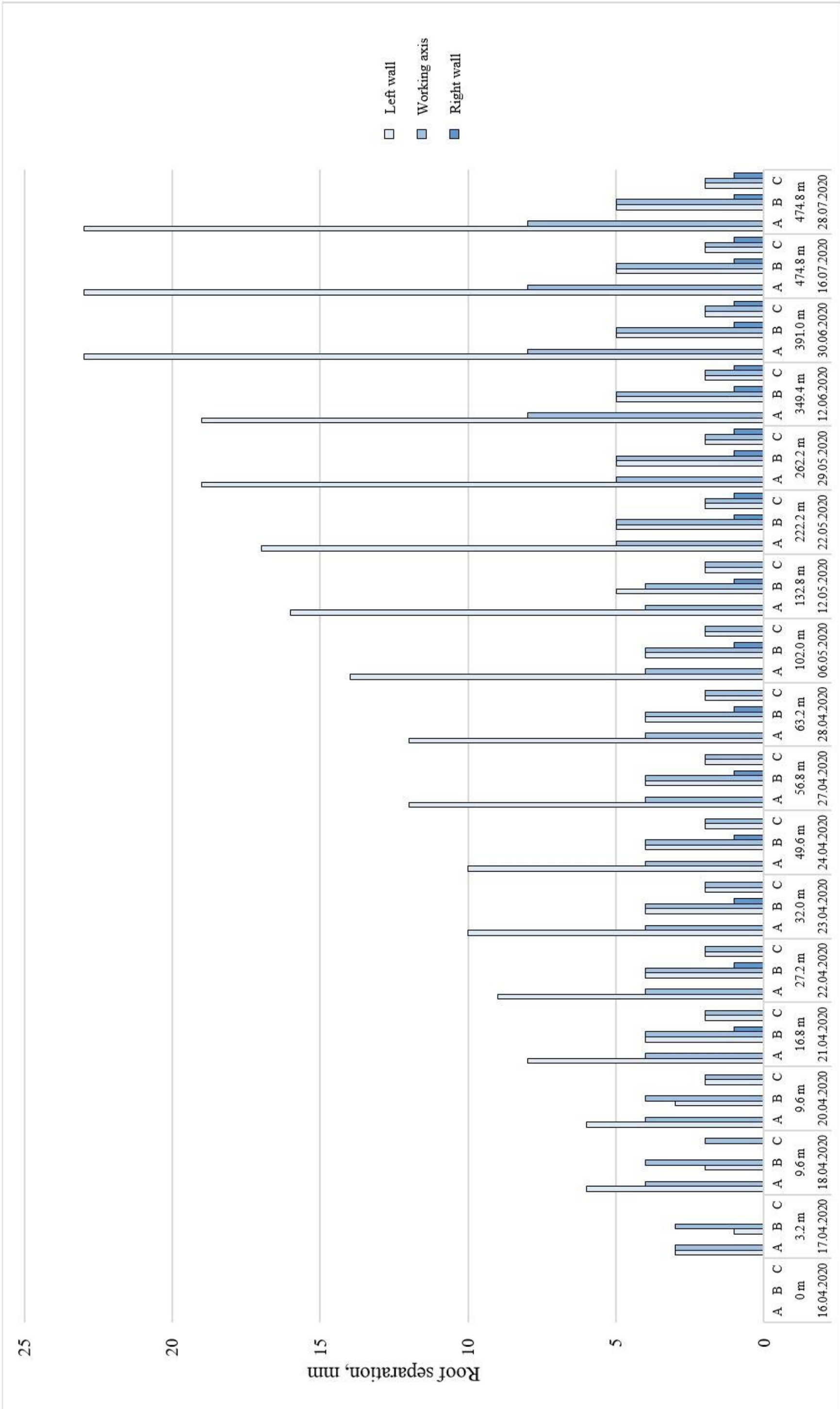

Fig. 12.20. Sizes of separations in the working roof at the periodical control station No. 4 [135]

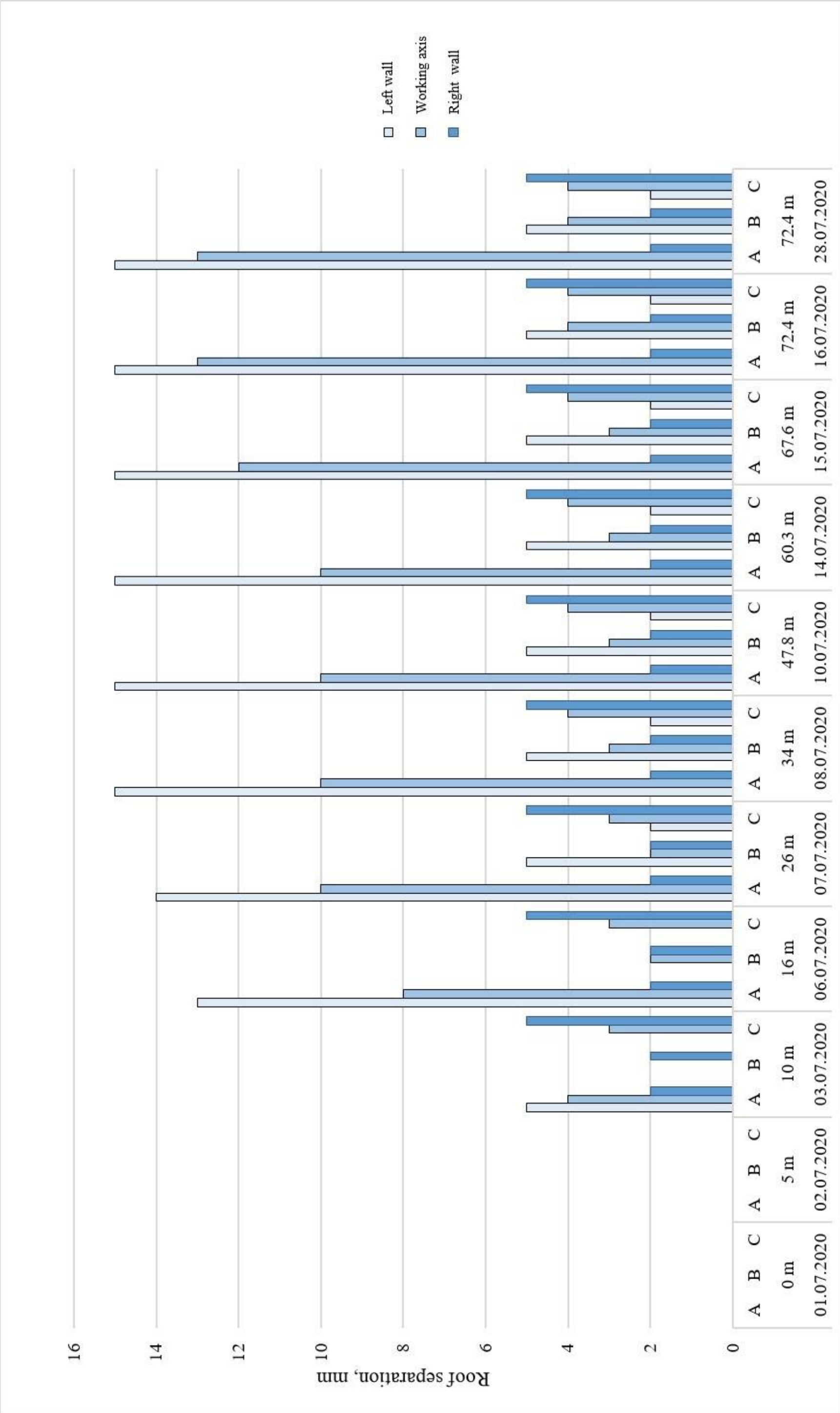

Fig. 12.21. Sizes of separations in the working roof at the periodical control station No. 5 [135]

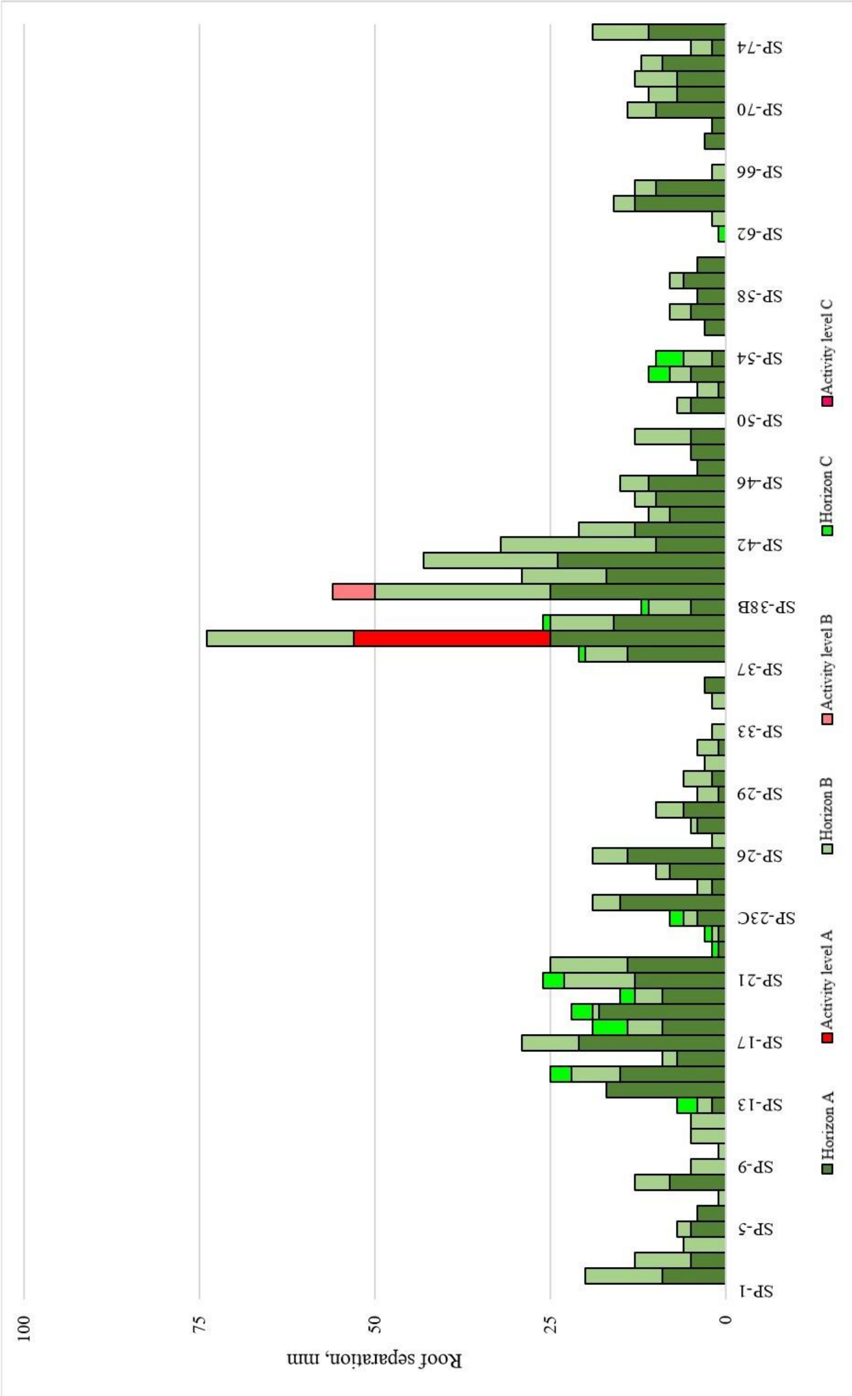

Fig. 12.22. Sizes of separations at the measuring points of current control [135]

Control of roof stratification – hand displacement meters

The range of separations of roof rocks, monitored with a hand displacement meter probe at the first station and with use of a mechanical displacement meter at the other stations, stays on an unchanged level. The measurements from the hand displacement meter probe (Fig. 12.23 and 12.24) indicate minor changes both in the case of separations sizes as well as displacements. A sensitivity and accuracy of a measurement with use of the probe is high enough, so these sizes can be left out of account and regarded as insignificant. Also in the case of mechanical displacement meters (Fig. 12.25-12.28), a lack of separations increase has been observed for a longer time period. It concerns the station No. 5, situated closest to the face front as well [135].

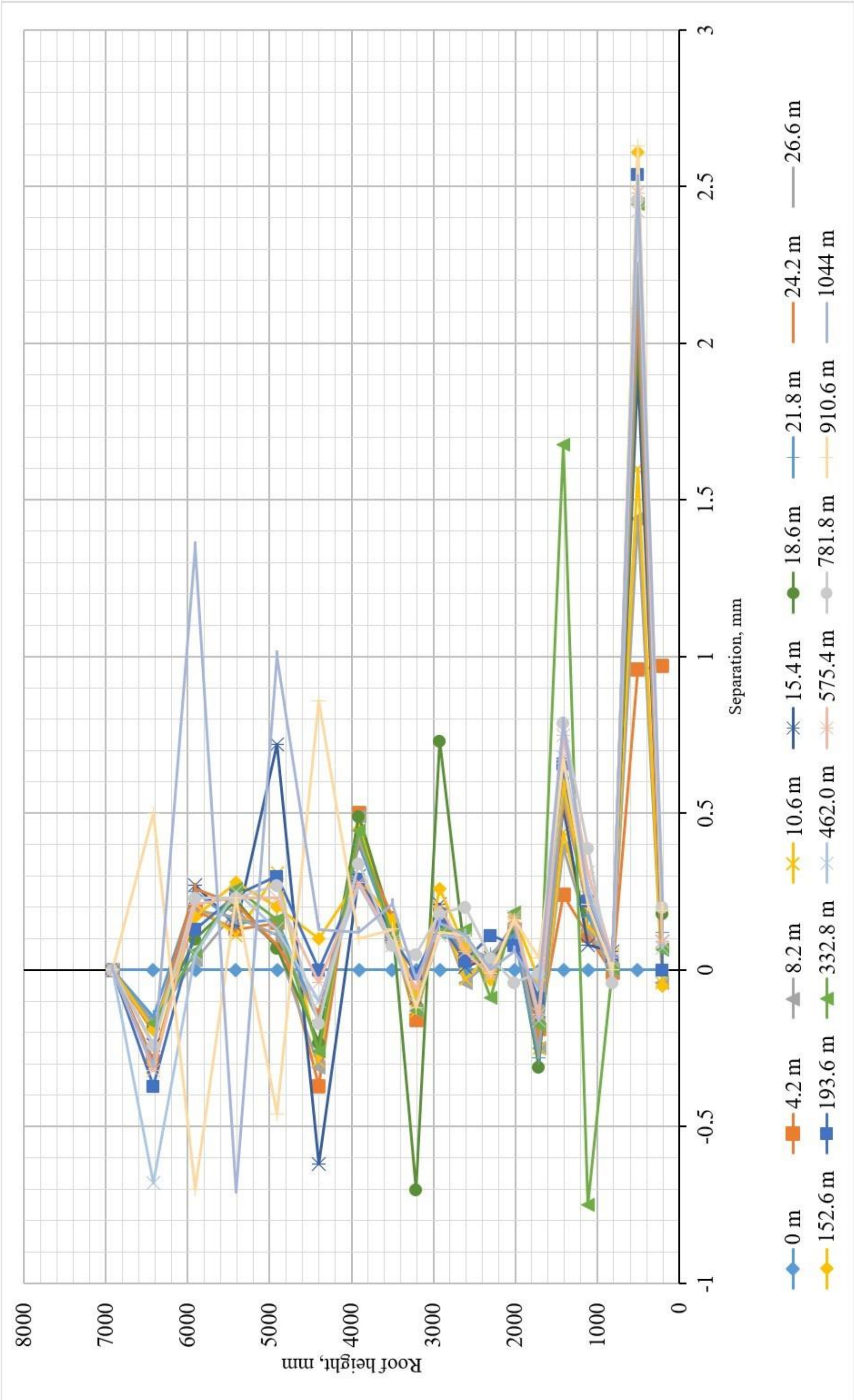

Fig. 12.23. Separations of roof rocks based on measurements with use of hand displacement meter probe – station No. 1 [135]

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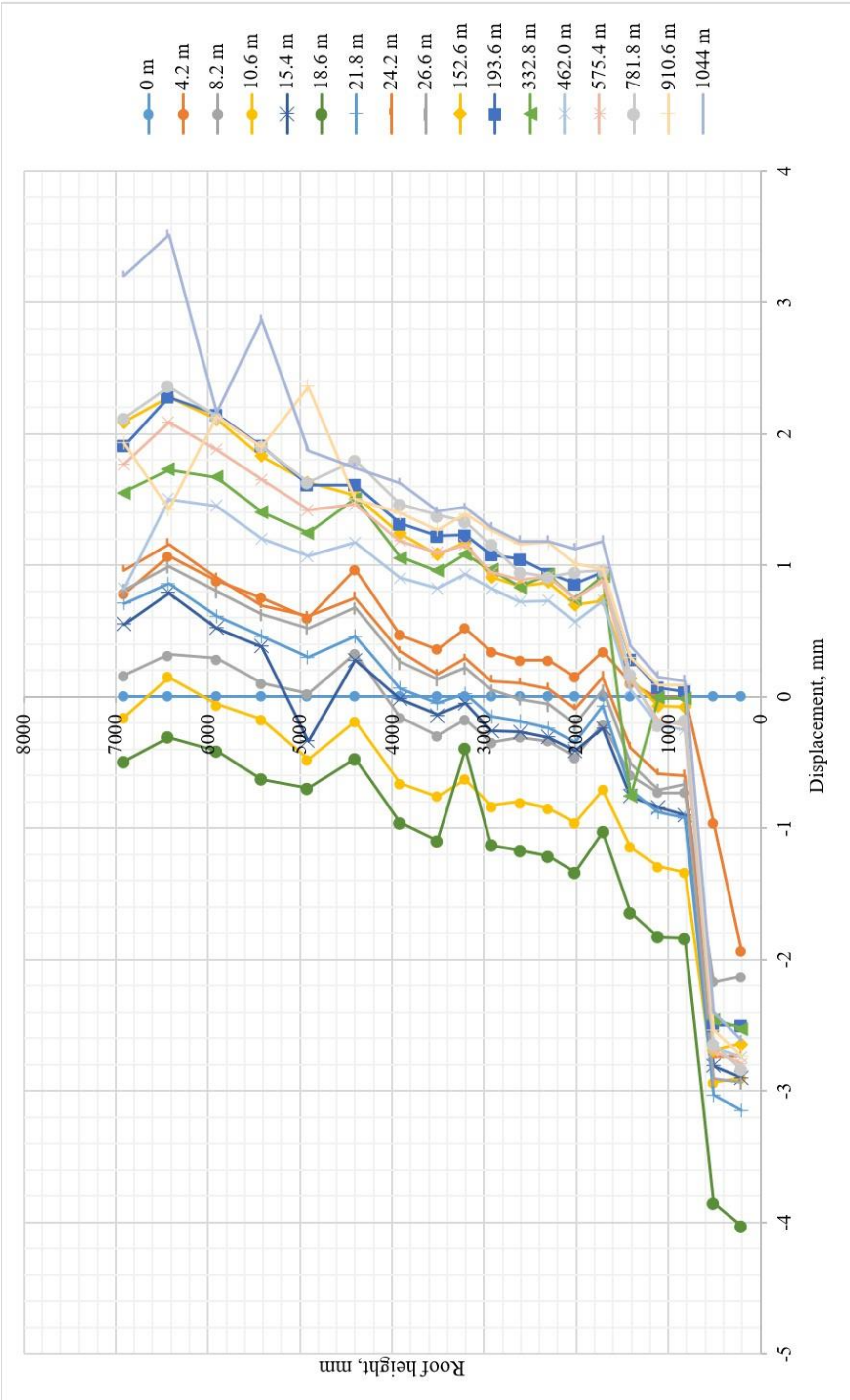

Fig. 12.24. Displacement of roof strata based on measurements with use of hand displacement meter probe – station No. 1 [135]

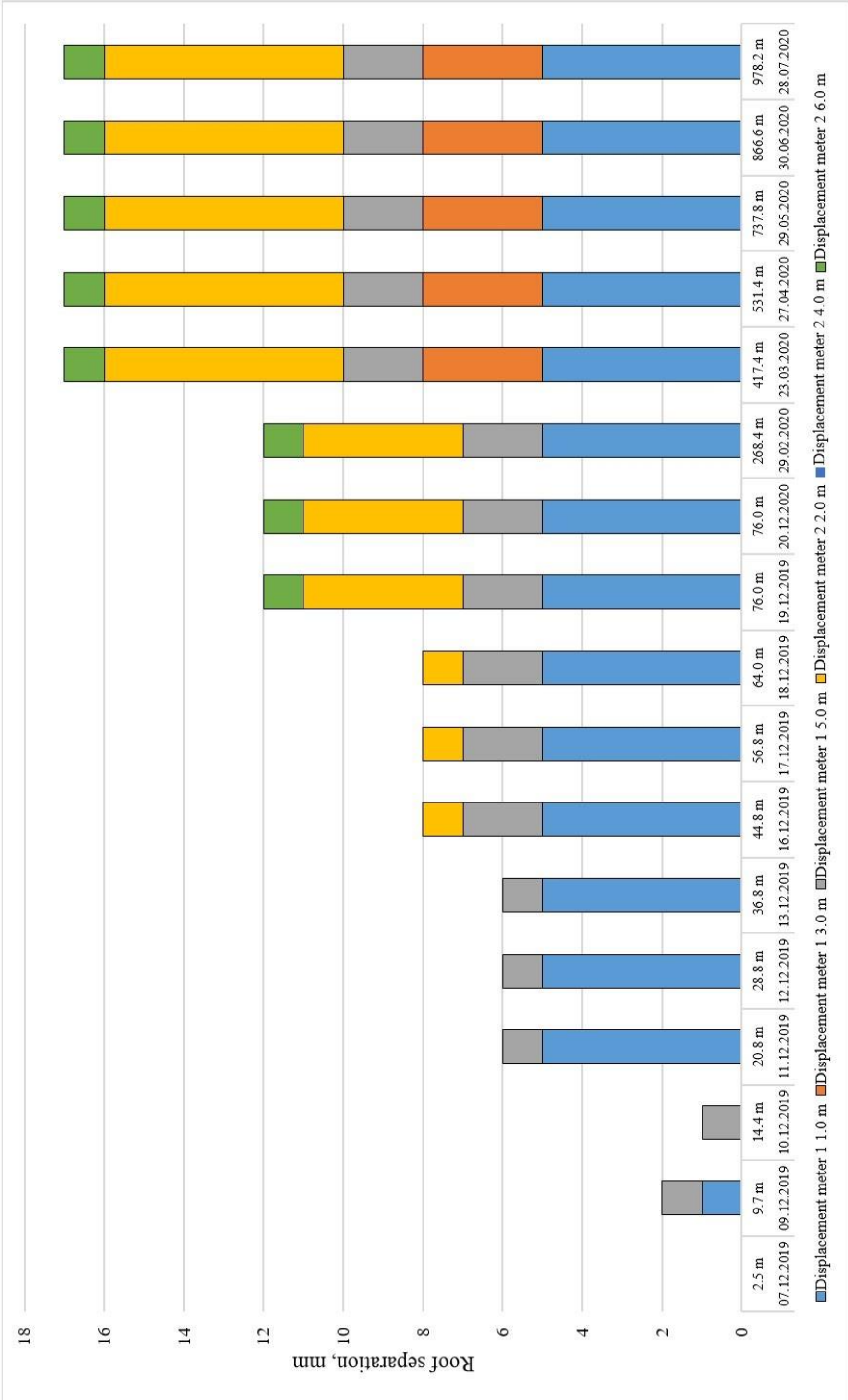

Fig. 12.25. Separations at the mechanical displacement meter – station No. 2 [135]

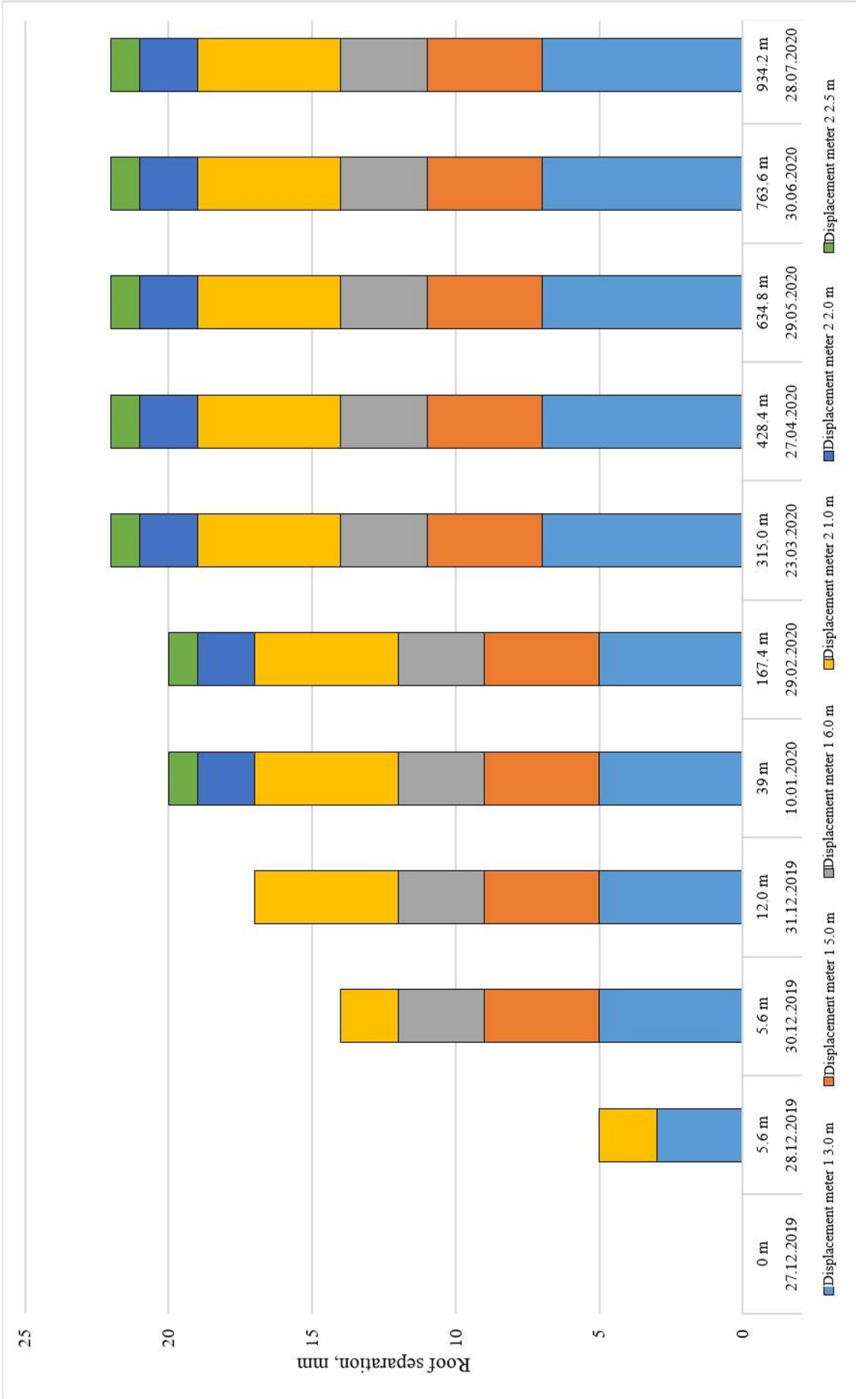

Fig. 12.26. Separations at the mechanical displacement meter – station No. 3 [135]

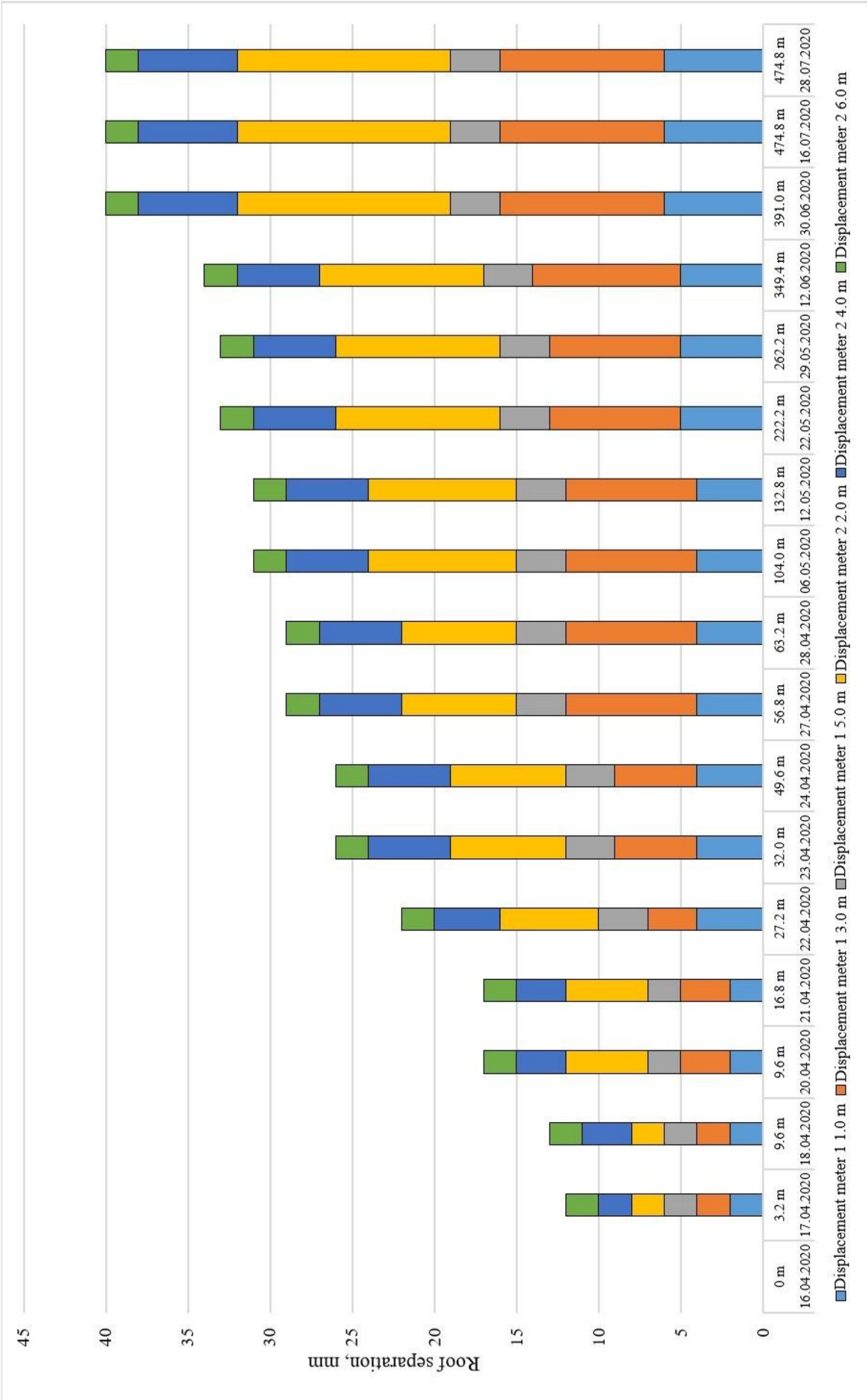

Fig. 12.27. Separations at the mechanical displacement meter – station No. 4 [135]

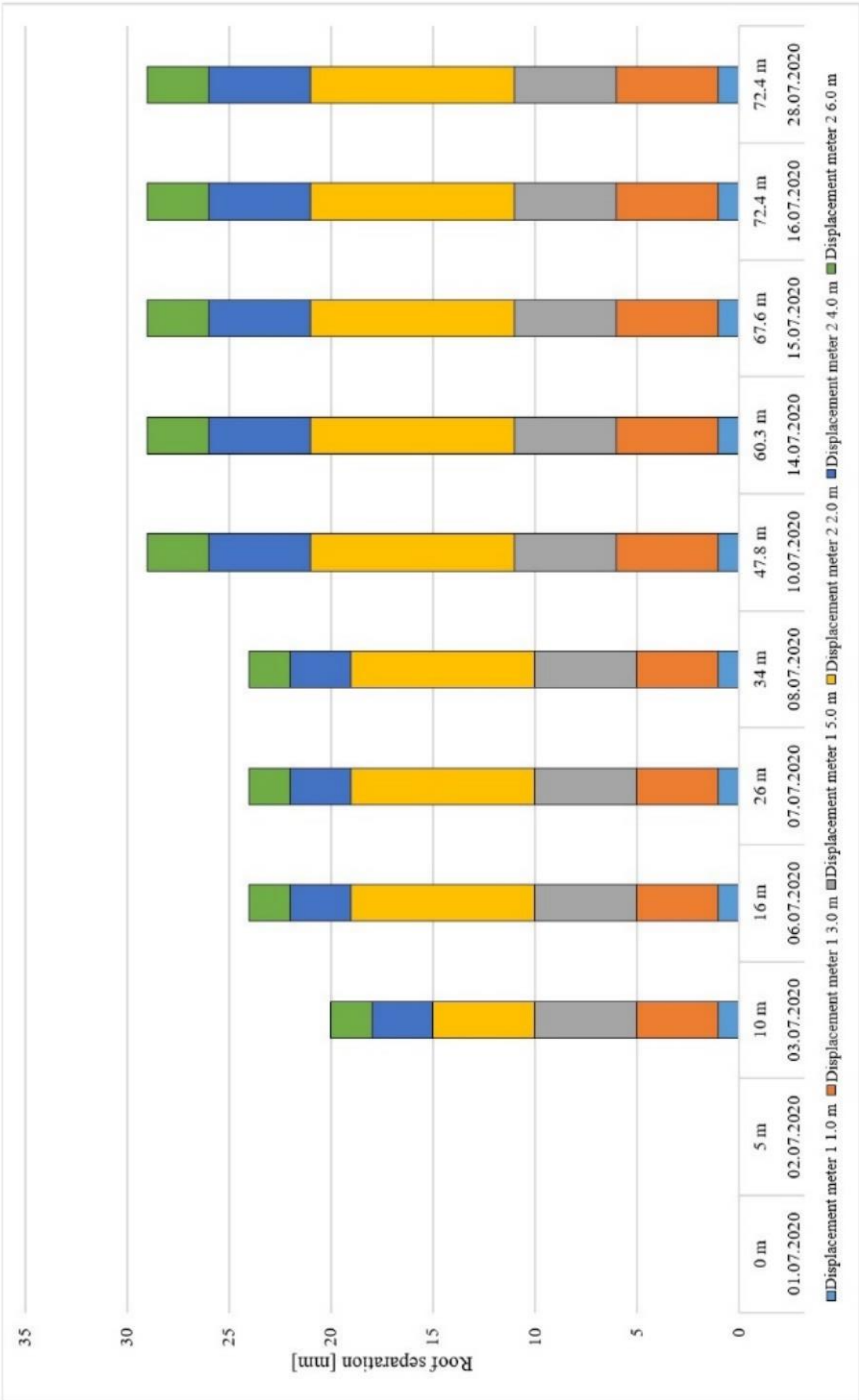

Fig. 12.28. Separations at the mechanical displacement meter – station No. 5 [135]

Load control of bolts

A load control of bolts is conducted with use of bolts with instrumentation, equipped with extensometers. Intervals of axial forces between negative and positive values confirm an occurrence of a fissure or a separation between the levels. Horizontal movements of separating rock strata, bound with the roof bolting support may cause a hazard of bolts shearing, which in turn will result in noticeable amplitudes of forces values between individual levels [135].

At the stations, situated far away from the face front (stations 1 - 4), a load stabilization is noticeable. A minor deviation is mainly caused by a very high sensitivity of extensometers to any movements of the rock mass. A phenomenon of bolts load stabilization is particularly visible at the station No. 4, for which values of axial forces have stayed practically unchanged since the moment of driving off the front face to the distance of 70 m from the measuring point [135].

At the measuring station No. 5, situated in the vicinity of the face front, very high loads, causing a destruction of measuring extensometers even during the first readings, can be observed. This phenomenon is caused by difficult geological conditions and a significant densification of the fractures network occurring in the area of the measuring point. It causes a displacement of the rock strata and thus significant values of axial forces in the bolts [135].

Values of axial forces in bolts are calculated with use of a specialistic software EXBOLT [135].

Control of working convergence

Similarly to the roof separations and bolts loads, there is a clear correlation between the values of the convergence increase and the distance of the measuring station to the face front. At the stations 1-4 a convergence increase was observed in the direct vicinity of the face front. However, for quite a long time changes of the working overall dimensions have not been observed. A different situation occurs at the station No. 5 where a dynamic convergence increase, resulting from a proximity of the face front and disadvantageous geological conditions, occurs. The graphs of the working transverse dimensions changes at the individual stations are presented in Fig. 12.29-12.33.

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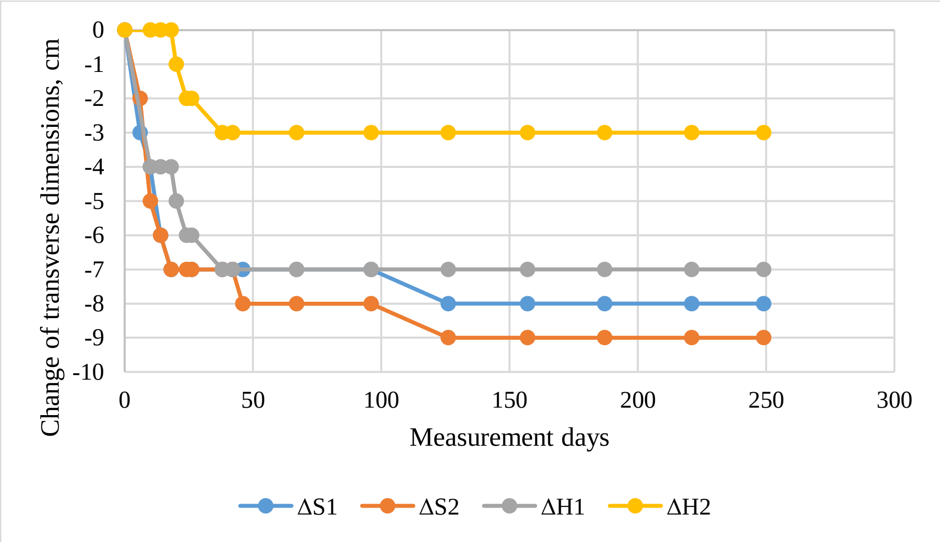

Fig. 12.29. Change of the working transverse dimensions – convergence measuring station No. 1 [135]

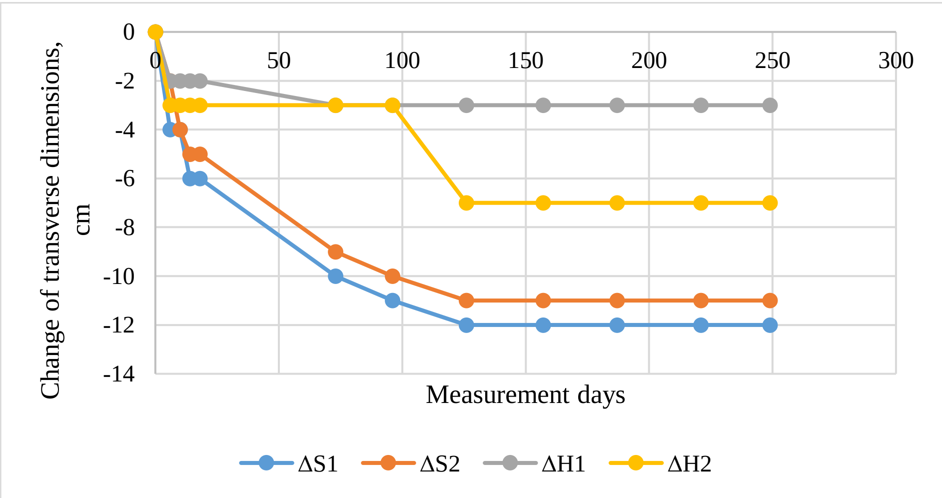

Fig. 12.30. Change of the working transverse dimensions – convergence measuring station No. 2 [135]

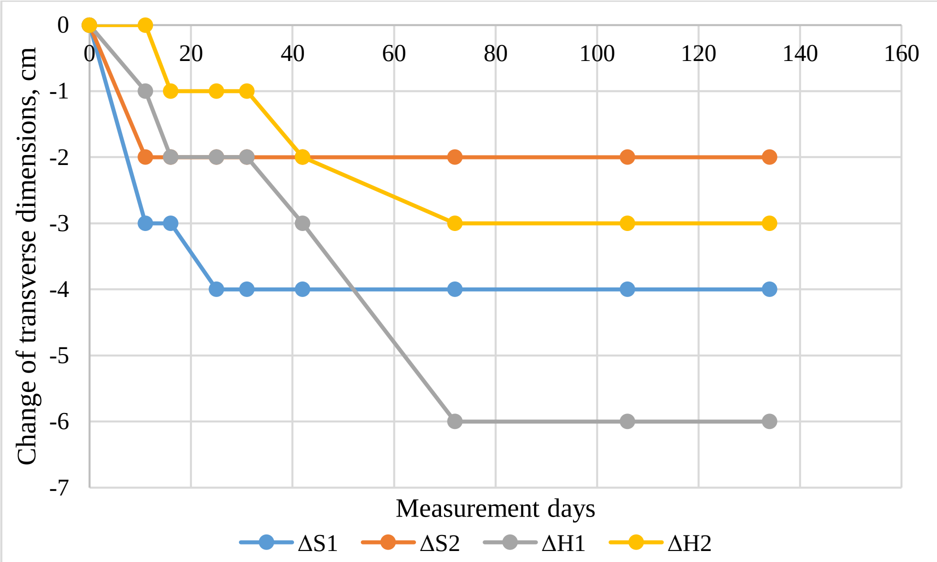

Fig. 12.31. Change of the working transverse dimensions – convergence measuring station No. 3 [135]

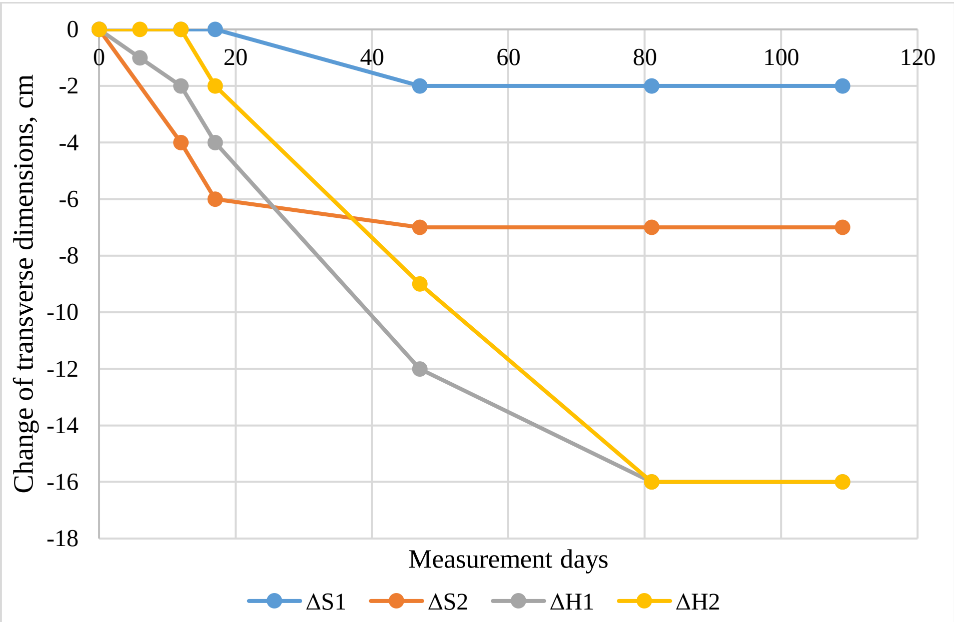

Fig. 12.32. Change of the working transverse dimensions – convergence measuring station No. 4 [135]

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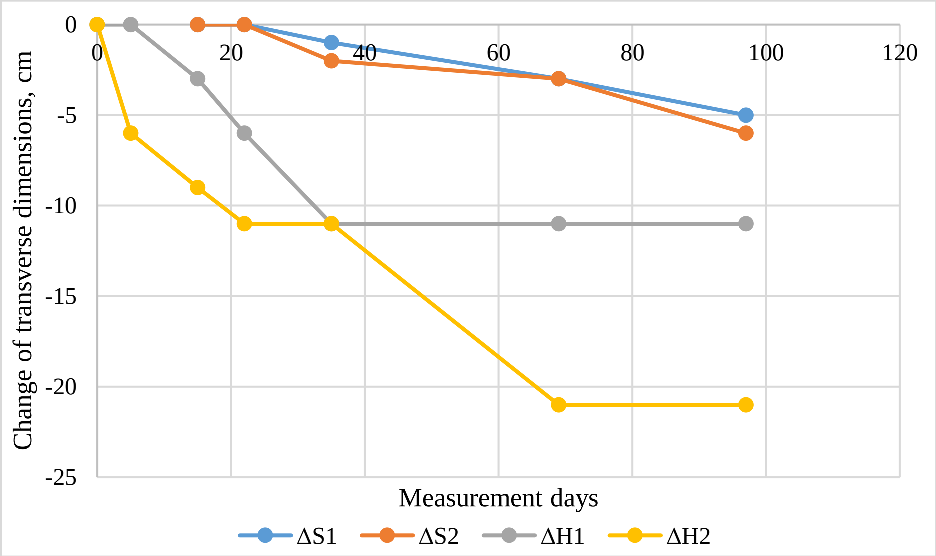

Fig. 12.33. Change of the working transverse dimensions – convergence measuring station No. 5 [135]

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13. Analysis of driving bends during the drivage of the Bw-1n test roadway with use of the Bolter Miner machine

Adam Kamiński[1], Jacek Korski[1], Marek Majcher[1], Bartosz Polnik[2], Sylwia Pietras[1]

Bolter Miner machines, in principle, enable a drivage of bends, resulting from the technique of driving multiple workings (e.g. double). In Fig. 13.1. a model of a bend realization at the angle of 90° with use of the Komatsu/Joy Global 12CM30 machine, developed by the producer, is presented. This model was elaborated with use of the Shuttle Car vehicles for a haulage of the run-of-mine, commonly used in the mining industry.

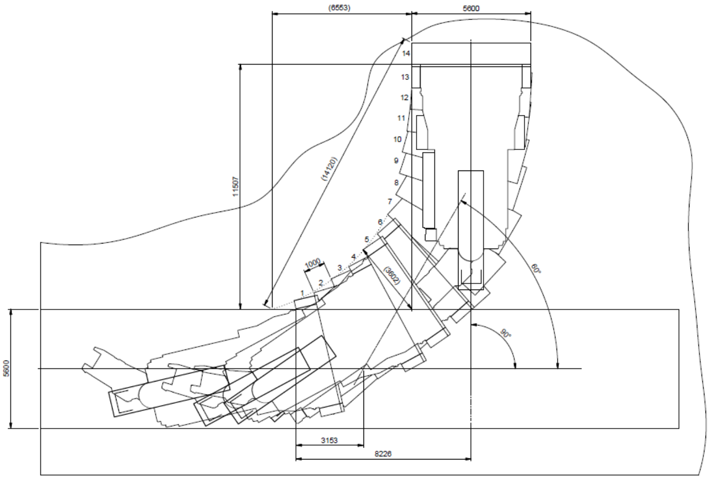

Fig. 13.1. Model of the bend drivage at 90° with use of the 12CM30 machine according to analyses of Komatsu/Joy Global

In the case of the solution of the run-of-mine haulage, applied in the JSW S.A. – Budryk mine, the Bolter Miner machine operates with the run-of-mine haulage system consisting of stationary belt conveyors (Bogda), belt conveyors (Boa) and Sigma belt feeder suspended to the machine, what unfortunately impedes the machine manoeuvring, in particular while driving bends.

During a realization of the project "Independent Roof Bolting Support" two bends were made:

– the bend No. 1 (Phase II) at the angle of 135°,

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[2] KOMAG Institute of Mining Technology

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– the bend No. 2 (Phase IV) at the angle of 146°.

In connection with conducting a more precise seam identification with the Bw-1n test roadway, taking into consideration new geological conditions, a new course of the Bw-1n test roadway was designed, in which a realization of the bend at the angle as big as 77° was planned.

Due to the fact that so called bends were made after having driven the working, from which the drivage of the new working outside the external side wall of the new working was started, it seems that a more correct name will be a crossing and not a bend.

13.1. Drivage of the bend (crossing) No. 1

The bend (crossing) No. 1, embracing the section from 216 to 310 m of the working, changed the direction of the Bw-1n test roadway drivage from the north-eastern to the eastern one. More exact data concerning the parameters of the support and the bolting network (developed by GIG) are given in Chapter 5.4.2.

An execution of the bend (crossing) No. 1 was realized over the period from 14.01. till 05.02.2020 according to the scheduled work programme:

- since **14.01.2020** – preparatory work for the machine withdrawal, machine withdrawal, a protection of the face in the blind part of the working, its strengthening through a construction of wooden chocks and SV-type chocks, a reconstruction of haulage installations. Widening of the roadway on the right side wall, in the place of the future bend, bolting of side walls,
- since **21.01.2020** – a termination of cutting the right side wall (an execution of the arch section), advance start of the straight widened section 6.8 m,
- since **05.02.2020** – according to the reported hectometric area and the drivage design **-** since that date a drivage of the normal working width – 5.6 m has started, which was recognized as the finish of the bend execution according to the design.

In Fig. 13.2 a distribution of activities during a drivage of the bend (crossing) No. 1 and in Fig. 13.3 its drivage advance are presented.

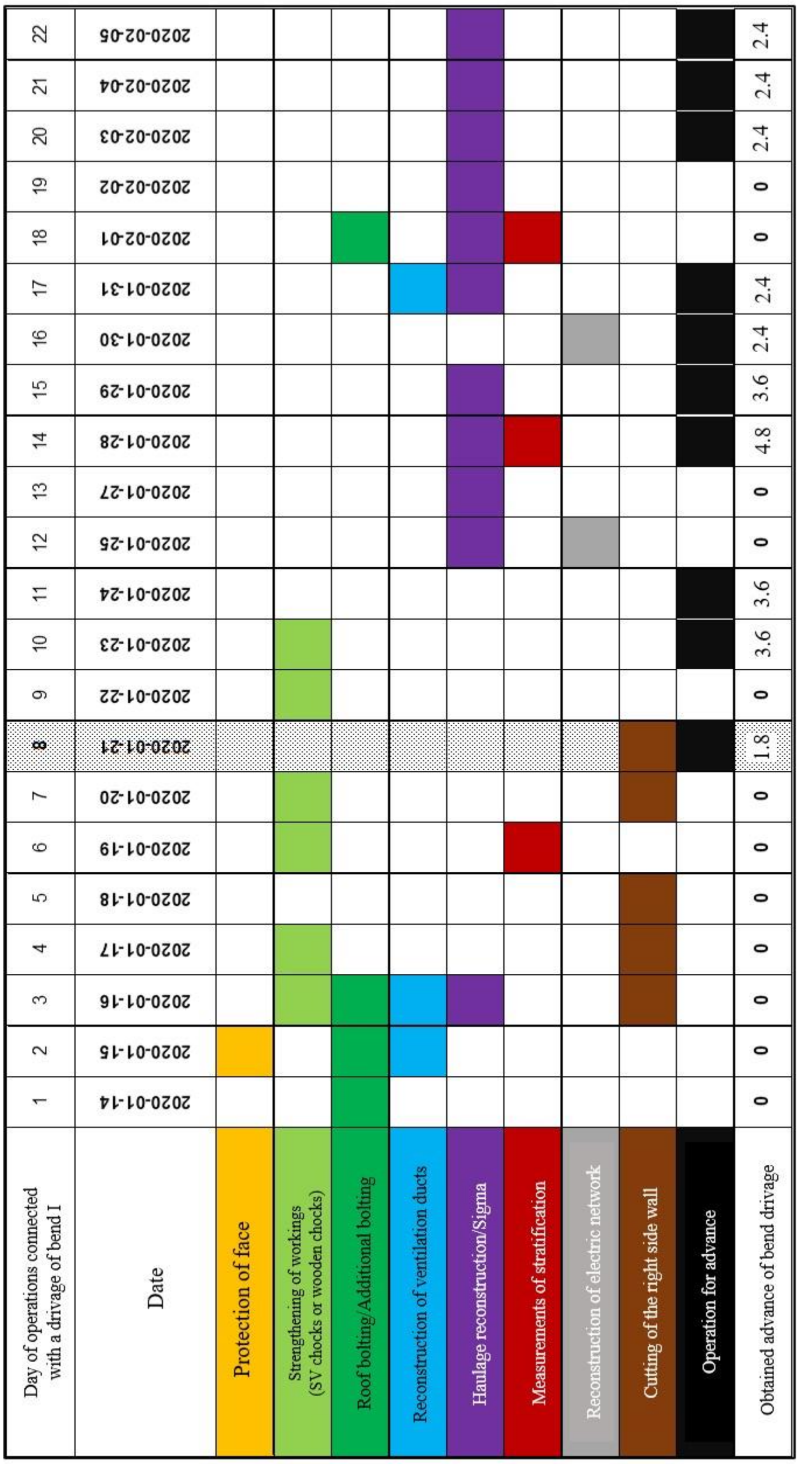

Fig. 13.2. A distribution of activities during an execution of the bend (crossing) No. 1

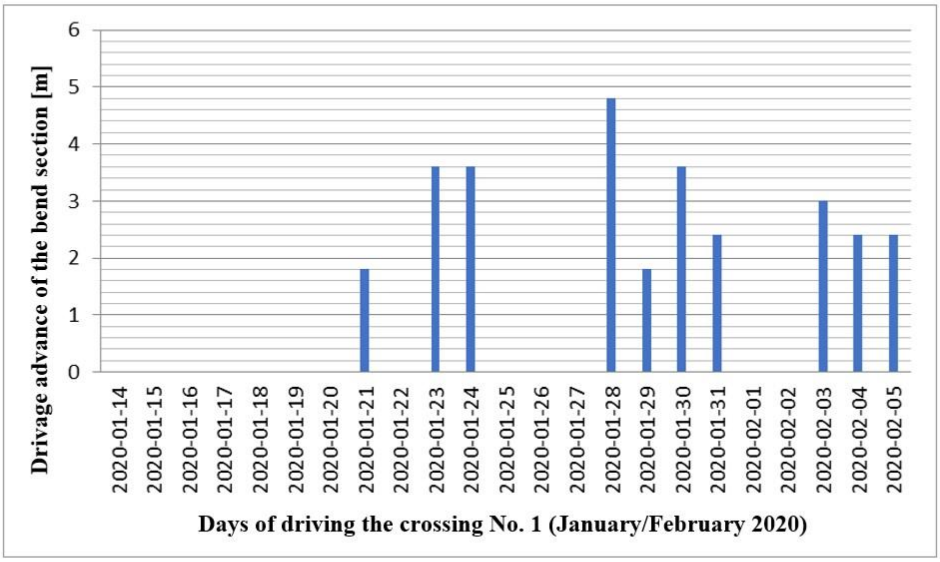

Fig. 13.3. Drivage advance during an execution of the bend (crossing) No. 1

13.2. Drivage of the bend (crossing) No. 2

Due to a change of geological conditions it was decided to change a location of the bend (crossing) No. 2. Based on an underground inspection on 20.07.2020, it was stated that the most optimum bend location, from the point of view of geological and technological aspects, is on the 984 ÷ 1009 m mark. Finally, it was decided that the junction will be made at the mark 994 m. More exact data concerning the parameters of the support and the bolting network (developed by GIG) are given in Chapter 5.4.2.

An execution of the bend (crossing) No. 2 was realized over the period from 17.07. till 18.08.2020 according to the scheduled work programme:

– since **17.07** till **21.07.2020 -** 3 workdays during which the machine was withdrawn (a decision about an earlier start of making the bend, after a change of geological conditions in the face),

– since **22.07** till **23.07.2020** – dinting, a construction of isolation dam (cutting off the blind part of the working), an installation of reinforcements at the isolation dam (a construction of wooden chocks and SV type chocks), a reconstruction of haulage, installations,

– since **28.07.2020** – work in 5-shift system,

– **29.07.2020** – widening of the roadway on the right side wall, start of driving the arch section,

- **5.08.2020** – finish of the bend (arch section), advance start of straight, widened section to 6.8 m (according to the reported hectametric area and the design),
- **18.08.2020** – since that date, driving of the normal working width – 5.6 m has been started, which can be treated as a termination of the bend execution according to the design.

In Fig. 13.4 a distribution of activities during a drivage of the bend (crossing) No. 2, and in Fig. 13.5 its drivage advance are presented.

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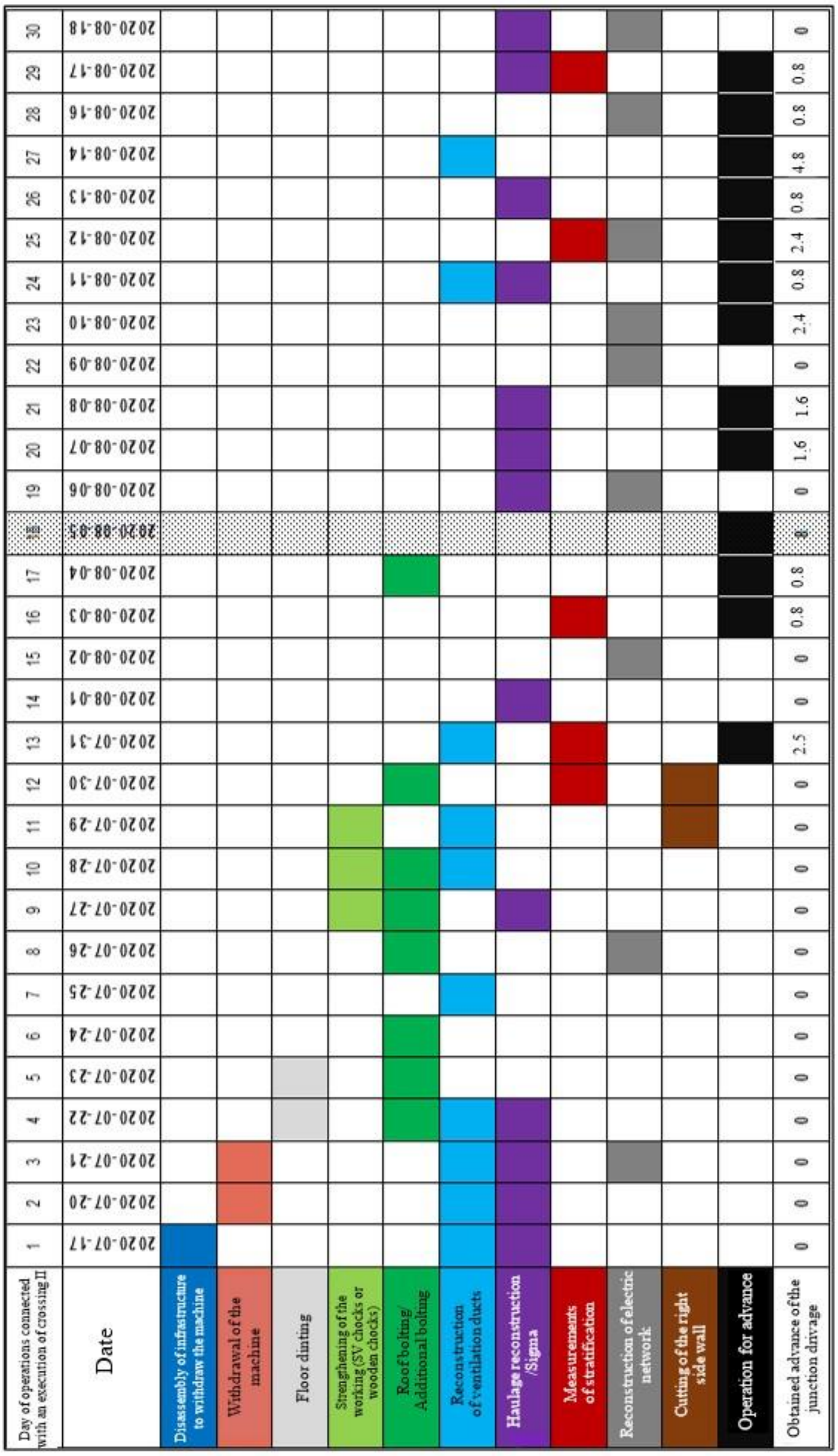

Fig. 13.4. A distribution of activities during an execution of the bend (crossing) No. 2

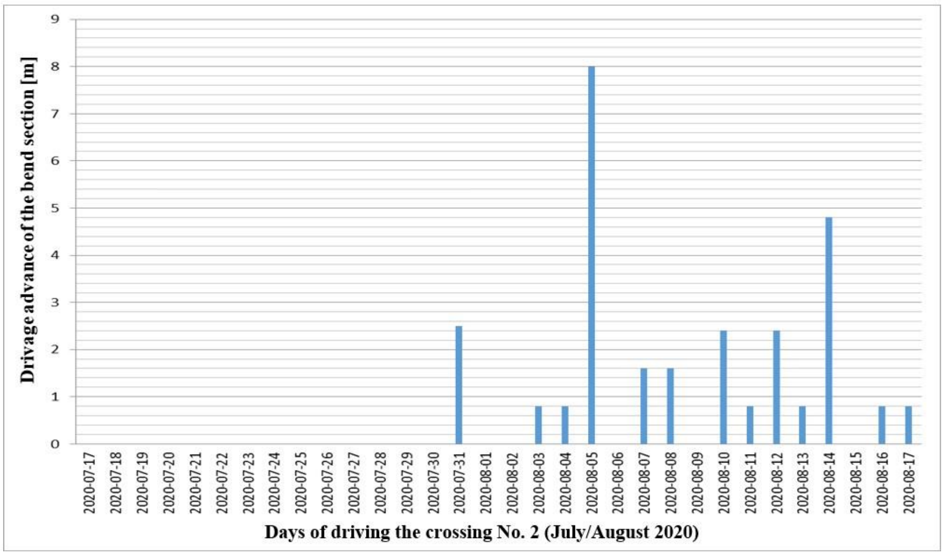

Fig. 13.5. Drivage advance during an execution of the bend (crossing No.2)

13.3. Drivage of the bend (crossing) No. 3

The bend (crossing) No. 3 differentiated significantly as regards the angle of drivage direction from the bends (crossings) No. 1 and No. 2. The bend (crossing) No. 3 at the angle of 77° (in fact 103°) in relation to the initial drivage direction was to be executed on the basis of the design elaborated by the specialists from the Central Mining Institute, taking into consideration the roof and side walls bolting of the working. Due to the fact that the crossing was made as a junction from the existing workings, it was necessary to consider a large roof opening and, in the design, the final support of the crossing was a mixed support (bolting of the roof and side walls and wooden chocks of box construction).

The crossing No. 2 was executed over the period from 26.09. till 5.11.2020 divided for the following operational periods:

- since **26.09** till **5.10.2020** (7 workdays) – operations enabling a withdrawal of the machine and its withdrawal as well as an additional strengthening of the roof (an installation of additional bolts),
- since **6.10** till **19.10.2020** (14 workdays) – cutting of the workings right side wall and its widening to an arch section (bolting and indispensable reconstruction of the haulage system and of the ventube ventilation),
- since **19.10** till **31.10.2020** (10 workdays) – on 19.10 a break-down of the machine drum which stopped an advance of driving the arch section. During

the break-down additional bolting operations and auxiliary activities connected with a reconstruction of the haulage system, ventube ventilation, electrical network or strengthening the working with use of wooden chocks, were performed,

– since **2.11** till **3.11.2020** (2 workdays) – the arch section has been finished,
– **4.11.2020** – a drivage of widened section of the rectilinear working was started,
– **5.11.2020** – a termination of the bend drivage (of arch section) together with making 10 m of the working section behind the bend widened to 6.8 m.

During a drivage of the crossing No. 3 the cutting drum of the 12CM30 machine was damaged due an occurrence of sphaerosiderite concretion in the roof. However, the photos taken after the break-down (Fig. 13.6) indicate also some failures to do current exchanges and supplements of cutting tools. A necessity of repair caused a nearly 10-day break in the machine operation.

Fig. 13.6. A view of damaged cutter-bar of the 12CM30 cutting drum

In Fig. 13.7 a distribution of basic activities connected with the bend (crossing) No. 3 drivage is presented.

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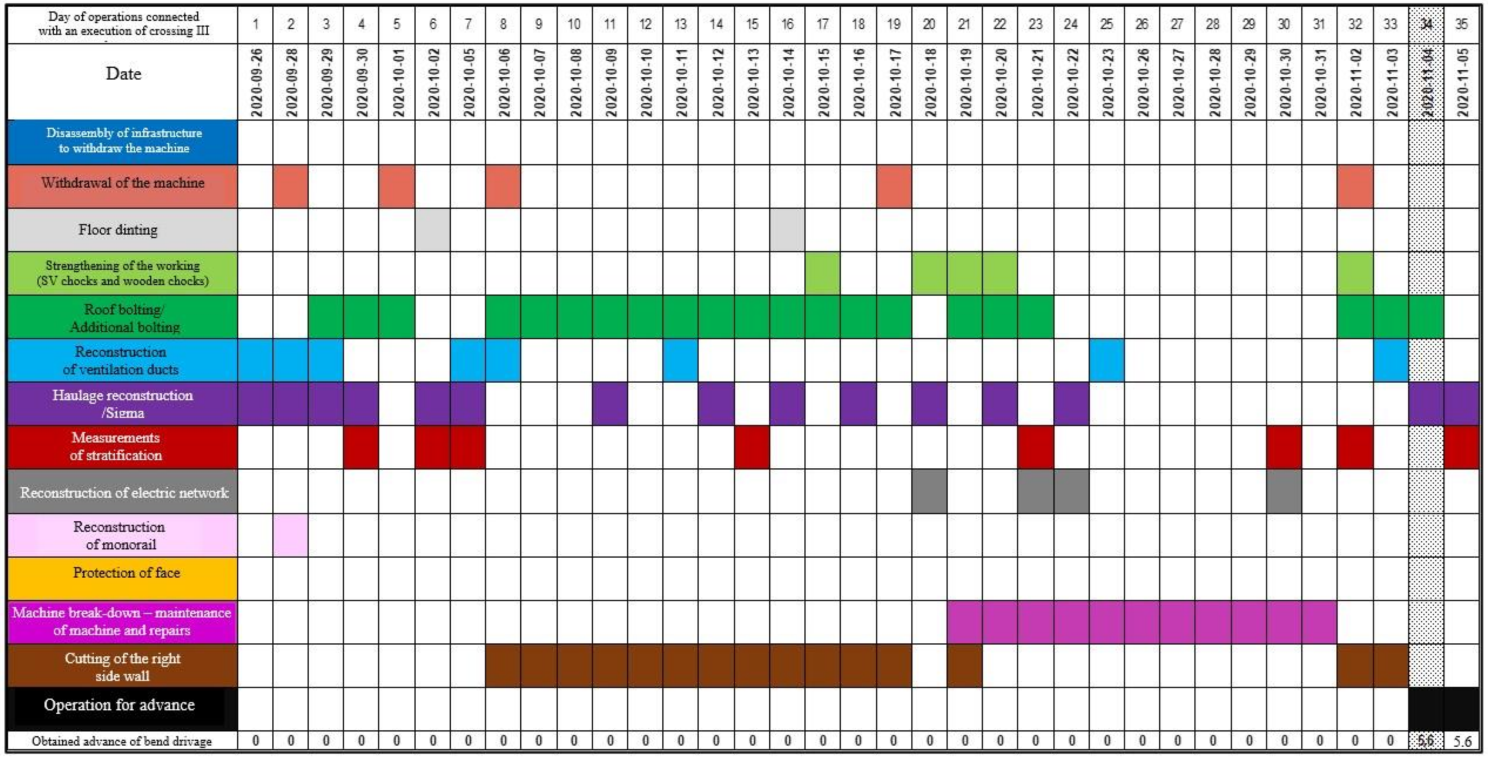

Fig. 13.7. A distribution of activities during a drivage of the bend (crossing) No. 3

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13.4. Analysis of time spent on drivage of bends (crossing) No. 1, 2, 3

In Table 13.1 time of individual operations at driving bends (crossings) No. 1, 2, 3 is presented.

Time statement of individual operations at driving bends (crossings) No. 1, 2, 3

Table 13.1.

	Bend I	**Bend II**	**Bend III**
Total number of workdays for a drivage of arch sections	8	18	34
Total number of workdays for a drivage of arch sections – **excluding the days with the machine drum break-down**	8	18	24
Preparatory operations enabling a withdrawal of the machine and cutting	3	10	6
Full day withdrawal of the machine	0	2	2
Cutting of the right side wall – drivage of the arch section (excluding the time of machine break-down)	5	6	16

For the operations connected with a drivage of individual bends (until the arch section was made) the time was used as follows:

- the bend No. 1 - 8 days,
- the bend No 2 - 18 days,
- the bend No 3 – 34 days, excluding the time of the cutting drum break-down – 24 days.

The preparatory activities (such as a reconstruction of haulage, of ventube ventilation, of electrical network, strengthening of the working with additional bolting or chock support), enabling to cut the right side wall or in the case of a necessity a withdrawal of the machine, lasted respectively for the bends: No. 1 – 3 days, No. 2 – 10 days, No. 3 – 6 days. An extension of this time for the bend No. 2 could be related to a necessity of the machine withdrawal at a longer section (in connection with a decision about an earlier start of the bend drivage, after the change of geological conditions). A withdrawal of the machine for driving the bend took place only for the bends No. 2 and 3. In both cases it lasted 2 days each.

During a drivage of the bend No. 2 since 28.07.2020 the work has been organized in the 5-shift system (which increased the time of the staff presence in the face and for sure it had an impact on a quicker drivage of the arch section which was realized within 6 days).

The bend No. 3 was redriven in the 4-shift system, which explains partly a much longer time of driving the arch during the drivage of the bend No. 3 – 16 days. The reason may also be a sharp angle of this bend (77°) and difficulties with a technical realization of the project.

13.5. Drivage of the bend (crossing) No. 3 in the light of the project

The design of the bend (crossing) No. 3 (Fig. 13.8), elaborated at the Central Mining Institute, took into consideration the limitations resulting from the Polish mining regulations and from ensuring the working stability, however it omitted the issue of a later use and functionality of the working driven in such a way. The bolting support of the hypothetical bend (crossing in fact) was designed with the greatest care, but the design did not take into consideration real possibilities of driving the crossing under consideration. The design of the crossing (bend) No. 3 was realized on the base of simulation models elaborated by the producer of the machine and it was feasible under the condition of applying a non-linear method of the run-of-mine haulage in a form of tyred vehicles (e.g. of Shuttle Car type or other of articulated - frame type). However, in the example, under analysis, belt conveyors were used exclusively. Besides, a realization of the crossing, according to the design, required very high skills and experience from the Bolter Miner machine operators. This condition was difficult to be fulfilled. Bearing the above in mind, the executors of the crossing were forced to take advantage of their own skills and experience. Lack of the workings widening in the place of starting the bend caused that making a bend by the machine was started earlier that it had been assumed in the design project. An earlier start of turning and later corrections resulted in the crossing of a bigger surface of the uncovered roof than it had been planned and having the maximum span in the middle of the arch reaching 19.2 m. After having stated this fact, experts on supports twice (15.11 and 15.12.2020) indicated a necessity of strengthening the crossing support with use of wooden chocks. In the result the crossing in reality was essentially different from that one in the design project (Fig. 13.9).

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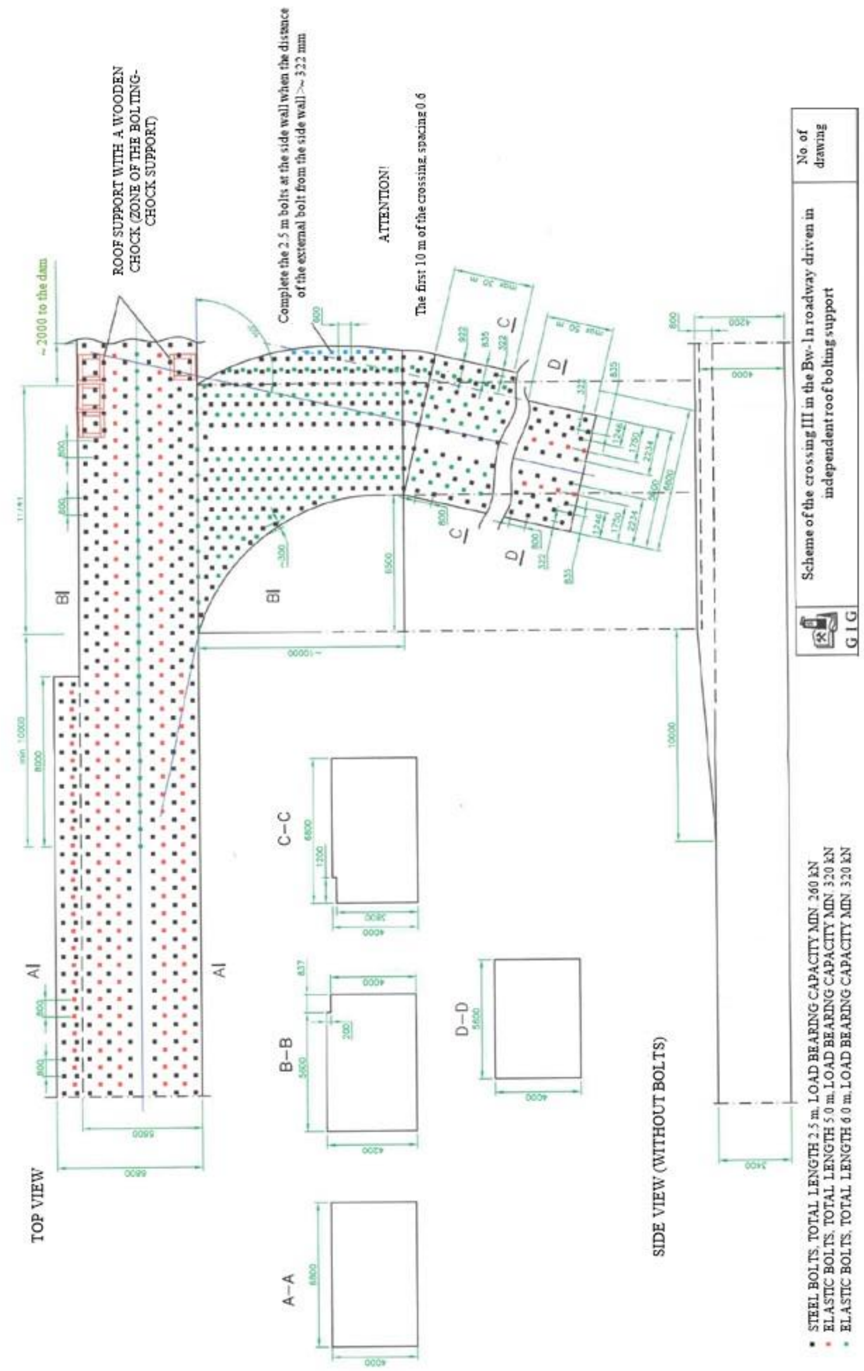

Fig. 13.8. Bolting scheme and geometry of the crossing No. 3 according to the GIG design project

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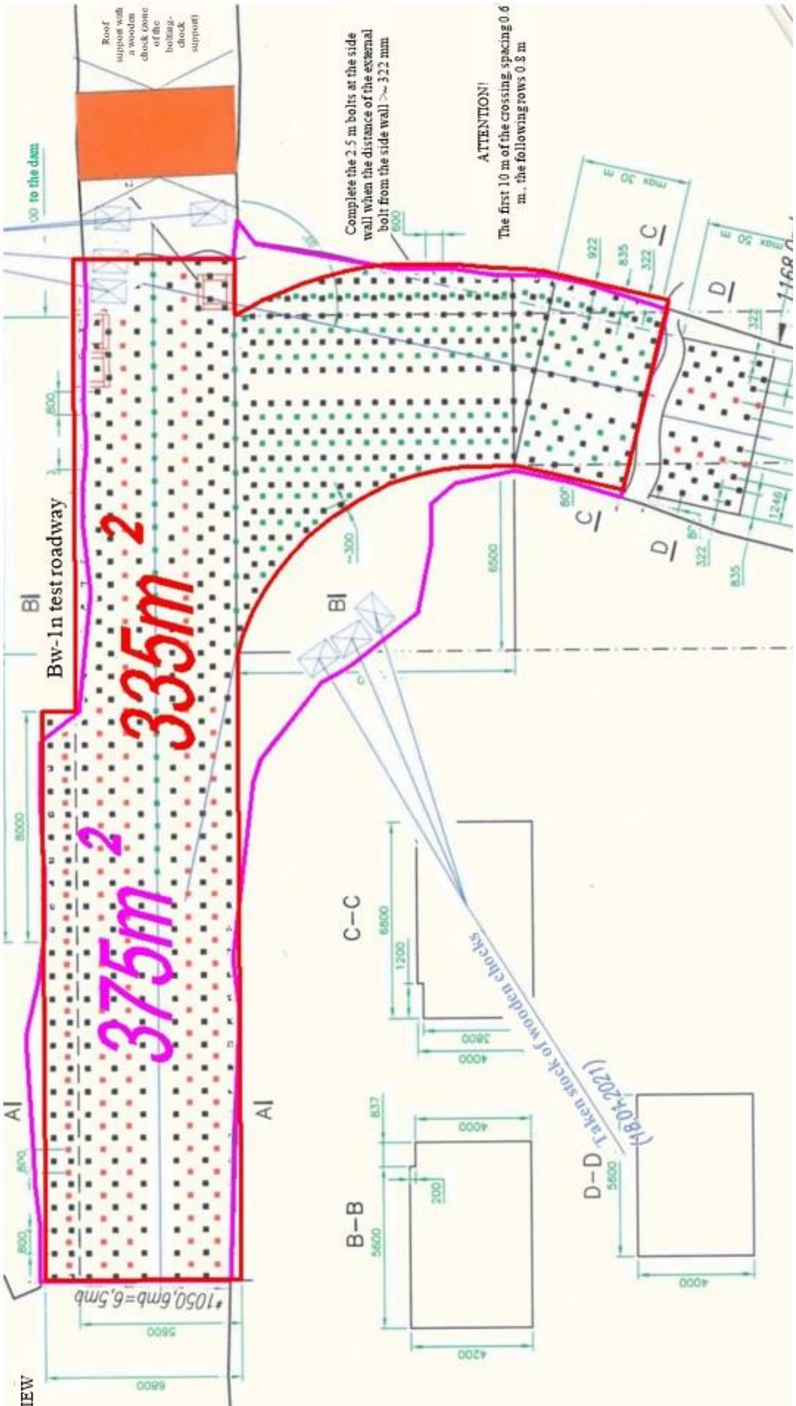

Fig. 13.9. Design of the crossing No. 3 and its real execution in the top view

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13.6. Alternative possibilities of driving the crossing No. 3 in the existing system of the devices operating together with the 12CM30 machine

During an assessment of feasibility and functionality of the crossing No. 3 an attempt of developing a technical design project of the crossing, meeting the requirements and possible to be executed with a smaller number of the Bolter Miner machine manoeuvres, was undertaken. In each of the designed situations a long departure of the machine and a gradual widening of the working, enabling a final execution of one turn of the machine at a significantly bigger angle, was needed. To enable a turn of the machine of a significant width, in relation to the working transverse dimension, it was necessary to make a widened section of the working. This widening should be executed on the opposite side (side wall) of the working in relation to the turning direction (Fig. 13.10.).

There is a possibility of widening the workings up to 6.8 m in the whole zone of the crossing (two variants: Fig. 13.11 and 13.12) with a reservation concerning the remark related to the permissible working width in independent roof bolting support according to the Polish mining regulations.

In Fig. 13.13 and 13.14 a possibility of analogical driving of bends or crossings at the angle of 90° with a longer departure distance – a withdrawal of the machine at the distance of 39 m is presented. However, in Fig. 13.15 and 13.16 the crossing (bend) at the angle of 90° with a shorter machine withdrawal distance of 29 metres is illustrated.

In Fig. 13.17. a simulation of the 12CM30 Bolter Miner turn from the driving direction so far, at the angle of 135° corresponding to the crossing geometry (bend) No. 2, is shown.

A widening of the working, at the opposite side wall to the direction of turn up to 6.8 m, is only needed in the zone of starting the machine turn (as it is shown in Fig. 13.10), and its elongation can be useful due to other technical-and organizational reasons.

The bends (in fact crossings) designed in such a way facilitate an installation of linear solutions of the run-of-mine haulage (belt conveyors) and an introduction of suspended monorails routes and they do not make such big demands to the machine operators as those included in the design project. The suggested solutions also facilitate bolting of the roof and side walls – it is easier to realize the bolting network.

Maintaining the workings span in the bolting support, according to the Polish mining regulations (up to 7.0 m), requires a consideration – from the conducted analyses it can be concluded that this condition cannot be fulfilled in the zone of forks and crossings driven with the machine of the Bolter Miner type due to its big linear dimensions.

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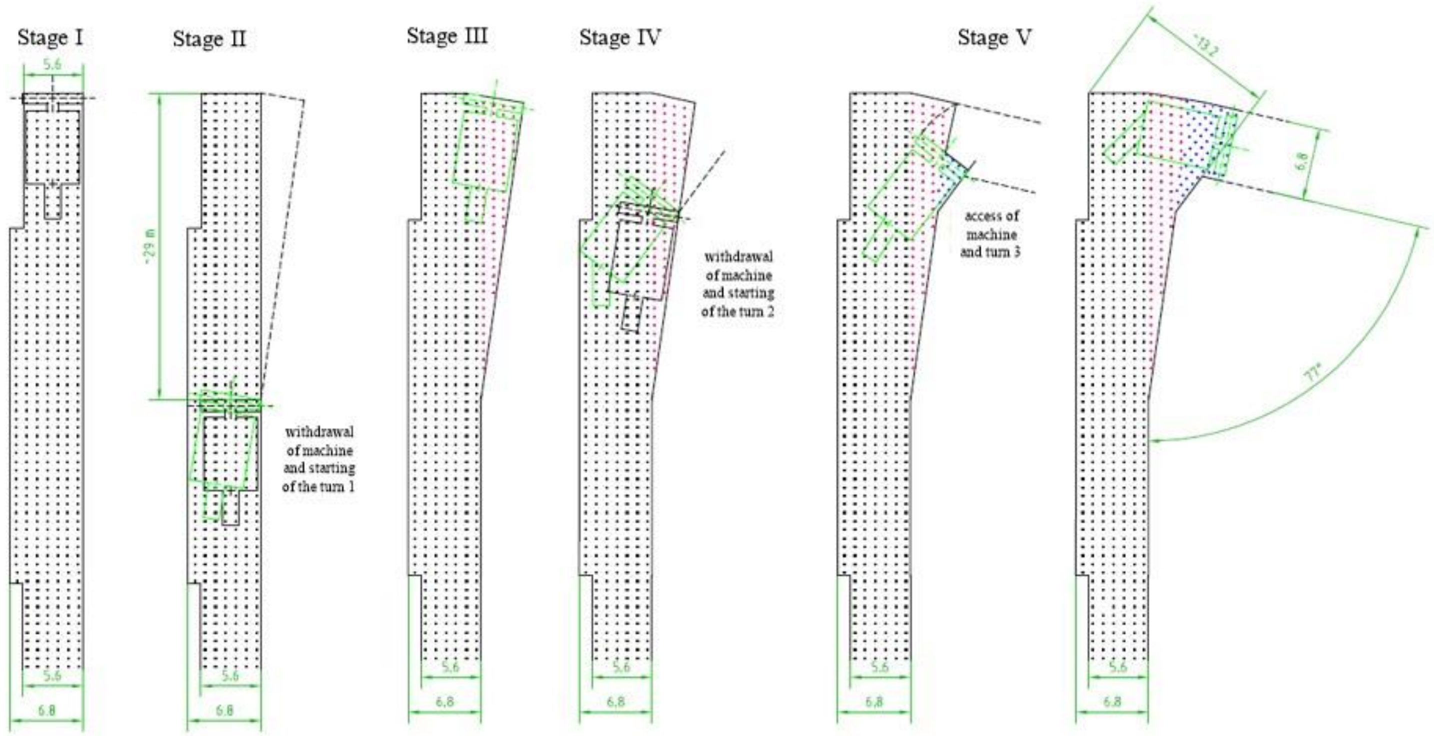

Fig. 13.10. Schematic diagram of driving the bend in the crossing No. 3 according to the Author`s conception (turn of 77°, departure from the bend of the 6.8 m width) – widening only in the turn zone

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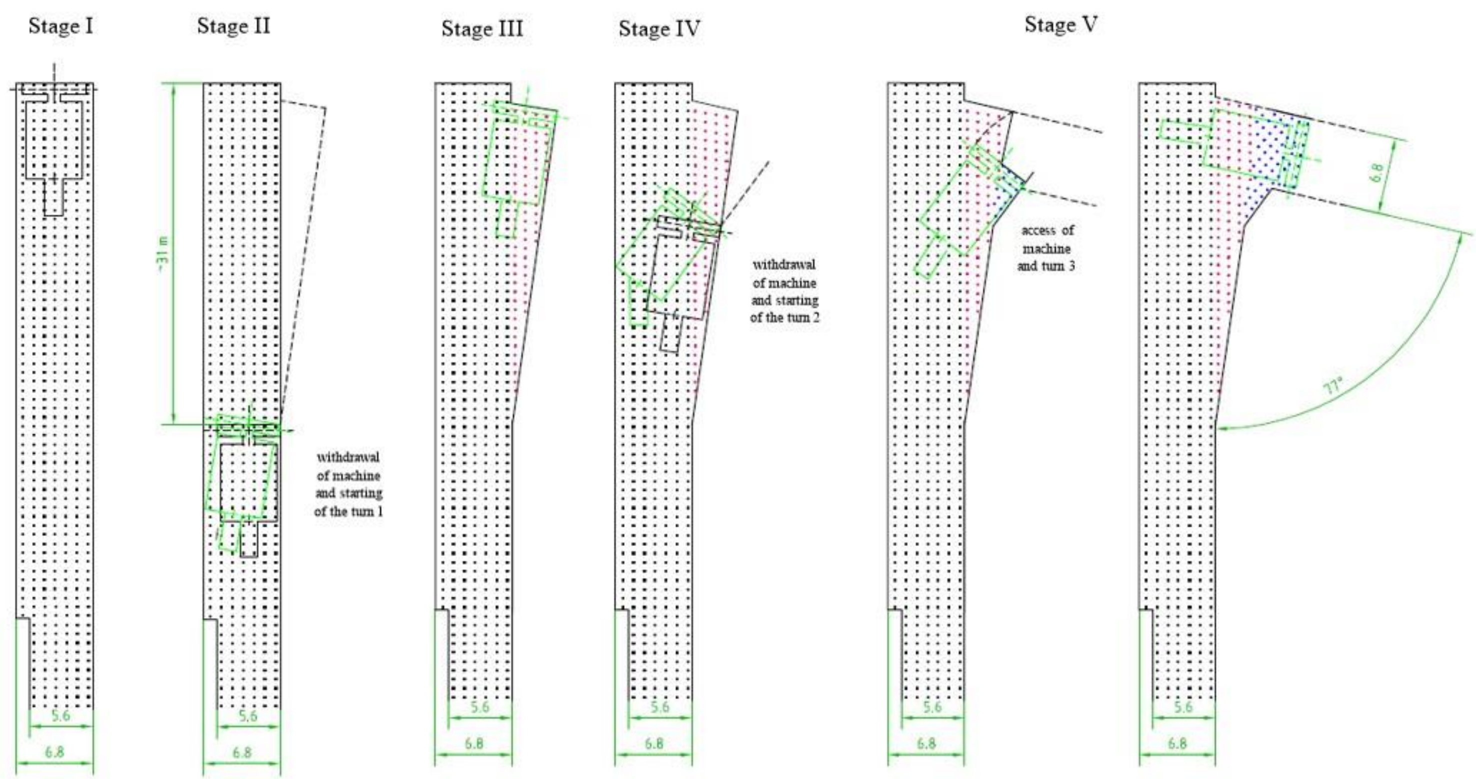

Fig. 13.11. Schematic diagram of driving the bend in the crossing No. 3 according to the Author`s conception (turn of 77°, departure from the bend of the 6.8 m width) – with widening at the full length of the crossing zone – variant I

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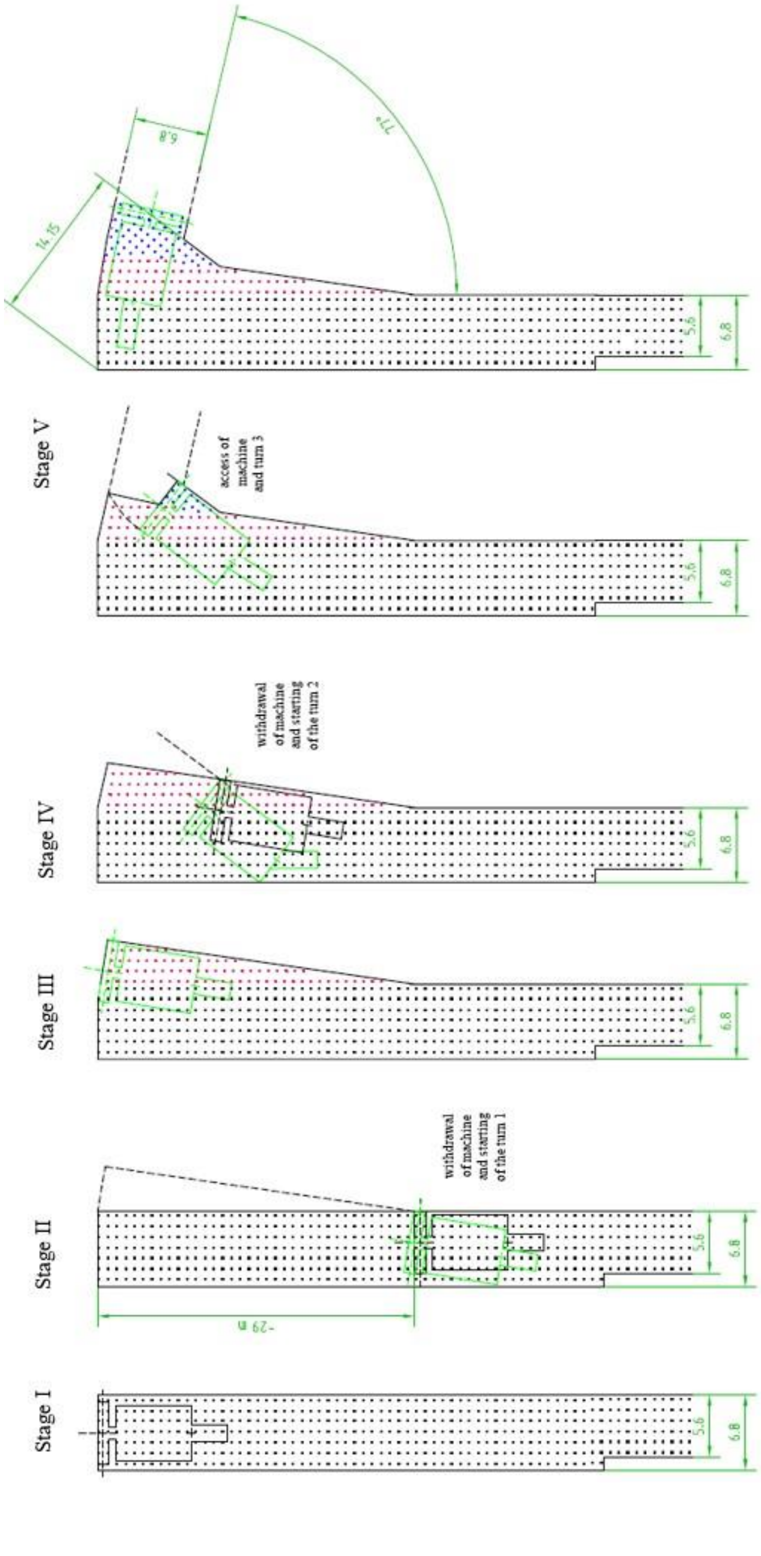

Fig. 13.12. Schematic diagram of driving the bend in the crossing No. 3 according to the Author`s conception (turn of 77°, departure from the bend of the 6.8 m width) – with widening at the full length of the crossing zone – variant II

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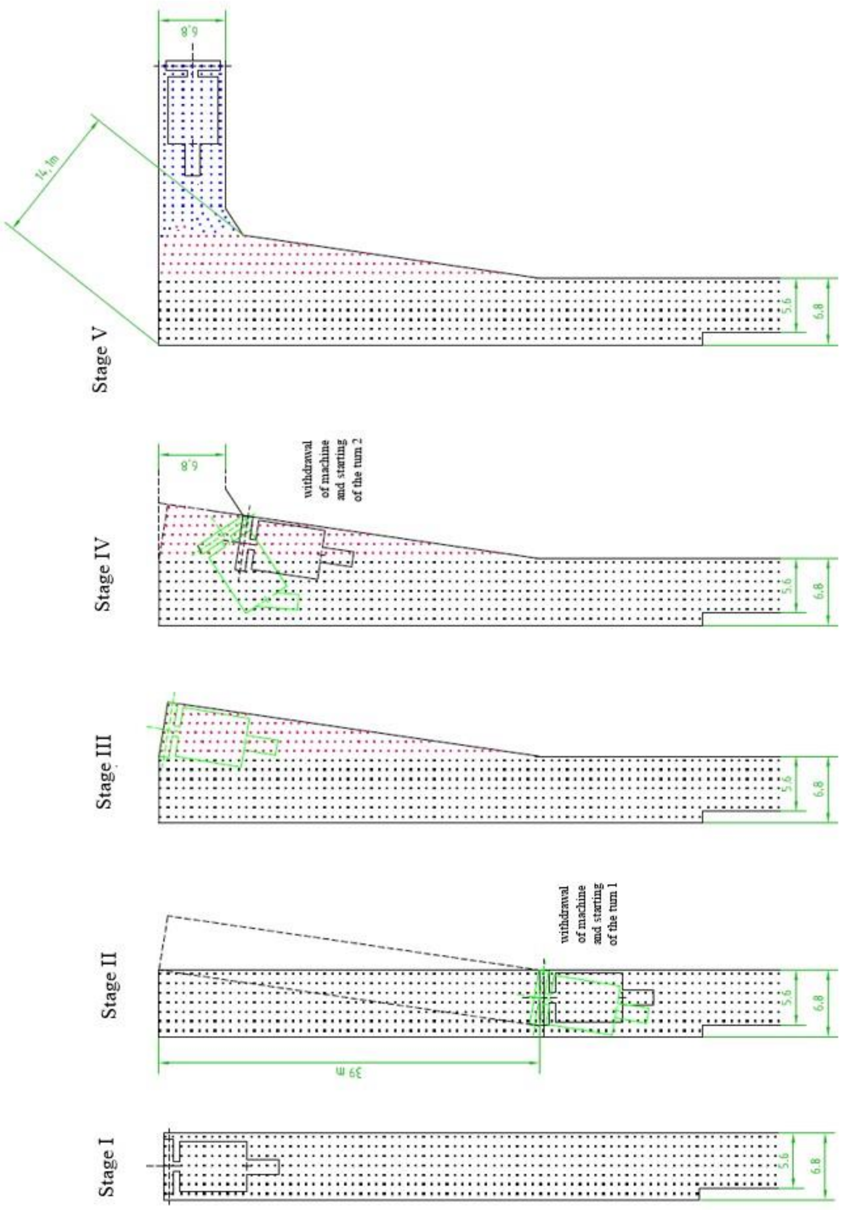

Fig. 13.13. Schematic diagram of driving the bend at the angle of 90° with a long departure - 39 m and with a wide (6.8 m) inlet to a new working

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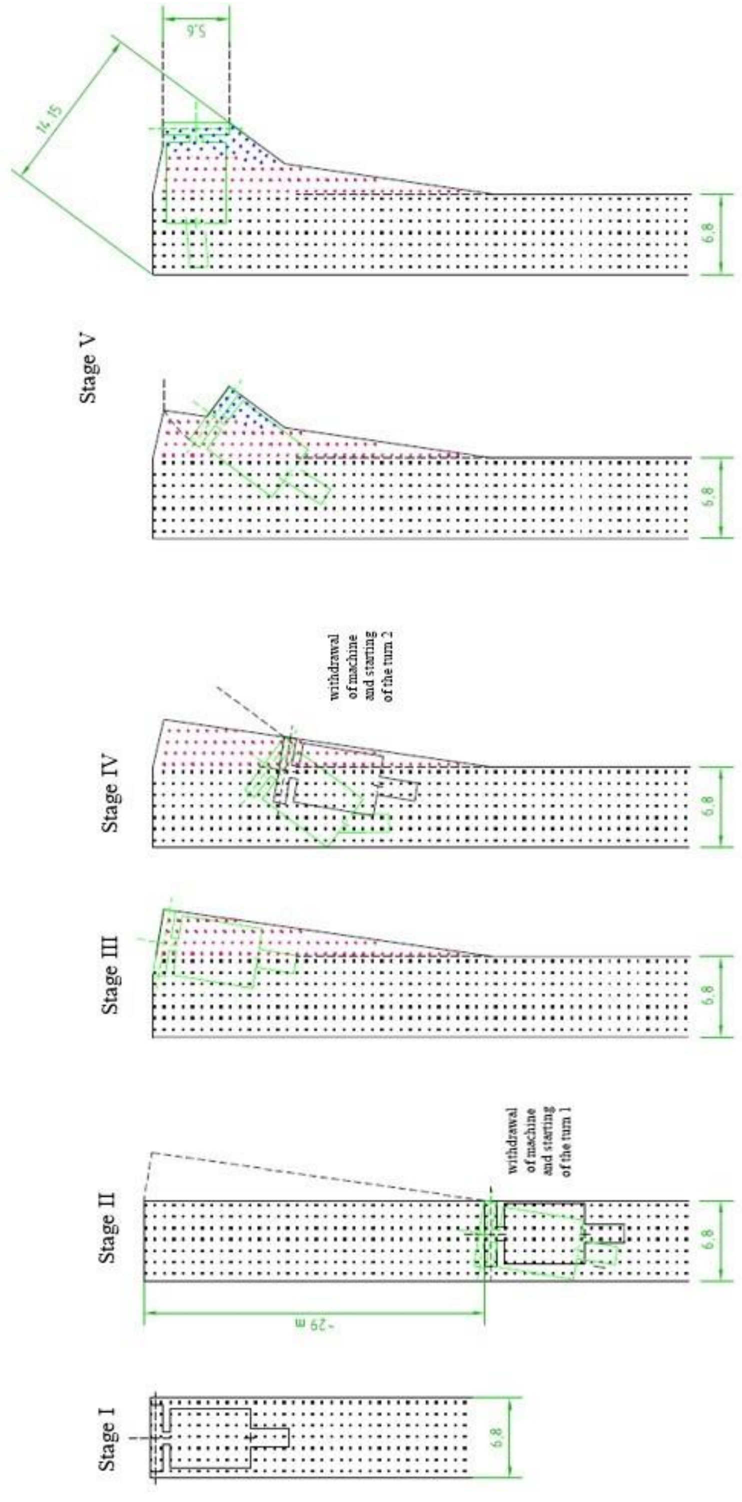

Fig. 13.14. Schematic diagram of driving the bend at the angle of 90° with a long departure - 39 m and with a narrow (5.6 m) inlet to a new working

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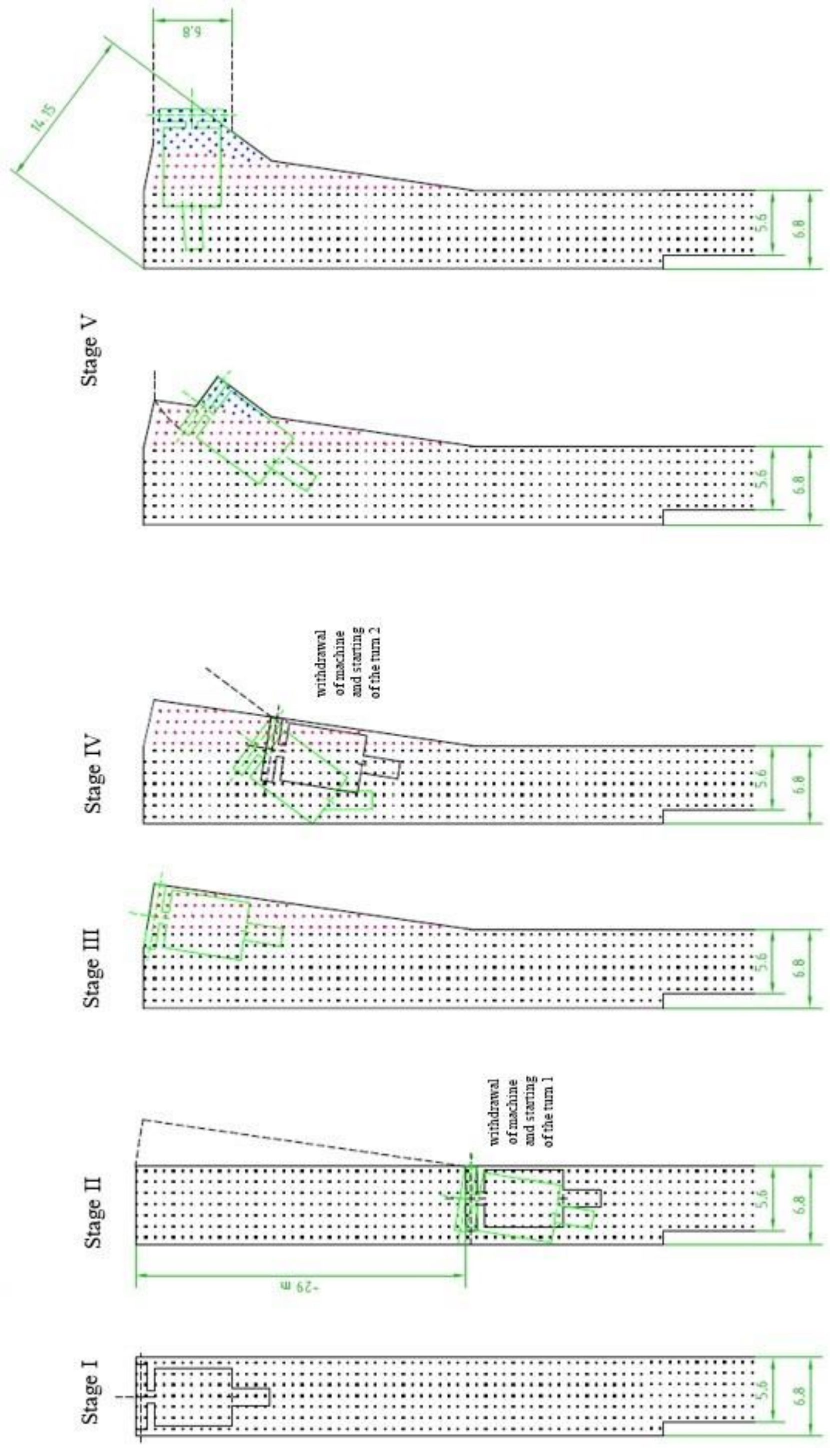

Fig. 13.15. Schematic diagram of driving the bend at the angle of 90° with a " short" departure - 29 m and with a wide (6.8 m) inlet to a new working

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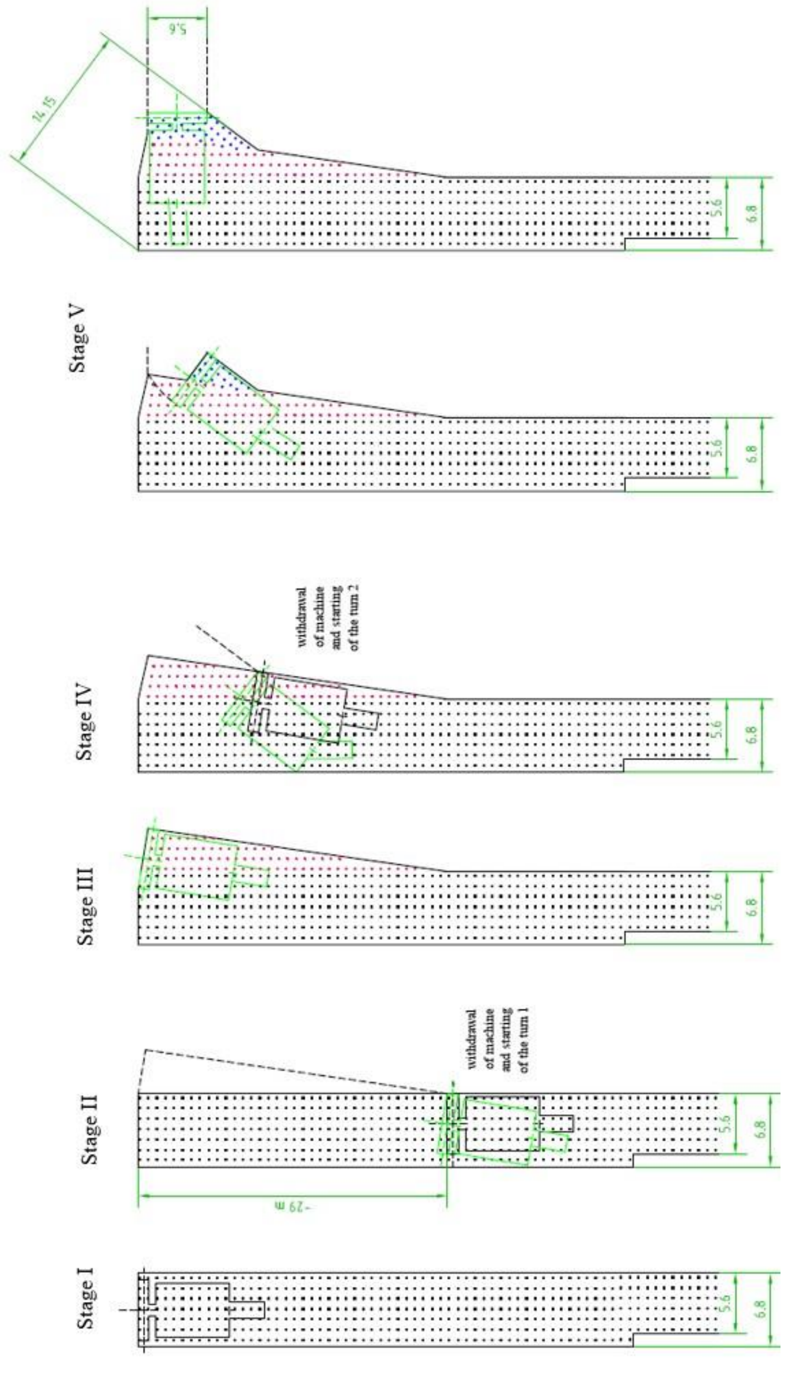

Fig. 13.16. Schematic diagram of driving the bend at the angle of 90° with a “ short” departure - 29 m and a narrow (5.6 m) inlet to a new working

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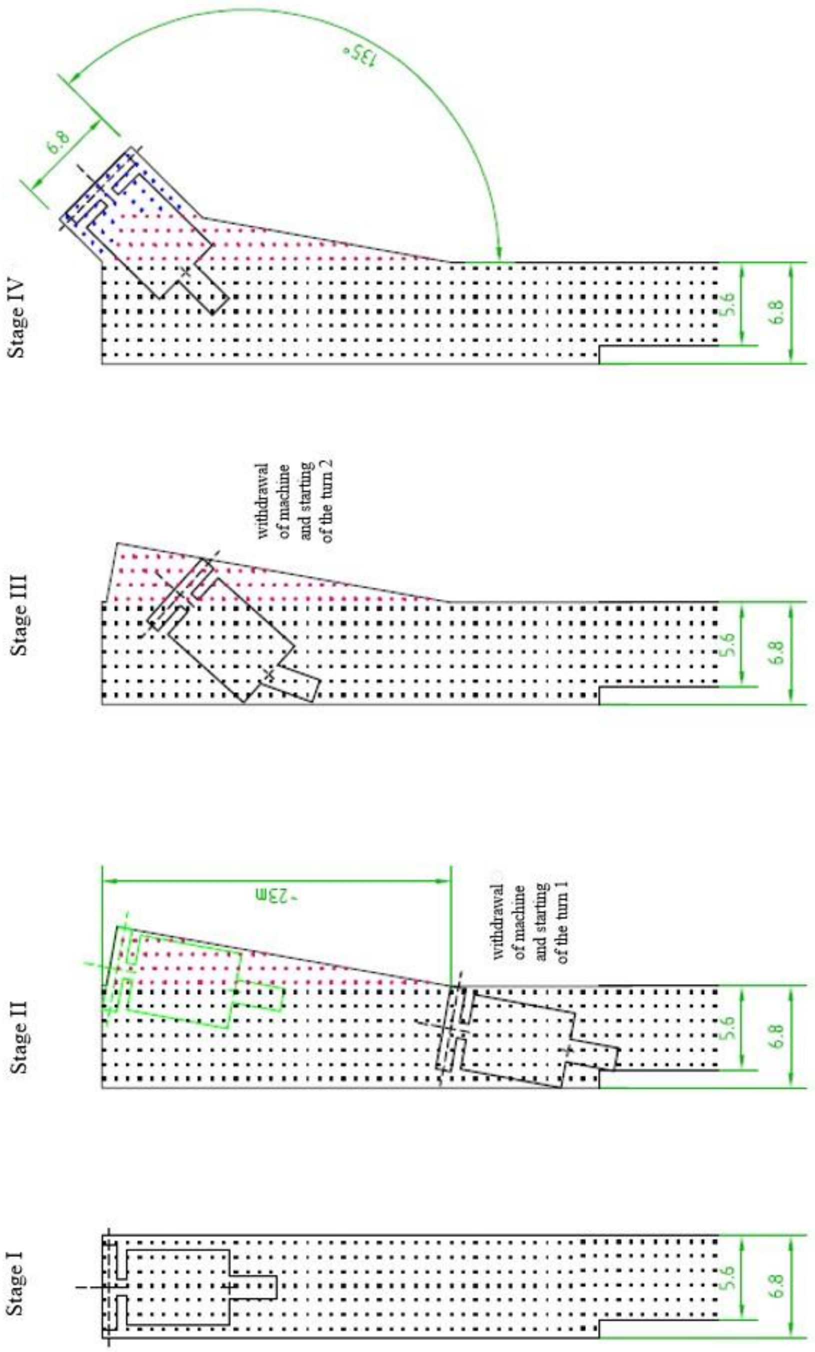

Fig. 13.17. Schematic diagram of driving the bend in the crossing No. 2 according to the Author`s conception (turn 135°, departure from the bend of 6.8 m width

13.7. Conclusions resulting from the analysis of cutting bends during the drivage of the Bw-1n test roadway use of the Bolter Miner machine

The conducted analysis leads to the conclusions given below:

I. Designing of a turn, bend or crossing in the case of the working, driven with a use of the machine of Bolter Miner type, should be started at a significant advance before beginning the operation.

II. All the factors, having an impact on the turn, bend or crossing feasibility such as: competences of the staff, technical capabilities of equipment and its overall dimensions, a possibility of protecting the working should be taken into consideration during the designing process.

III. A functionality of driven workings in the aspect of a further drivage, future functions and destination of the workings after the drivage termination should be taken into consideration during the designing process.

IV. The scheduled work programme should be prepared after completing the design project and the executors and supervisors should get acquainted with it.

V. Due to a necessity of maintaining the working and ensuring the expected functionality, the mining operations in the zone of driving a turn, bend or crossing should be performed at a special supervision.

VI. The Polish mining regulations are an essential issue because their statements do take into cosideration all the aspects of using the machine of Bolter Miner type, including a drivage of crossings. Taking advantage of world experience in this scope seems to be advisable.

Taking into consideration the conclusions, presented above, during a drivage of the following crossings or other changes in the direction of driven workings, enables to avoid mistakes and losses of time and it also has a positive impact on a reduction of widely understood risk.

14. Analysis of advance rates and efficiency assessment of conducted mining operations

Artur Dyczko[1], Jacek Korski[2], Marcin Mieszczak[3], Sylwia Pietras[2], Dariusz Prostański[4]

In Table 14.1 the scheduled work programme of the project of the Bw-1n test roadway drivage in independent roof bolting support is presented in a form of a block diagram. The project was divided into seven phases, including two phases comprising an assembly and disassembly of the test rig (in blue) and five phases of drivage. Five phases of the working drivage include three phases of driving straight sections (in green) and two phases of driving bends (in yellow).

Scheduled realization programme of the project of driving the Bw-1n test roadway in the independent roof bolting support [135]

Table 14.1.

Item	Type of individual stages of R+D work of the SOK Project	Unit	Number	Individual months of the SOK Project realization												
				1	2	3	4	5	6	7	8	9	10	11	12	13
1	R&D work, consisting in a creation of a test rig in the real environment and operational conditions, including an assembly of the Joy 12M30 machine, face devices and equipment for haulage of the run-of-mine together with an elaboration of reporting documentation	pcs.	1													

[1] Mineral and Energy Economy Research Institute of the Polish Academy of Sciences
[2] FAMUR SA
[3] Enterprise of Shafts Construction JSC
[4] KOMAG Institute of Mining Technology

2	R&D work, consisting in conducting development tests on the test rig in the real environment and operational conditions during a drivage of the Bw-1n test roadway to the first bend, according to the GIG design together with a preparation of the monthly report	m	200													
3	R&D work, consisting in making the bend No. 1 together with driving 70 m of the Bw-1n test roadway for an installation of the run-of-mine haulage devices according to the documentation of the SIGMA S.A., an installation of the run-of-mine haulage devices and a preparation of the monthly report	pcs.	1													
4	R&D work, consisting in conducting development tests at the test rig in the real environment and operational conditions, taking into consideration the conclusions from former stages at driving the second segment of the Bw-1n test roadway and a preparation monthly report	m	800													

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5	R&D work, consisting in specifying possibilities of the SOK technology at making the bend No. 2 together with driving 70 m of the Bw-1n test roadway for an installation of the run-of-mine haulage devices according to the SIGMA S.A. documentation, an installation of the run-of-mine haulage devices and a preparation of the monthly report	pcs.	1													
6	R&D work, consisting in conducting development tests at the test rig in the real environment and operational conditions with a demonstration of the final technology at driving the third segment of the Bw-1n test roadway and a preparation of the monthly report	m	830													
7	R&D work, consisting in an analysis of the current state of the SOK technology and in an elaboration of a methodology for an liquidation of the test rig in the real environment and operational conditions,	pcs.	1													

	including a disassembly of the JOY 12CM30 machine, of the face devices and of the SIGMA S.A. run-of-mine haulage equipment, together with a preparation of the final report on the SOK Project															

14.1. Analysis of operations advance

I month of the project – October/November

The first month of the project ”Independent Roof Bolting Support” realization started on 10th October 2019 from an underground visit and selecting an assembly spot of the JOY 12CM30 machine. The machine assembly began from an assessment of the supply completeness of the machine components, their dimensions and weights. The components were located in the assembly chamber prepared earlier in Seam 401 of the Budryk mine. The machine was assembled, an assembly correctness and operational conformity of all the machine sub-assemblies after their assembly [135] were checked.

II month of the project – November/December

In the second month of the project realization a drivage of the Bw-1n test roadway in Seam 401 of the Budryk Mine was started. During the settling period 65.8 m of the working were driven. The main reasons of down-times were mechanical failures of the machine and belt conveyors, also technological activities, e.g. a construction of a periodical control station. The daily advance of the working in the second month of the project is presented in Fig. 14.1 [135].

III month of the project – December /January

The third month of the project realization was ended at 200 rm of the Bw-1n test roadway. The operations connected with the working drivage, realized in this period, sped up significantly in relation to the face advance during the previous month. Down-times during operations were mainly connected with technological reasons (levelling of the monorail route), mechanical reasons (machine failures) and other incidents, e.g. lack of water or lack of personnel transport to the face. The daily advance of the working in the third project month is presented in Fig. 14.2 [135].

IV month of the project – January /February

The fourth realization month of the project ”Independent Roof Bolting Support” included an execution of a bend, which increased the labour significantly, among others at a reconstruction of the technical infrastructure in the working and at an installation of a bigger number of bolts (including rope bolts). The fourth month was ended at 255.2 rm of the Bw-1n test roadway. To make the bend at 210 rm of the working, the machine was withdrawn and an installation of the chock support in the face front and at the left wall was started. After having been withdrawn, the machine was cut-in the right wall, dismantling the installed side lacing there and cutting composite bolts. After having made the bend on 21st January 2020, the mine surveying services set out the direction of driving and a new starting point from which measuring of an advance was started again. The working of the dimensions 6.8 m x 4.2 m was driven until 5th February, when the basic dimensions 5.6 m x 4.2 m were returned, which enabled to increase a daily advance. In the period under discussion, breaks in driving were only connected with technological reasons i.e. a withdrawal of the machine, an installation of mixed support or a re-installation of the belt conveyor at the bend. The daily advance of the working in the project fourth month is presented in Fig. 14.3 [135].

V month of the project – February /March

The fifth month of the project realization ended at 392 rm of the Bw-1n test roadway. Down – times during the working drivage, resulting in a small daily advance (smaller than an average advance of faces at the JSW – Jastrzębska Spółka Węglowa – Jastrzębska Coal Company), were mainly caused by mechanical failures, i.e. lack of control of the Pioma conveyor, a damage of the bolter ram, a failure of water sensor on the machine cutting head, a failure of the main haulage system at the Budryk mine, a failure of the Pioma conveyor. Besides, some activities connected with earthing the housing and bolting the roof were conducted. Two failures of methanometric sensors occurred as well. A daily advance of the working in the fifth month of the project is presented in Fig. 14.4 [135].

VI month of the project – March/April

The sixth month of the project realization ended at 556.4 rm of the Bw-1n test roadway. Since 23rd March 2020 due to an introduction of the epidemic state in Poland, an organization of work was changed. The worktime was reduced to 3 working shifts every 24 hours, each lasting 6 hours. Until an introduction of restrictions daily advances were bigger than an average advance of face activities at the JSW. After having introduced the epidemic state and after having changed the work organization, the daily advances dropped below the average. Down-times were caused by failures of the main haulage system, a failure of the machine roof drill-rig, a failure of the machine cutting head, a failure of the water flow in the machine and also lack of water and pressure jumps in the fire pipeline. The daily advance of the working in the sixth month of the project is presented in Fig. 14.5 [135].

VII month of the project – April/May

The seventh month of the project realization ended at 689.2 rm of the Bw-1n test roadway. During the first days of the seventh month the restrictions, caused by the epidemic state, were still obligatory. However, on 20th April they were annulled and four-shift work time was returned. A continuity of driving the working, despite restrictions, faced no hazard. A daily advance in the seventh month was bigger than an average of the faces advance for the JSW. The down-times were caused by mechanical failures (control of the machine), a maintenance of machines and an extension of the run-of-mine haulage route. Additionally low pressure of water and its lack were noted once and a permissible methane concentration was exceeded once. Besides, the machine sank in weak floor rocks. A daily advance of the working in the seventh month of the project duration is presented in Fig. 14.6 [135].

VIII month of the project – May/June

The eighth month of the project realization ended at 903.2 rm of the Bw-1n test roadway. The working drivage was conducted in a continuous way without any longer down-time. The daily face advance was more than average daily advance of faces at the JSW. In the eighth month the average daily advance of the Bw-1n test roadway was 9.70 m and for the other faces driven conventionally – 5.14 m. It should also be taken into consideration, that the roadway was conducted in the concentrated bolting network, so in the case of better roof conditions the daily advance could be even bigger. Down-times were mainly caused by mechanical failures, i.e. of the main haulage system, of the SIGMA conveyor, of the machine control or an electrical failure – a voltage decay in the grid. The methane concentration in the working was exceeded twice, once a down-time was caused by lack of water in the fire pipeline. The daily working advance in the eighth month is presented in Fig. 14.7 [135].

IX month of the project – June/July

The ninth month of the project realization ended at 1015.4 rm of the Bw-1n test roadway. The driving advance and a realization advance of the project were very limited in the ninth month. Due to big concentrations of the COVID-19 cases in Silesia, an operation of mines, including the Budryk mine, was restricted. The working drivage was continued in June during single shifts a day. Normal operation started again at the beginning of July. A smaller number of workers and conducting operations in the sanitary system caused a significant decrease of the average daily advance of the working: in June – 6.22 m (for the whole JSW 5.91 m), in the accounting period – 5.13 m. Small advance rates were mainly caused

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by a reduction of the work-time due to the epidemic. Additional down-times, slowing down the operations did not often happen. Some mechanical failures were recorded, i.e. loading of the main haulage system, breaking the chain on the feeder and lack of main haulage. Initially, in the ninth month of the project realization it was planned to make the second bend, but the work connected with this task was not done due to delays, caused by the epidemic. The daily advance of the working in the project ninth month is presented in Fig. 14.8 [135].

X month of the project – July/August

The tenth month of the project realization ended at 1059.4 rm of the Bw-1n test roadway. The working drivage advance in the tenth month was very limited. A limitation of operations resulted from a sudden change of mining-and-geological conditions, mainly due to an increase of the Seam 401 inclination angle to about 15^{o}, which made further working drivage impossible. In the result of that, the machine was withdrawn to a new location of the second bend of the working, the floor underworking was done and the infrastructure was reconstructed. Additionally, it was indispensable to elaborate an appendix to the Operational Plan of the Mining Plant. Since 27th July 2020 a five-shift system was introduced. It also caused a decrease of the advance rate. An average daily advance in July was only 3.49 m (in the case of the other workings in the JSW S.A., driven with use of traditional technology – 5.09 m). An average daily advance in the project accounting period was 2.01 m. A decrease of the daily advance rate in the Bw-1n test roadway in the tenth month of the project realization did not result from other serious down-times. The daily advance of the working in the tenth month is presented in Fig. 14.9 [135].

In Fig. 14.10 a growing advance rate of the Bw-1n test roadway drivage during 10 months of the ”Independent Roof Bolting Support” project realization is shown.

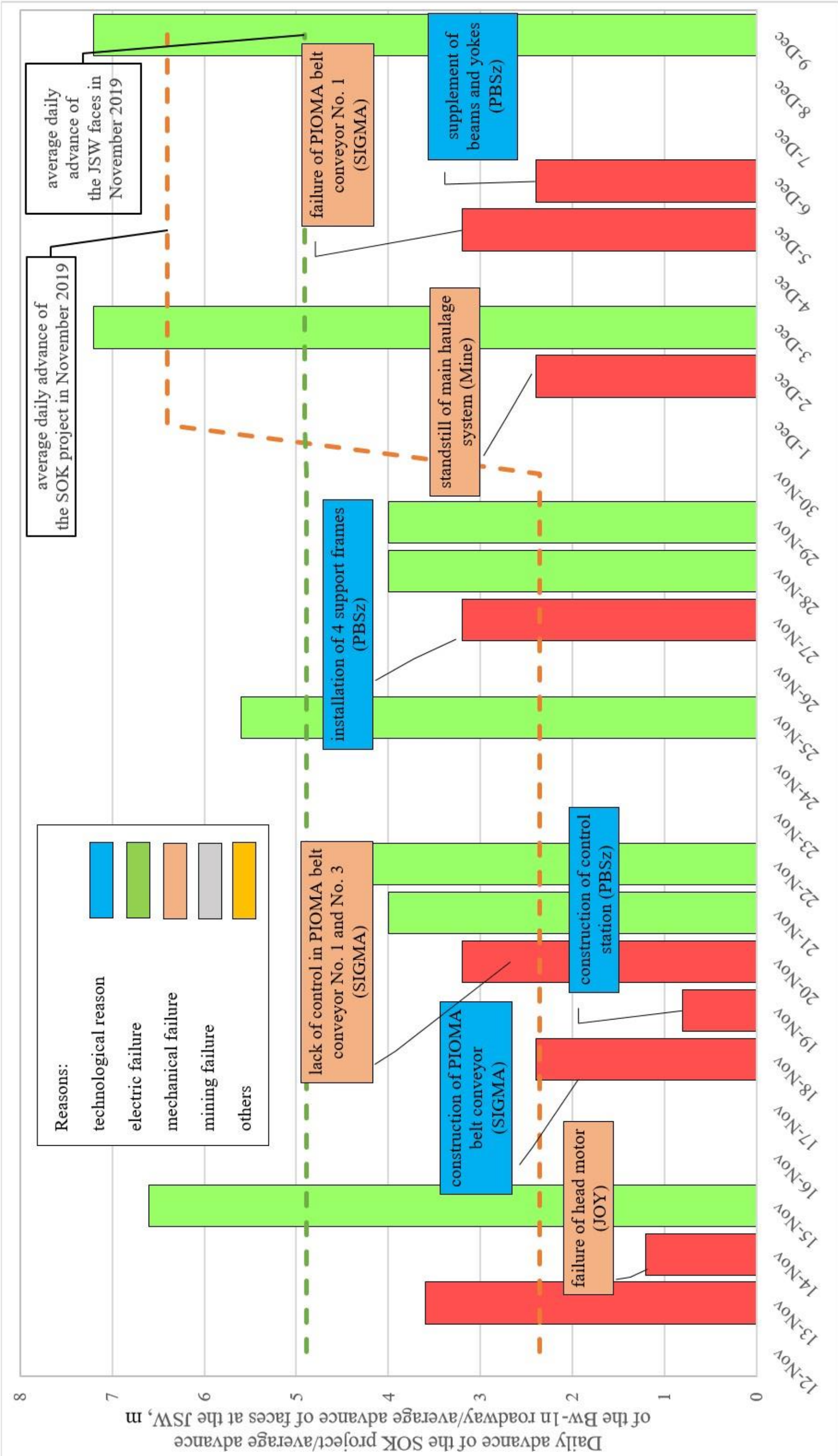

Fig. 14.1. Advance of operations in the second month of the “ Independent Roof Bolting Support” project realization [135]

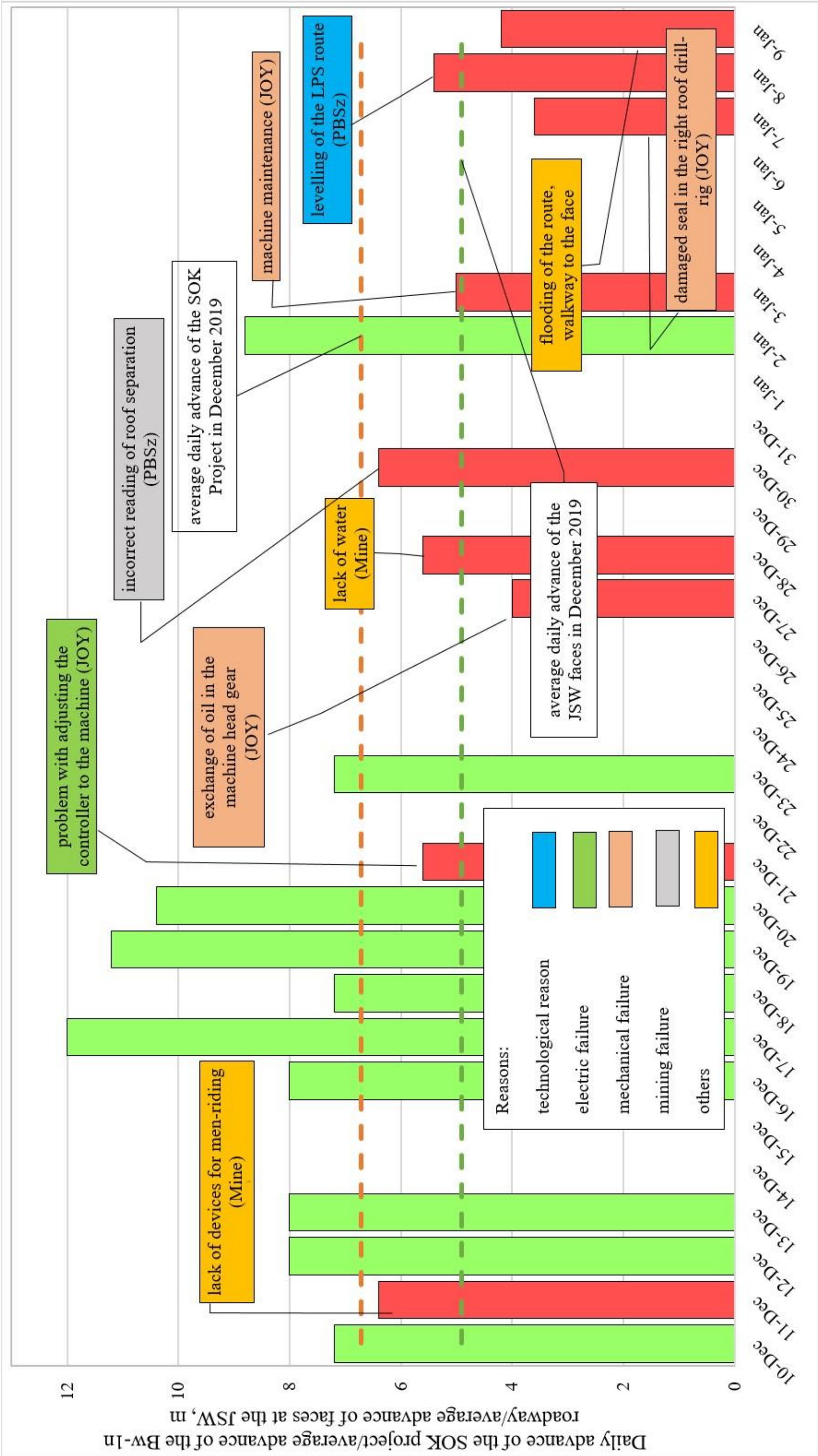

Fig. 14.2. Advance of operations in the third month of the “ Independent Roof Bolting Support” project realization [135]

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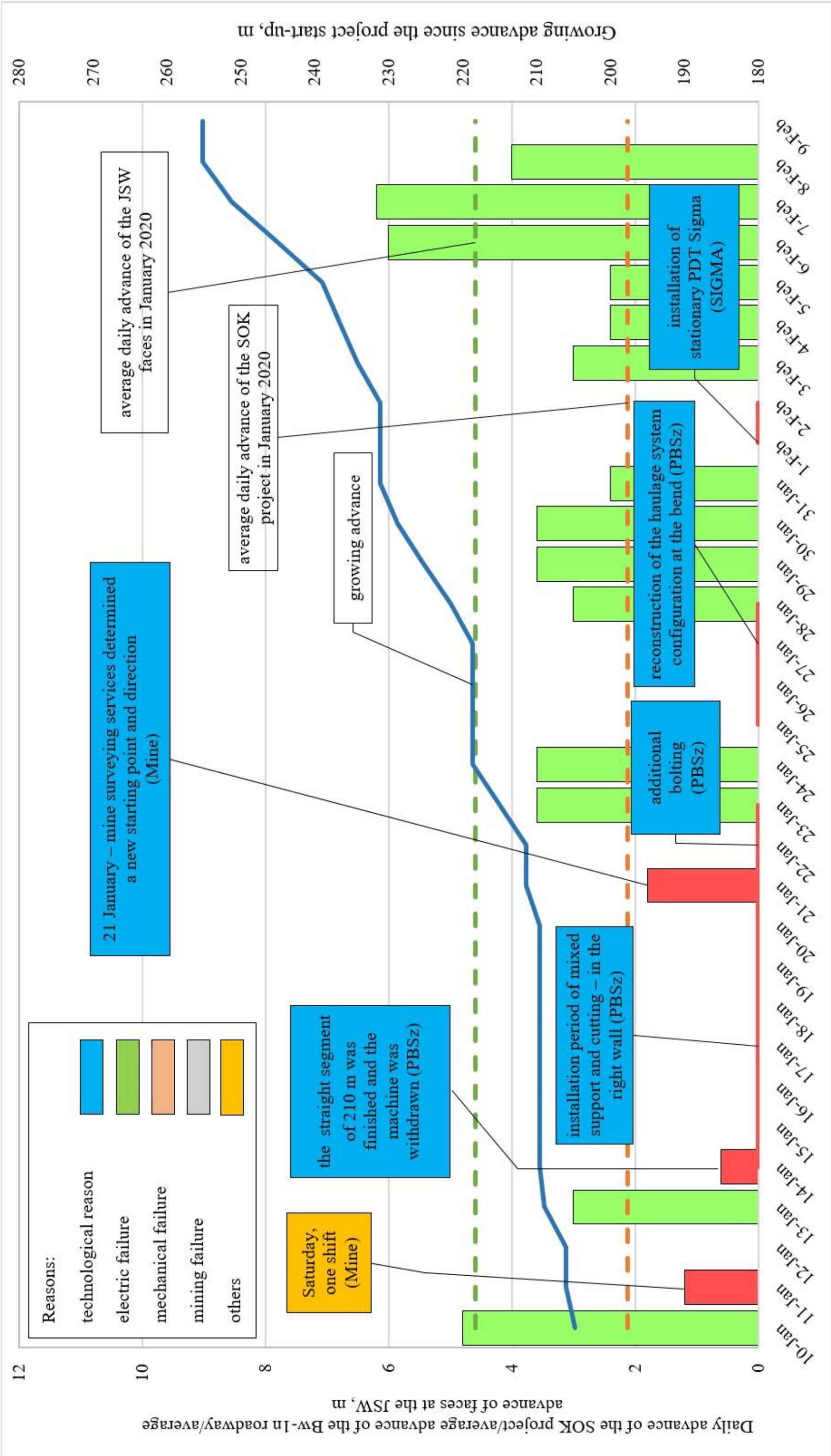

Fig. 14.3. Advance of operations in the fourth month of the " Independent Roof Bolting Support" project realization [135]

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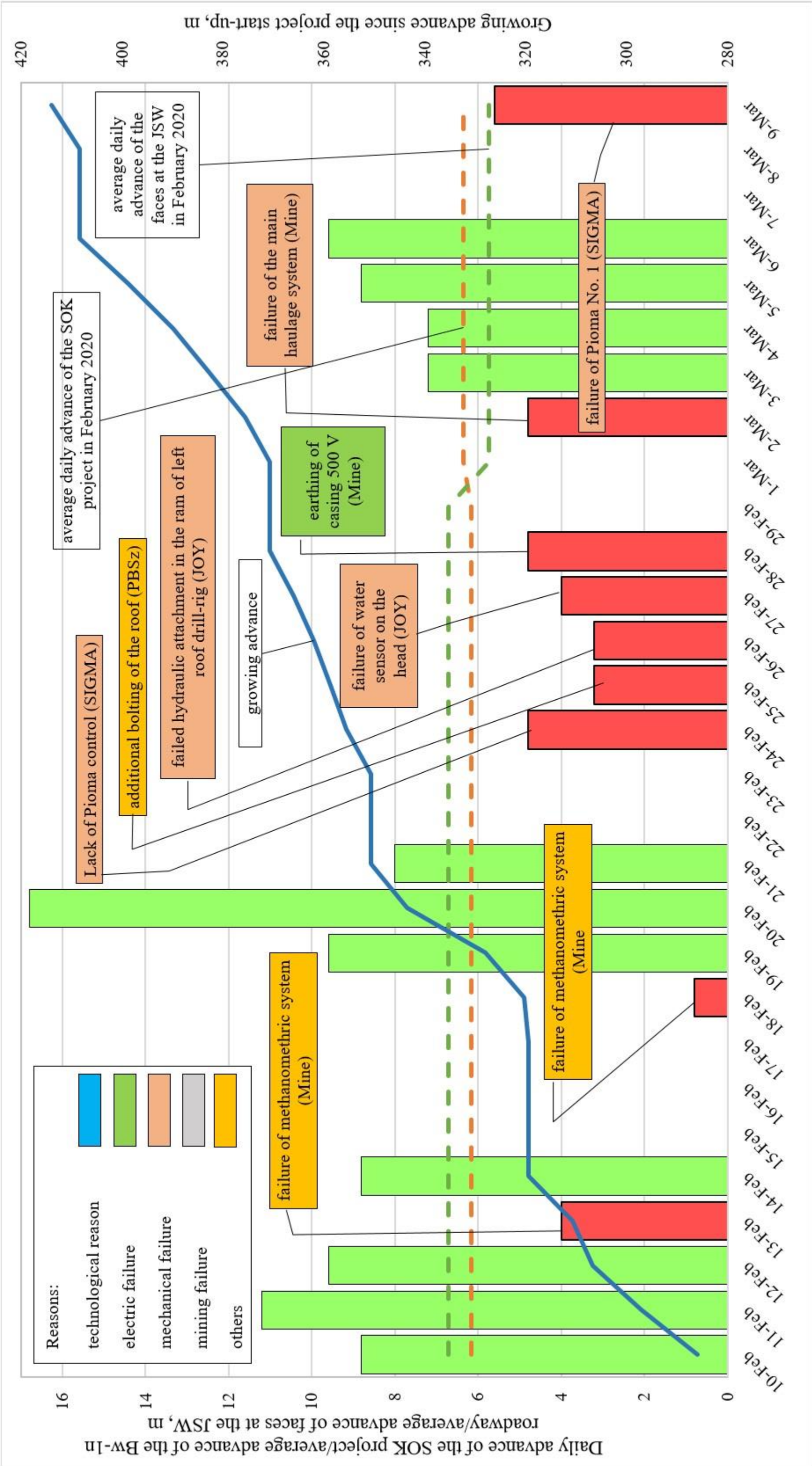

Fig. 14.4. Advance of operations in the fifth month of the " Independent Roof Bolting Support" project realization [135]

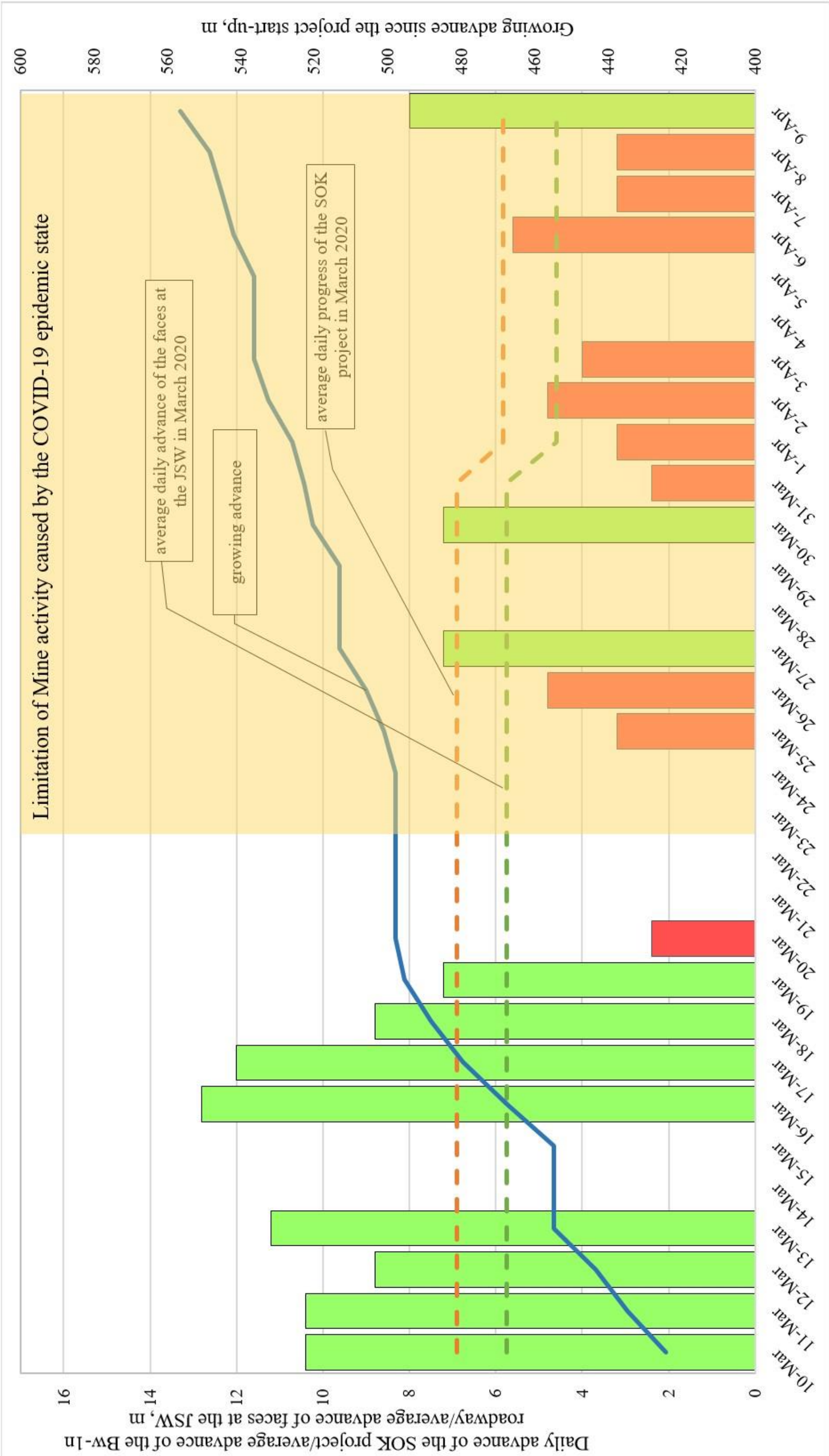

Fig. 14.5. Advance of operations in the sixth month of the “ Independent Roof Bolting Support” project realization [135]

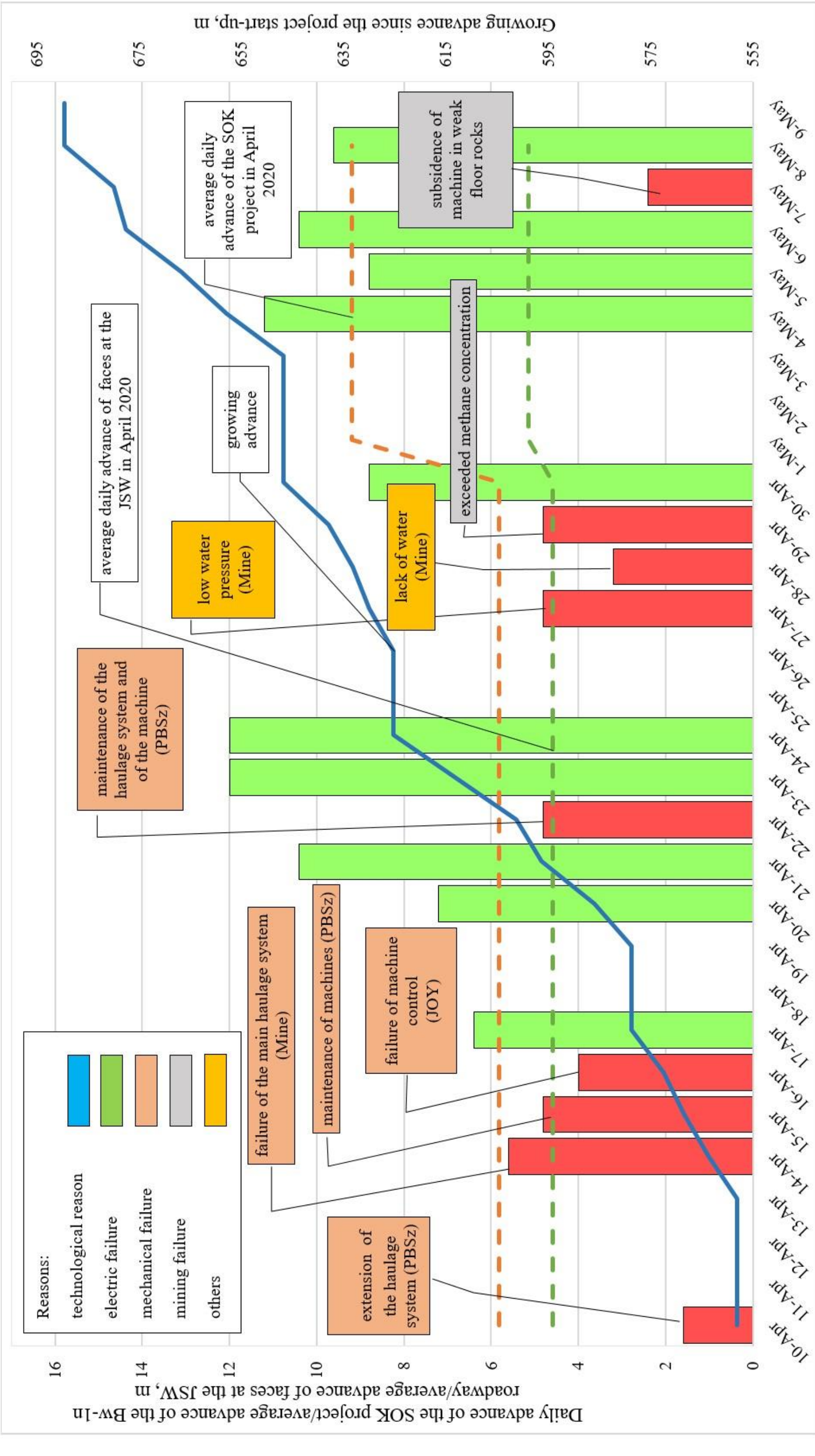

Fig. 14.6. Advance of operations in the seventh month of the " Independent Roof Bolting Support" project realization [135]

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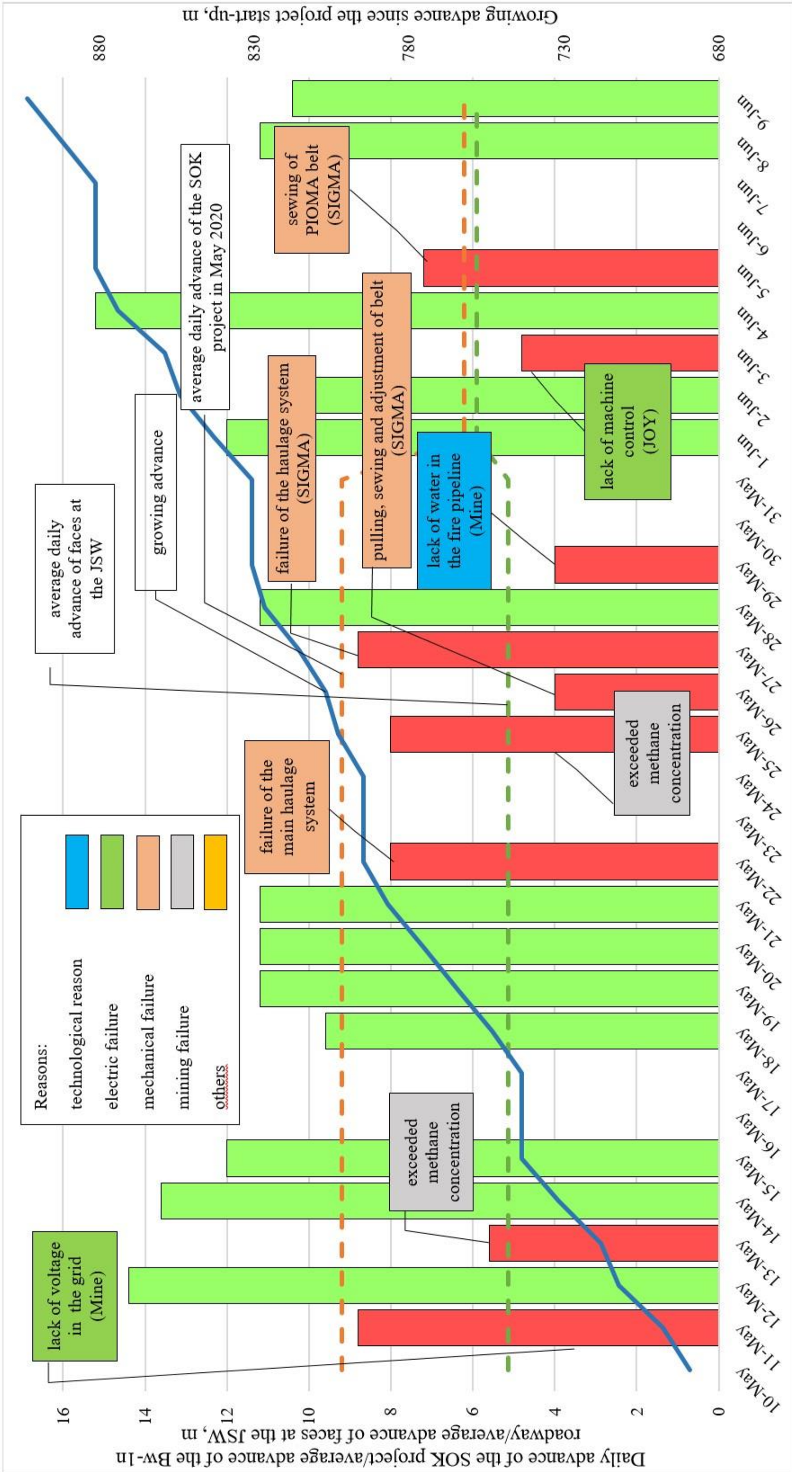

Fig. 14.7. Advance of operations in the eighth month of the “ Independent Roof Bolting Support” project realization [135]

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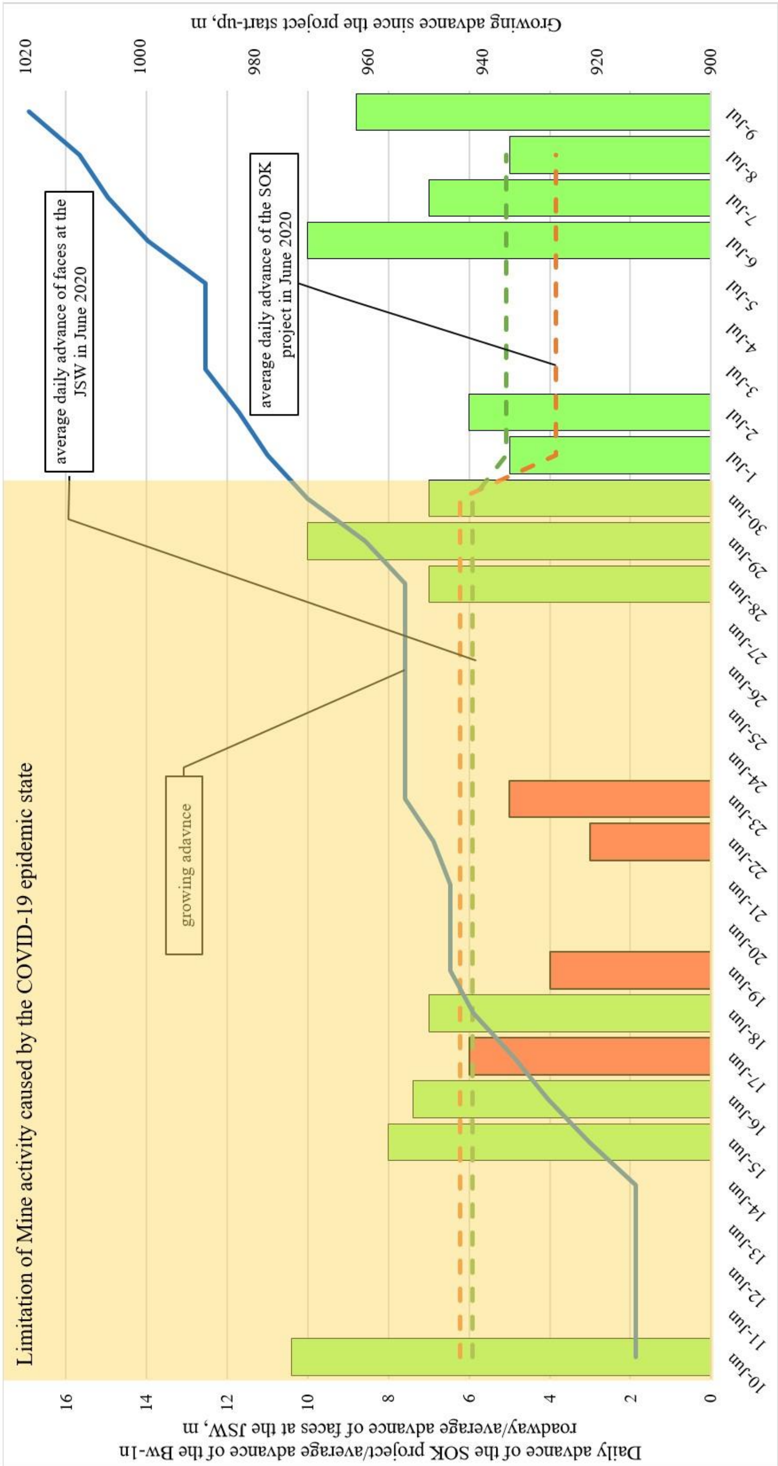

Fig. 14.8. Advance of operations in the ninth month of the " Independent Roof Bolting Support" project realization [135]

 https://doi.org/10.32056/KOMAG/Monograph2021.2

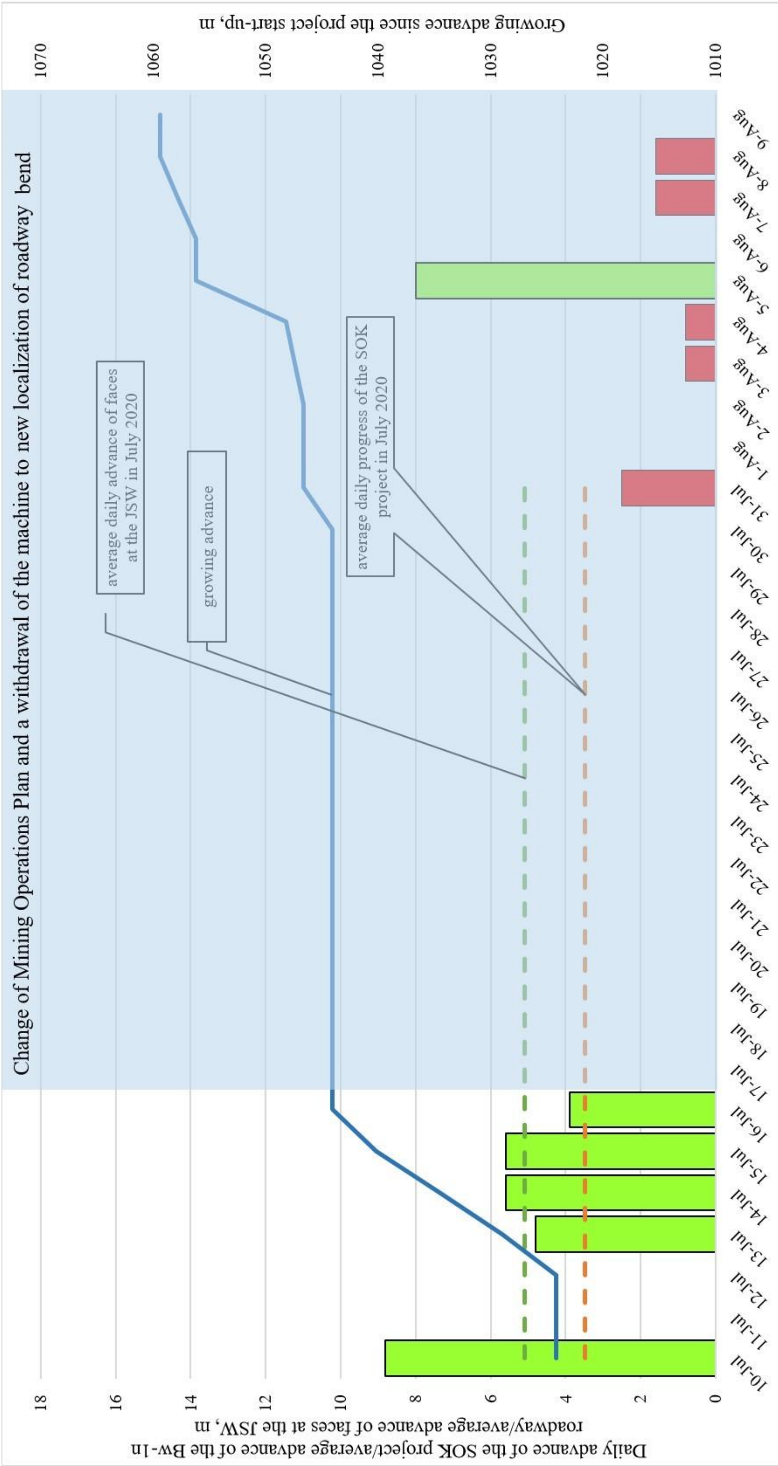

Fig. 14.9. Advance of operations in the tenth month of the “ Independent Roof Bolting Support” project realization [135]

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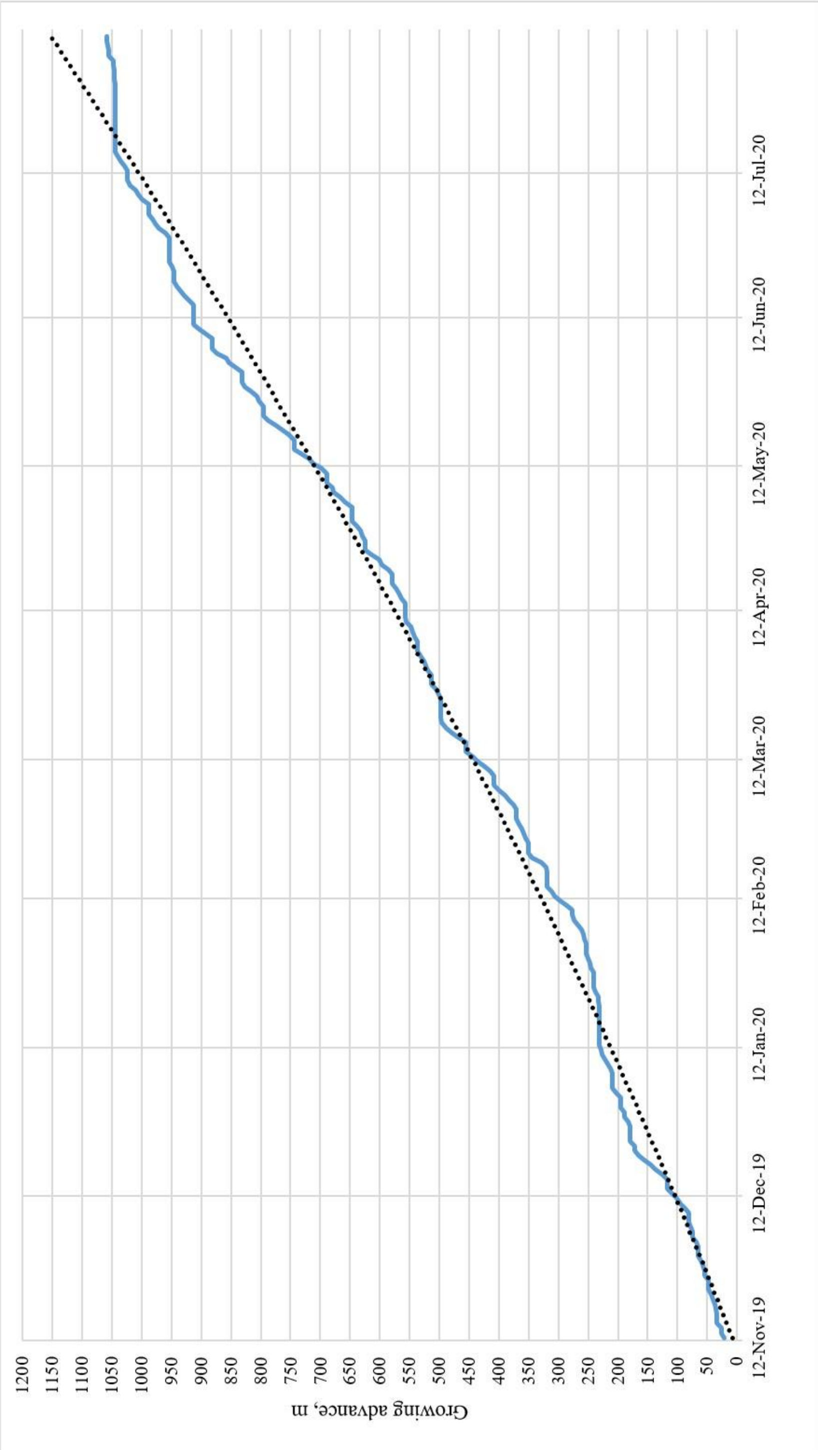

Fig. 14.10. Cumulative advance of the Bw-1n test roadway drivage during 10 months of the project realization [135]

14.2. Analysis of the work organization

The activities in the Bw-1n test roadway were conducted by the employees of the Przedsiębiorstwo Budowy Szybów (PBSz) S.A. (Enterprise of Shaft Construction). A new division was established for a realization of the project and 26 new workers were employed. Altogether 113 people are employed in the division, 100 people on working positions and 13 people of the supervising personnel. In the PBSz S.A. 18 experienced workers (including 11 miners in the face) were employed. For 3.5 years they have gained experience in driving workings in independent roof bolting support in Czech hard coal mines. These workers had to be trained in the scope of conveyor haulage system because in the Czech mines haulage vehicles of *Shuttle Car* type were used [128].

At present the workers, experienced in operation of the *Bolter Miner machine* and independent roof bolting support, in Czech mines, are the essential part of the team, having to a large extent, a decisive impact on the efficiency of conducted operations. Their possible leaving might result in serious difficulties connected with the efficient working drivage [128].

In Fig. 14.11 an average face employment rate at a shift in each month of the Bw-1n test roadway drivage is presented.

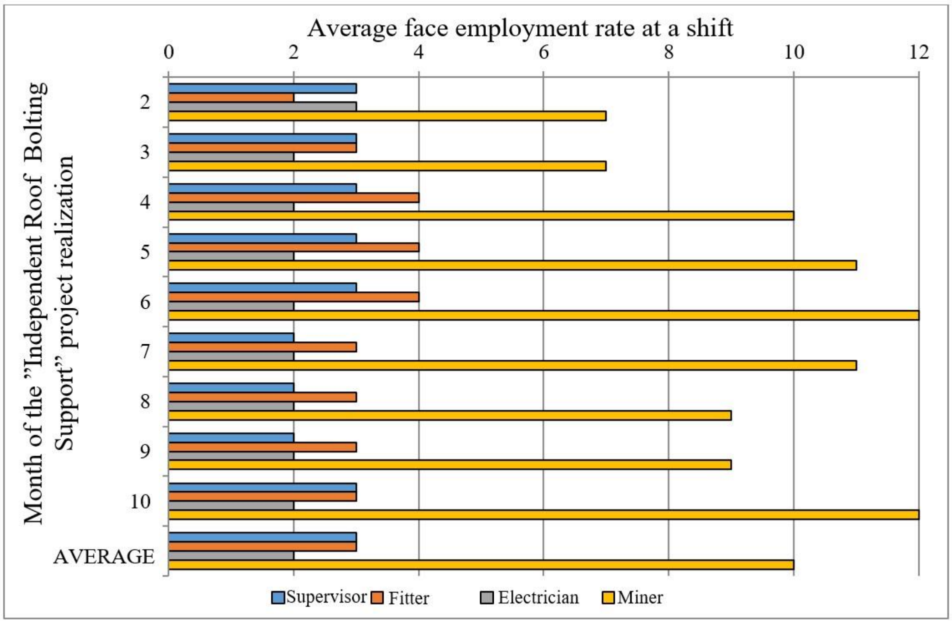

Fig. 14.11. Average face employment rate until the 10th month of the project realization [135]

The COVID-19 pandemic had a significant impact on the division work organization [128]:

– during 23rd March – 17th April a three-shift work organization as regards the employment rate; besides in the result of reducing an employment rate at the Budryk mine, the employment rate of each of the three shifts had to be reduced,

– during 14th June - 29th June, after having conducted tests for the presence of the SARS-CoV-2 virus, 33 members of the staff were given lay days connected with catching COVID-19 or having a lockdown,

– during 24th June – 29th June the employment was only at one maintenance shift and the rest of the personnel was directed to have swabs taken.

Since the beginning of the epidemic, the division has experienced problems resulting from an increased absence of workers which caused that the daily employment rate was limited even to 20 people. Besides, the epidemic had a negative impact on functioning of the Budryk mine itself, causing difficulties with the material logistics and a transportation of the personnel to the face [128].

An employment rate of people on working positions, over the period from January till July 2020, is shown in Fig. 14.12.

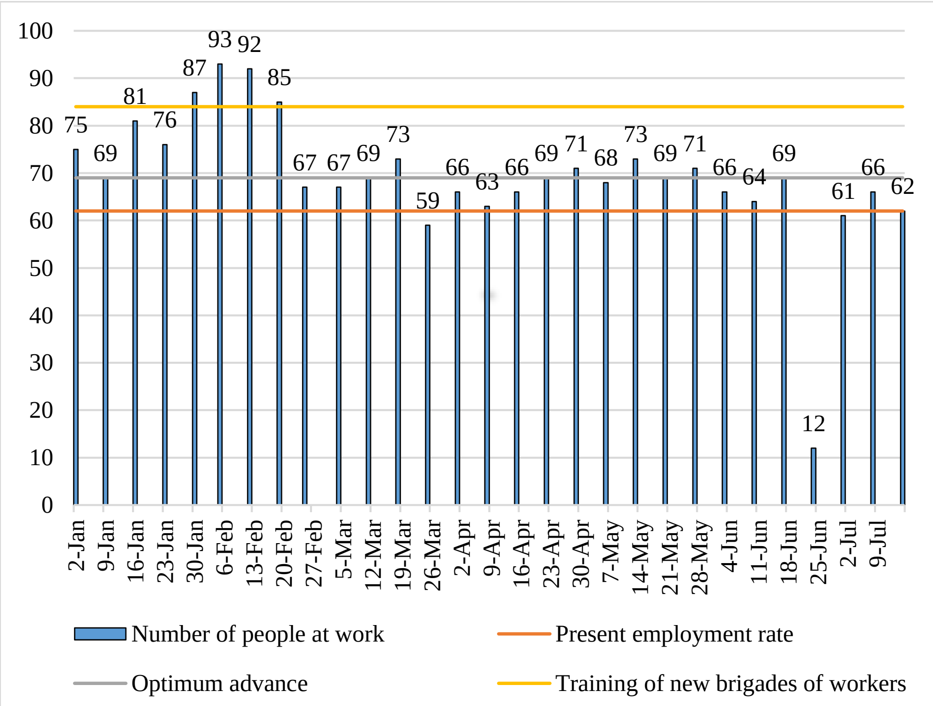

Fig. 14.12. Average weekly employment rates of personnel on working positions together with the in-coming employment rate [128]

An assessment of the required number of workers indispensable for an optimum advance and for a face organization (Variant 2) was carried out. A training of 4 new gangs of workers annually was assumed. They will be capable of carrying out efficient driving of workings with use of Bolter Miner machine and with use of independent roof bolting support (Variant 3). The analysis results are listed in Table 14.2.

Analysis of optimum employment rate in the division [128]

Table 14.2.

Type of shift	Type of position	Variant 1	Variant 2	Variant 3
		Present state	**Optimum advance**	**Training**
		Daily employment rate		
Production shift	Miner in the face	5	6	8
	Miner on haulage devices	4	4	4
	Transport worker	2	2	2
	Timberer	0	1	0
	Wireman	1	1	2
	Fitter	2	1	2
	SUM	14	15	18
Maintenance shift	Miner for advance preparation	9	10	12
	Fitter	6	8	10
	Wireman	5	6	8
	SUM	20	24	30
TOTAL (3 production shifts + 1 maintenance shift)		**62**	**69**	**84**

14.3. Analysis of down-times

A list of down-times in an operation of the JOY 12CM30 machine, based on monthly reports on research-and-development activities, is presented in Table 14.3.

A list of down-times during the Bw-1n roadway drivage [135]

Table 14.3.

Month of project	PBSz		Budryk Mine		JOY		SIGMA		In total	
	Number	**Time [min]**	**Number**	**Time [min]**	**Number**	**Time [min]**	**Number**	**Time [min]**	**Number**	**Time [min]**
2	10	1875	3	600	0	0	0	0	13	2475
3	10	1040	26	3095	0	0	0	0	36	4135
4	23	2515	0	0	0	0	0	0	23	2515

5	9	735	9	1390	0	0	0	0	18	2125
6	2	250	16	1870	6	755	0	0	24	2875
7	7	1035	27	1988	11	444	2	540	47	4007
8	18	1545	68	6720	8	795	0	0	94	9060
9	25	1905	25	2500	13	1380	3	110	66	5895
10	23	1485	28	5844	15	1945	0	0	66	9274
SUM	**127**	**12385**	**202**	**24007**	**53**	**5319**	**5**	**650**	**387**	**42361**
		206 h		**400 h**		**89 h**		**11 h**		**706 h**

The data, concerning failures and down-times over the period of nine months of driving the Bw-1n test roadway (months 2-10 of the project ”Independent Roof Bolting Support”), were analyzed. The total number of failures and down-times was 387, which caused a loss of time of 706 hours. These failures were caused by the following subjects:

– Przedsiębiorstwo Budowy Szybów (PBSz) S.A., responsible for 127 down-times in total, causing 206 hours of lost time, which gave 29% of the total time of down-times. The down-times, caused by the PBSz S.A., resulted from the failures of conveyors of local haulage, problems with the machine control system and also other mechanical and electric failures of the JOY 12CM30 machine.

– Budryk mine responsible for 202 down-times in total, causing 400 hours of lost time, which gave 57% of the total time of down-times. The down-times, caused by the Budryk mine, resulted from water pressure failures in the fire pipeline, failures of conveyors in the main haulage system and of the suspended monorail and also a technological down-time caused by a deterioration of geological conditions in the area of 1000 rm of the working.

– JOY Global Poland Ltd. responsible for 53 down-times in total, causing 89 hours of lost time, which gave 13% of the total time of down-times. The down-times, caused by the JOY Global, mainly resulted from failures of the hydraulic system and also from the problems with the JOY 12CM30 machine control system.

– SIGMA S.A., responsible for 5 down-times in total, causing 11 hours of lost time, which gave 1% of the total time of down-times. The down-times, caused by the SIGMA S.A. resulted from failures of the SIGMA belt conveyors.

In Fig. 14.13-14.21 time of down-times during the Bw-1n test roadway drivage with an indication of the subject responsible for the down-time and a number of down-times in the face each month of the working drivage are presented.

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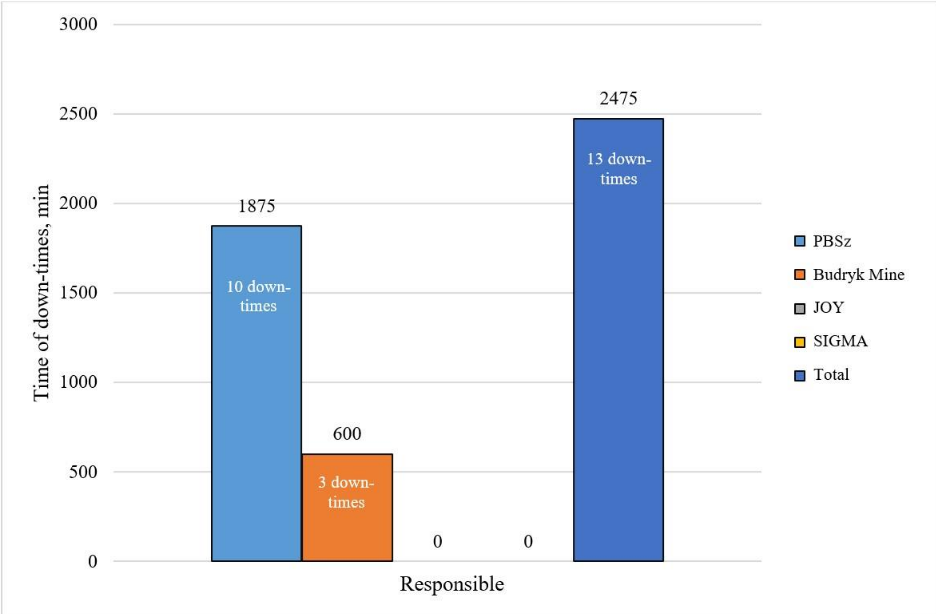

Fig. 14.13. Time of down-times in operation during the second month of the project "Independent Roof Bolting Support"

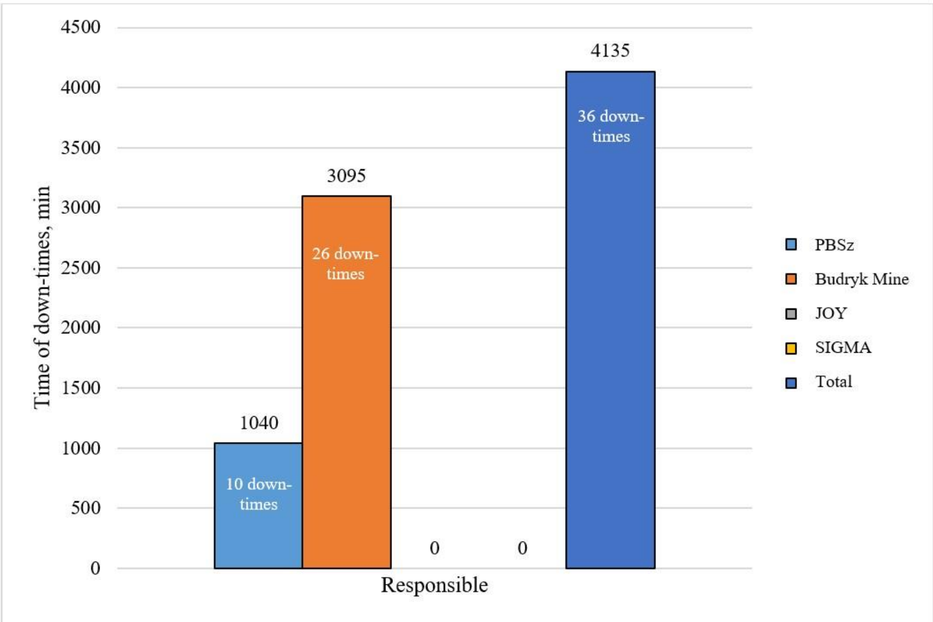

Fig. 14.14. Time of down-times in operation during the third month of the project "Independent Roof Bolting Support"

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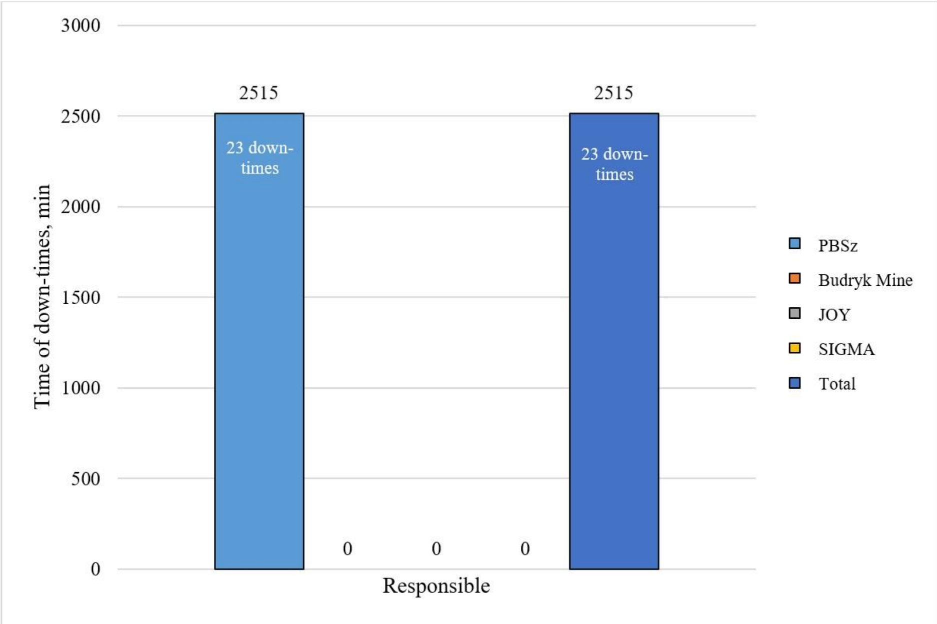

Fig. 14.15. Time of down-times in operation during the fourth month of the project "Independent Roof Bolting Support"

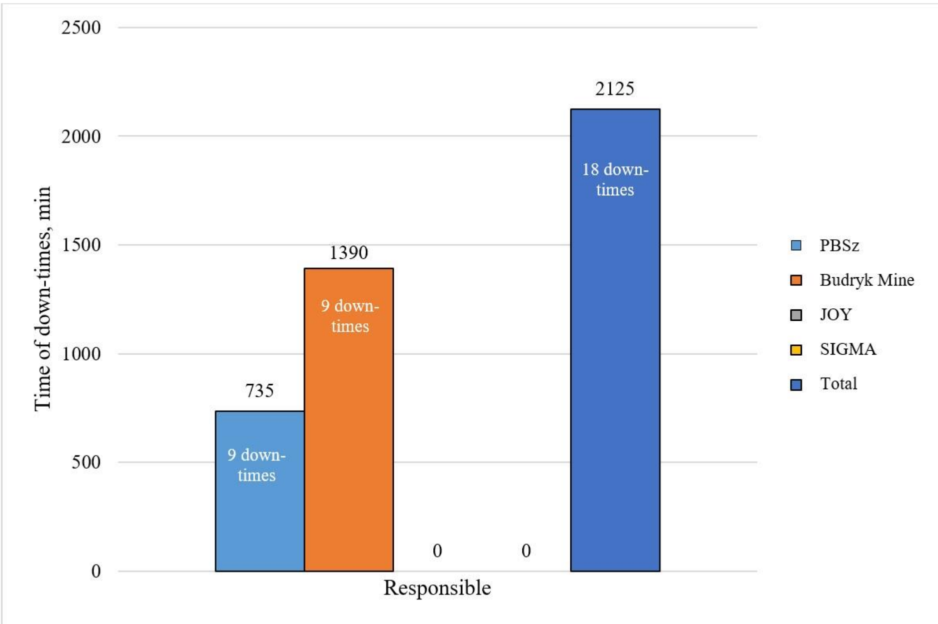

Fig. 14.16. Time of down-times in operation during the fifth month of the project "Independent Roof Bolting Support"

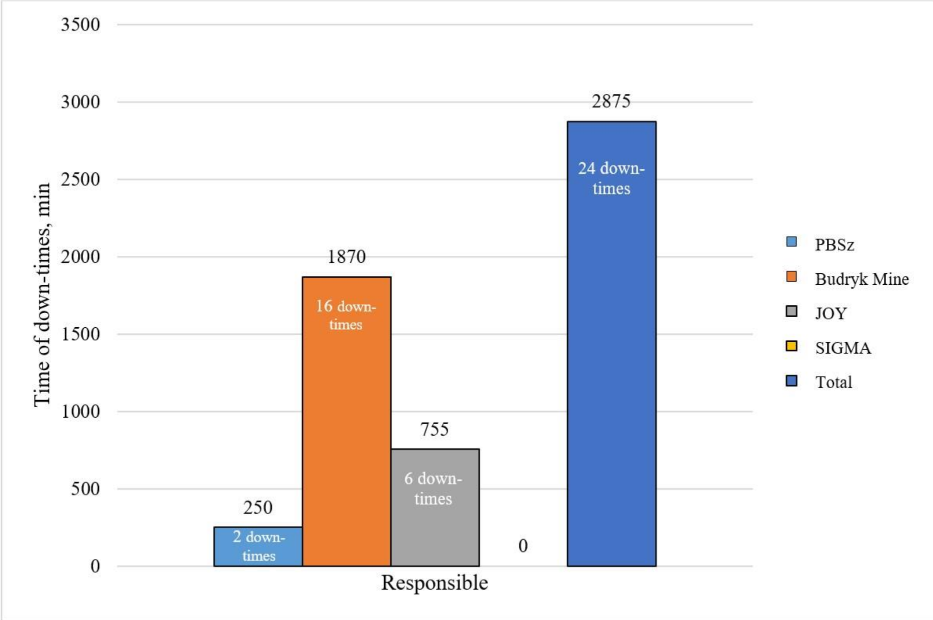

Fig. 14.17. Time of down-times in operation during the sixth month of the project "Independent Roof Bolting Support"

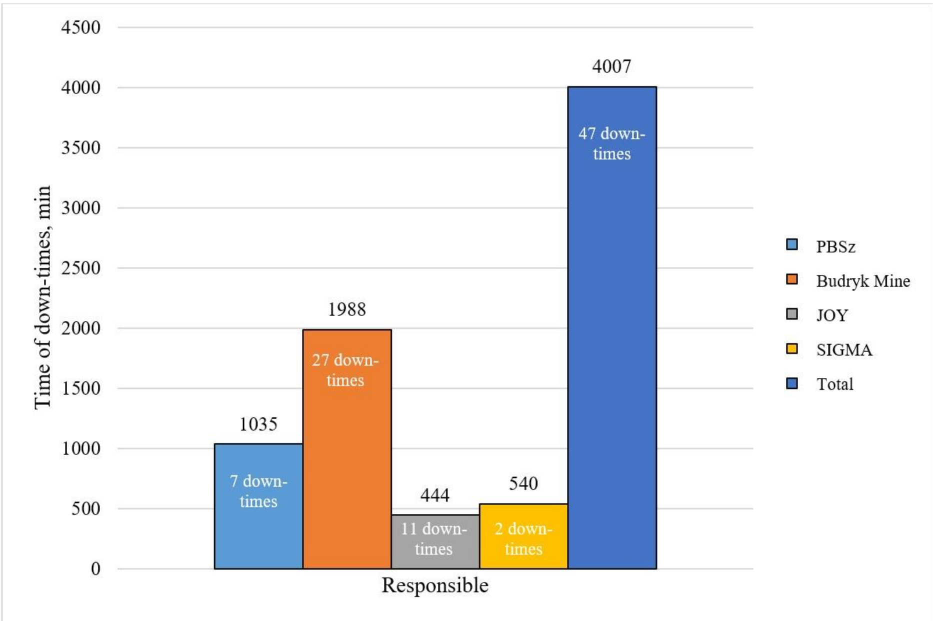

Fig. 14.18. Time of down-times in operation during the seventh month of the project "Independent Roof Bolting Support"

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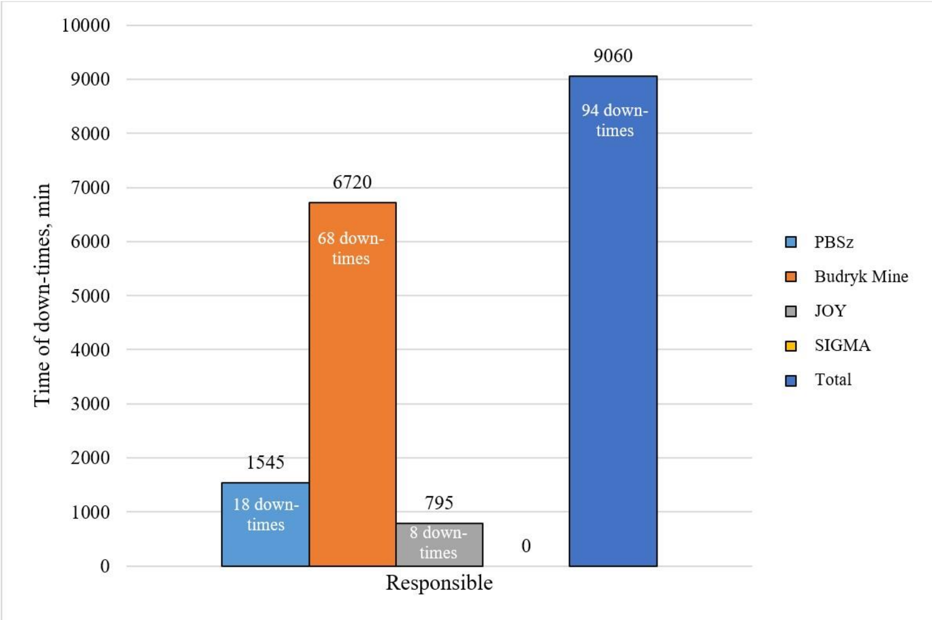

Fig. 14.19. Time of down-times in operation during the eighth month of the project "Independent Roof Bolting Support"

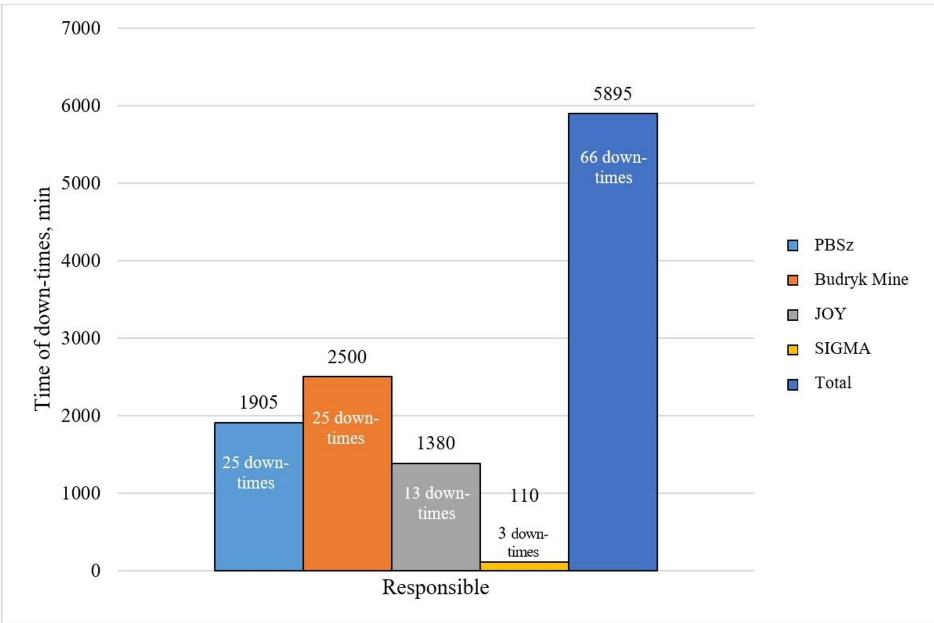

Fig. 14.20. Time of down-times in operation during the ninth month of the project "Independent Roof Bolting Support"

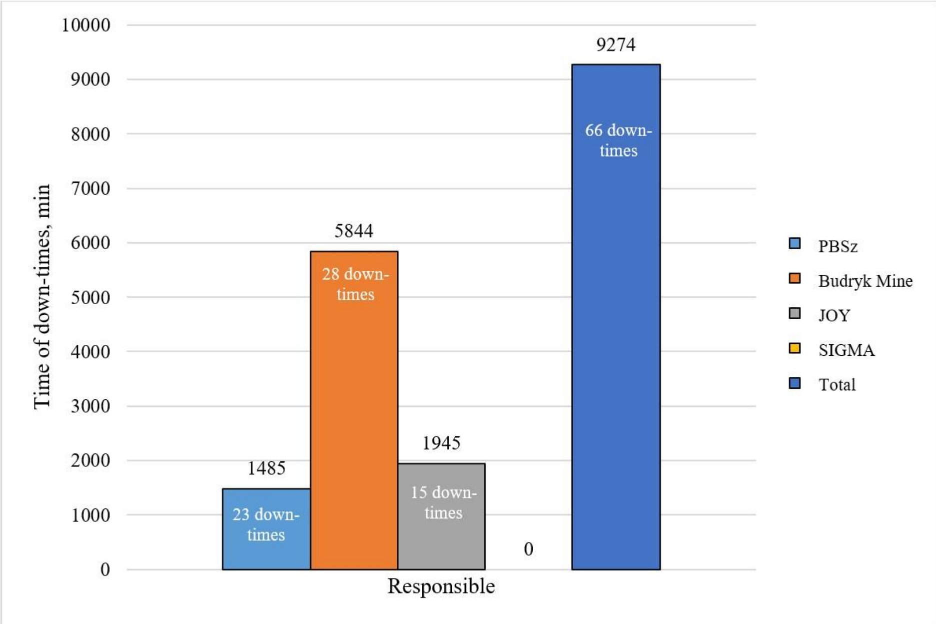

Fig. 14.21. Time of down-times in operation during the tenth month of the project "Independent Roof Bolting Support"

In Fig. 14.22 the total number of down-times during the period of ten months of the project "Independent Roof Bolting Support" and the total time of all the down-times with a specification of the responsible subject are presented.

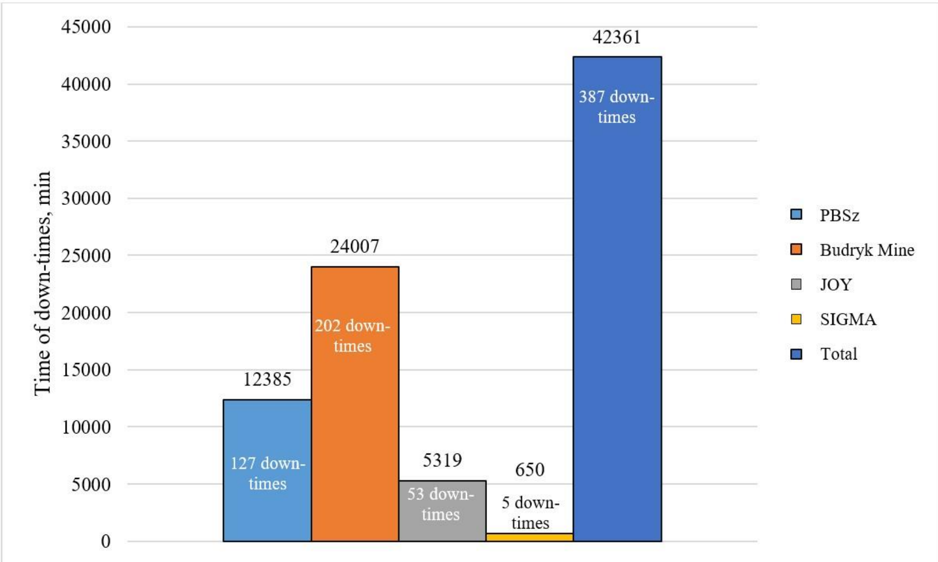

Fig. 14.22. Total time of down-times during 10 months of the "Independent Roof Bolting Support" project realization

In Fig. 14.23 and 14.24 a percentage share of down-times in the driving of the Bw-1n test roadway, in the total number in relation to the responsible subject, is presented.

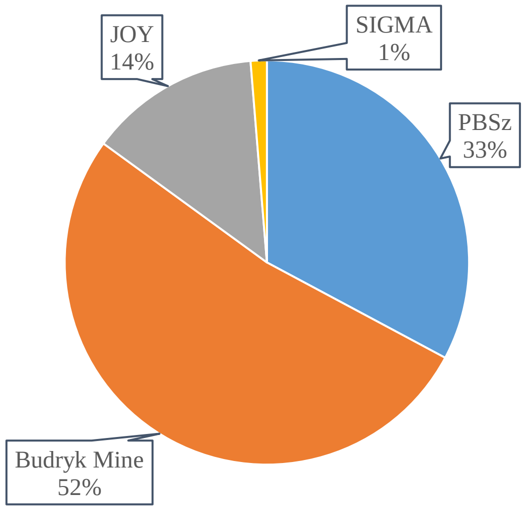

Fig. 14.23. Percentage share in the number of down-times according to the responsible subject

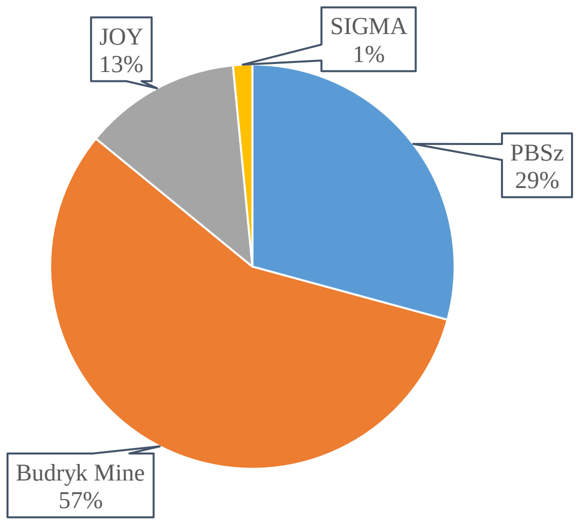

Fig. 14.24. Percentage share in the time of down-times according to the responsible subject

In Fig. 14.25 the time of down-times in the period of 10 months of the project realization with a specification of the reason is presented. Besides, the number of down-times, caused by a given reason, is included in the graph. The reasons of down-times are qualified to one out of nine categories:

– down-time of the local haulage conveyor,
– down-time of the machine,
– down-time of the main haulage system,
– down-time of the suspended monorail,
– a technological reason,
– a ventilation reason,
– a lack of water,
– a failure of electric grid,
– another reason.

Failures and other down-times of the local haulage conveyors: Bogda, Pioma and Sigma are regarded to be a conveyor down-time. Among others, mechanical failures of bolters, oil leakages, a maintenance and an exchange of the machine components as well as problems with the operational system and the machine controller were classified as the machine down-times. As down-times of the suspended monorail two failures of the monorail for a transportation of materials and personnel and also delays at work, caused by them, were numbered. Among others, breaks caused by conducting methane desorption measurements in the working, methanometric tests or geological-and-surveying measurements were classified as technological reasons. Besides, the work on changing the plan of the Bw-1n test roadway drivage after a change of mining and geological conditions in the tenth month of the project realization was numbered as a technological down-time. The failures of fans, methanometric sensors and exceeding the permissible methane concentration in the working were regarded to be ventilation reasons. Water pressure failures in the fire-pipeline were marked as a lack of water. Failures of transformers and voltage failures in the mine electric grid were numbered as failures of the electric grid.

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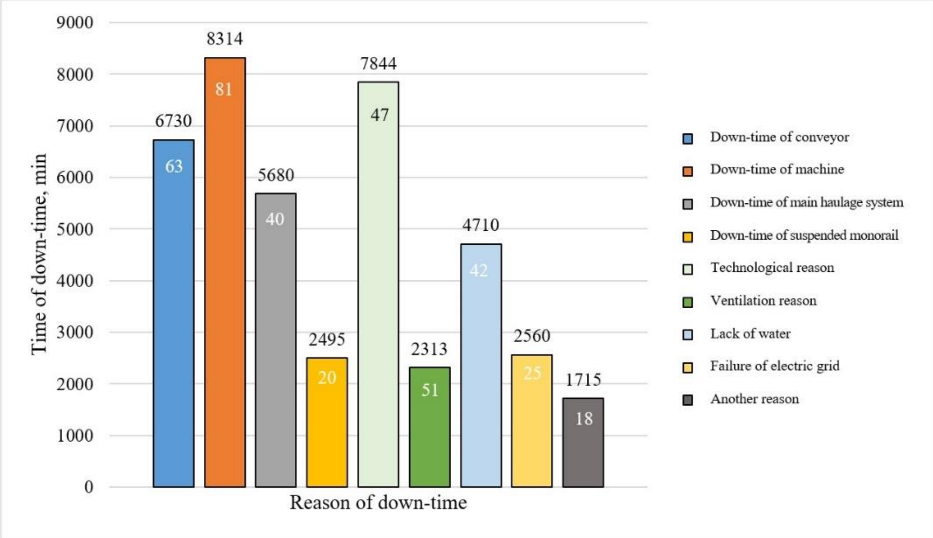

Fig.14.25. Time of down-times over the period of 10 months of the "Independent Roof Bolting Support" project realization with a specification of reasons

In Fig. 14.26 and 14.27 a percentage share of down-times over the period of ten months of the project realization in the total number and time of down-times, according to their reason, is presented.

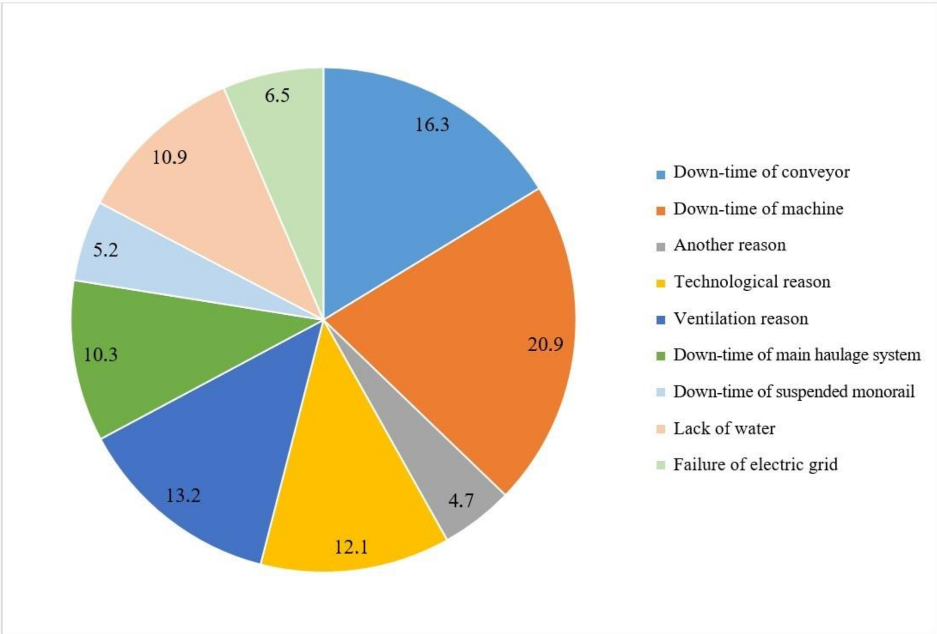

Fig. 14.26. Percentage share in the number of down-times according to the reason

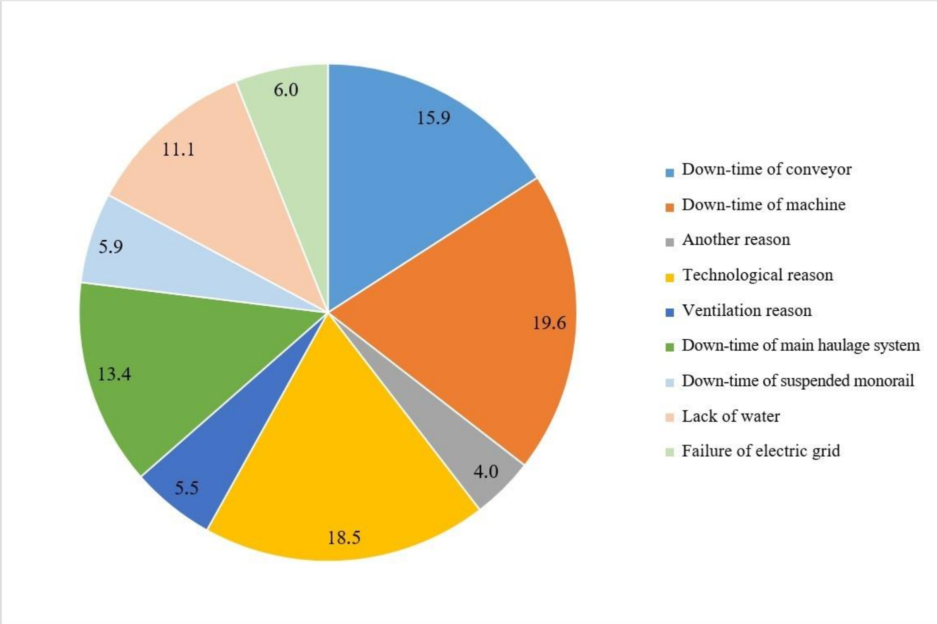

Fig. 14.27. Percentage share in the time of down-times according to the reason

Down times of the main haulage system in the Budryk mine caused in total 13.4% (95 hours) of the total time of down-times and failures during the period under consideration. Special attention should be paid to the fact, that each even minimum down-time of the main haulage system causes a stoppage of the local haulage system in the Bw-1n test roadway. The total time needed for a re-start of the local haulage equipment reaches about 8 min. In the result, if during a working shift the main haulage system of the Budryk mine is stopped minimum four times, then one cutting and bolting cycle [128] is lost in the Bw-1n test roadway.

Down-times of the local haulage conveyors, being the responsibility of the PBSz S.A., caused in total 16.3% (112 hours) of the total time of down-times and failures over the period under consideration. In this scope there are certain possibilities of undertaking corrective measures. It is mainly needed to take a current care of haulage devices. This issue was discussed during all the underground visits and inspections. A bad maintenance condition of the local haulage devices results from an insufficient number of the personnel, an insufficient experience of workers, a lack of materials and inadequate monitoring. A reduction of failures of the local haulage conveyors could improve the project efficiency level to a big extent [128].

Failures of water pressure in the fire pipeline (responsible Budryk mine) caused in total 11.1% (78 hours) of the total time of down-times and failures in the period under consideration. An essential fact is that each even minimum water pressure drops in the fire pipeline below the boundary values cause an automatic start of protections and a stoppage of the machine. In the result serious down-times happen and a course of the bolting process is disturbed, thus causing difficulties in a correct installation of bolts, resulting in further delays in a realization of the project. Due to repetitive problems with the water pressure failures on 12th May 2020 a flowmeter was built-in the fire pipeline, enabling a conditions verification of the water pressure values [128].

In connection with regular water pressure failures in the fire pipeline, certain difficulties in communication among individual subjects, realizing the project ”Independent Roof Bolting Support”, appeared. Some differences in reporting failures by the PBSz S.A. workers to the JSW Innowacje S.A. and to the Budryk mine controller were identified. It was detected that only 40% of the problems with water pressure in the fire pipeline, reported to the JSW Innowacje S.A., were simultaneously reported to the mine controller. On the other hand, after having installed the flowmeter in the pipeline, there were also situations, when problems with water were reported and the measured pressure was appropriate. It was indispensable to establish a clear division of responsibilities for individual failures and down-times. A situation, in which essential differences occur among reports, directed to individual subjects responsible for a realization of the project, cannot be tolerated [128].

Since the moment of driving the bend No. 1 (from the mark of about 240 rm) additional problems have appeared. They were connected with a low strength of the floor rocks, so it was needed to install sleepers of hard timber under the machine caterpillars. A difficulty of this type was not always reported as a down-time, nevertheless it had an essential impact on the efficiency of conducted operations. It is assessed that the total time, spent on the activities connected with an installation of timber sleepers under the machine caterpillars, was about 1 hour each shift [128].

The total time of down-times and failures during the period of ten months of the ”Independent Roof Bolting Support” project realization was 706 hours, which amounts to 29.4 twenty-four-hour periods. The most frequent down-times were caused by failures of the main and local haulage systems and also by the problems with the water pressure in the fire pipeline. Besides, in the project tenth month a serious technological down-time occurred. It was connected with a deterioration of geological conditions and a change of locating the designed bend. All these factors caused delays in a realization of the Bw-1n test roadway drivage and generated additional, unforeseen costs.

14.4. Economic analysis

In the preliminary economic analysis of the project ”Independent Roof Bolting Support” it was assessed that the cost of driving 1 m of the roadway will reach about 10 000 PLN/rm, assuming a daily advance on the level of 13.4 rm. During the period from 10th November 2019 till 30th June 2020 the total cost of driving 924 rm of the Bw-1n test roadway was 26 493 300 PLN, which shows that an average cost of driving 1 rm of the working was 28 672.41 PLN. The majority of these costs (90%) reflect real costs of drivage: costs of energy, materials and labour borne by the JSW S.A. All the other costs are connected with the payment to the JSW Innowacje S.A., resulting from the agreement on the research-and-development project realization. In the analysis no project costs were taken into consideration, so the values of the costs from the economic analysis and of the real costs can be compared only approximately [128].

The total costs of materials (elements of support and exploitational materials of the face system as well as the material costs borne by the Mine which were not directly connected with the applied technology), based on the data on the use of materials, delivered by the JSW Innowacje S.A. were [128]:

– average costs of materials directly connected with the technology – 2 269.88 PLN/m,
– average costs of materials indirectly connected with the technology – 1 756.01 PLN/m,
– total average costs of materials – 4 025.89 PLN/m.

The labour cost was assessed on the basis of the time-scale contained in the agreement between the JSW Innowacje S.A. and PBSz S.A. The costs concern the activities connected with the drivage of the Bw-1n test roadway (without the services of the Hargreaves Company and subsidiary activities) and the other activities evaluated separately. The costs were [128]:

– average costs of work connected directly with the drivage – 12 513.00 PLN/m,
– average costs of other work – 755.00 PLN/m,
– total average costs of labour – 13 268.00 PLN/m.

In Fig. 14.28 the statement of real and foreseen labour and materials costs during the drivage of the Bw-1n test roadway is presented.

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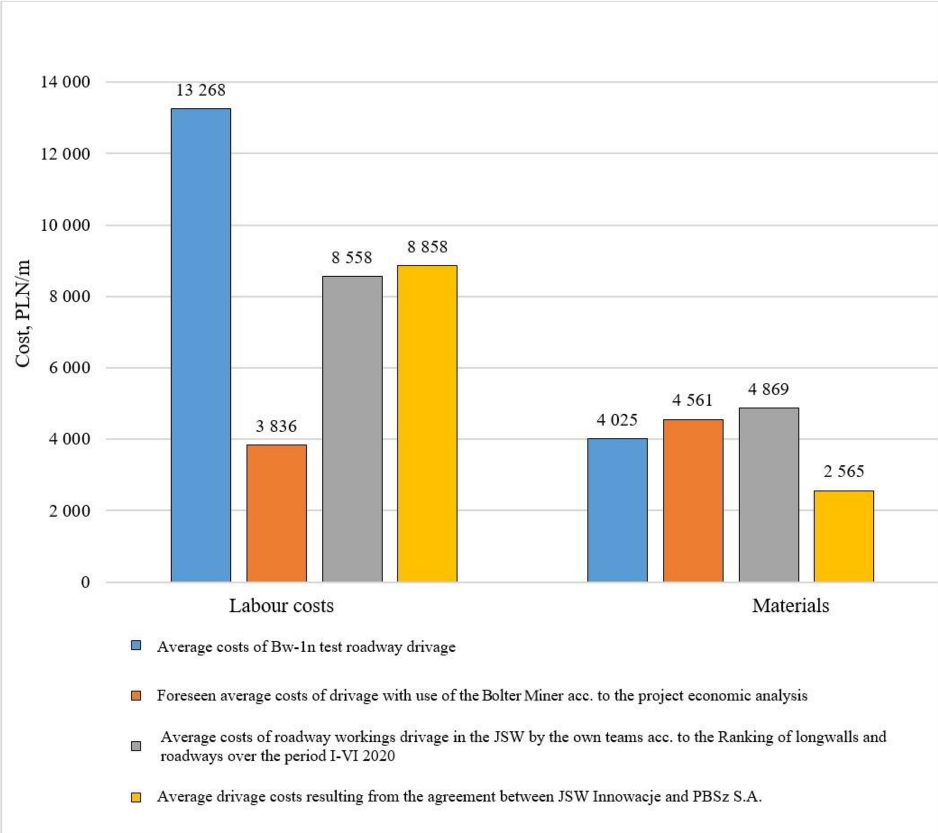

Fig. 14.28. Statement of real and foreseen costs of the Bw-1n test roadway drivage [128]

The average costs of the Bw-1n test roadway drivage are higher than it was assumed, what mainly results from a smaller advance than it was planned.

In Fig. 14.29 relationships among the working drivage costs in the independent roof bolting support as well as in the classical support of elastic arches and the monthly advance are presented.

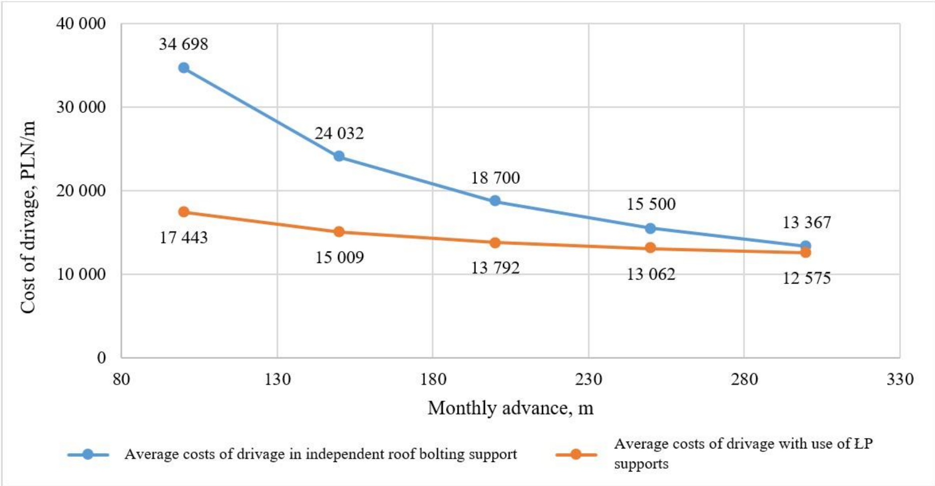

Fig. 14.29. Costs of the working drivage in relation to the monthly advance [128]

From the presented graph it can be seen that at low monthly advances the cost of the working drivage in the independent roof bolting support is nearly twice higher than the cost of driving the working with use of a traditional method. In the case of the monthly advance in the range of about 300 m, the costs of the working drivage with use of both methods are comparable. However, it should be taken into consideration that the costs of the working drivage in independent roof bolting support contain additionally the circumproject costs. Besides, there is a possibility of reducing costs of the working drivage in independent roof bolting support in such a way that at an appropriate monthly advance will be lower than the costs of driving the workings in the frame support of elastic arches [128].

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15. Summary

Artur Dyczko[1]

The project of implementing roof bolting support in the mines of Jastrzębska Spółka Węglowa S.A. (Jastrzębska Coal Company J.S.C), presented on the pages of this Monograph, aimed at a signicant efficiency increase of driving roadway workings. In the result of it, a technology of conducting mining operations with use of bolting in the system of continuous cutting and a support installation, appropriate for the conditions of deep, gassy mines of coking coal, was elaborated and implemented.

Cutting-and-bolting machines of the Bolter Miner type are commonly used in the world hard coal mining industry. They are widely used for driving and supporting roadway workings, however in room-and-pillar systems of mining they can be also used for conducting exploitational operations.

An implementation of independent bolting support in the system of continuous cutting and supporting, realized by the JSW S.A., undoubtedly is an innovative attempt of adapting this technology to difficult conditions of the Upper Silesian Coal Basin and longwall mining systems.

A detailed analysis of mining-and-geological conditions, and also of organizational factors, conducted by the Główny Instytut Górnictwa (Central Mining Institute), indicated the Bw-1n test roadway in the Seam 401 in the Budryk mine. After transporting the machine components to the pre-prepared assembly chamber, on 10th October 2019 a realization of the project "Independent Bolting Support" started.

The project was realized in the Budryk mine over a period of 13 months, during which 1168 meters of the Bw-1n test roadway located at the depth of about 900 m, were driven. A realization of the project showed that the technology of driving workings in independent bolting technology was safe and the driven working was stable. Besides, the operations, executed by the mining teams, directly in the working under drivage, were less arduous than in the case of traditional support. The day of 9th November 2020 was the last day of the project "Independent Bolting Support", whose contractors included the consortium of the JSW S.A. companies, JSW Innowacje S.A., Główny Instytut Górnictwa, JOY Global (Poland) Sp. z o.o. with a support of the Przedsiębiorstwo Budowy Szybów S.A. and the companies: SIGMA S.A. and Geofic.

A successful start of the project was possible due to an engagement of a group of workers, experienced at work with the machine of the Bolter Miner type, gained in the Czech mines, by the PBSz S.A. The fourth month of the project

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realization brought the first serious challenge, including an execution of bend No. 1, changing the direction of driving the working from the north-eastern to the eastern one. This task was accomplished with a full success. The experience, gained in previous months enabled an execution of the bend in an efficient way without any major problems or failures.

Between the fifth and the ninth months of the project realization a drivage of the straight working section of the length of about 800 m was conducted. In this period the geological conditions deteriorated which consisted in an increase of separations, observed in two measurement points of the current control of the working stability. After having conducted a local inspection, a decision about changing the bolting network, enabling a maintenance of the working stability and thus a further advance of mining operations, was taken. Besides, during the period of driving the mentioned straight section a possibility of making an enlargement for the station of transformers with use of the Bolter Miner machine was tested – the task was realized successfully.

During the drivage of the Bw-1n test roadway a geological investigation were conducted currently, enabling an appropriate adaptation to the production plans of the Budryk mine. In the area of the designed execution of the following bend No. 2 a serious deterioration of roof conditions and a local increase of inclination of rock layers happened. Based on the analysis of the geological situation a supposition was formulated that the area in which there was an adhesion of the seams 401 and 402 was ahead of the face front. In relation to that a decision was taken to change a location of the designed bend. The machine was withdrawn from the face and a part of the driven working had to be sealed. Then the machine was cut into the right wall on the mark 994 rm, starting an execution of the junction (bend No. 2), changing the driving direction from the eastern to the south-eastern one. The production plans of the Budryk mine were adapted appropriately to a new projection of the driven working.

A stability of the Bw-1n test roadway protected with the independent bolting support was monitored currently with use of five periodic control stations, five stations of convergence monitoring and also measurement stands of current control. A control of the working stability incorporated:

– testing of horizontal fissures propagation in the working roof with use of separation meters,
– testing of fissures sizes with use of introscopic cameras,
– testing of the separations scope with use of hand displacement meter probes,
– testing of bolts loading with use of measuring hand displacement meter bolts,
– testing of roof rocks strength with use of a penetrometer,

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- correctness testing of bolt gluing-in applying attempts of pulling out,
- a measurement of the working convergence.

The conducted tests and monitoring of the rock mass and of the support in the Bw-1n test roadway confirmed an efficiency of applying independent bolting support. During the period of drivage, lasting nine months, the working was fully functional and stable. A stabilization of the roof separations, as the face front was driven away from the measuring points, showed an operational correctness of independent bolting support, efficiently linking individual rock layers with one another.

The problems connected with a dynamic increase of the roof separations, occurring in two measurement points of current control in the sixth month, were stopped quickly and efficiently. A change in the bolting network, including an increase in the number and a change of the type of installed bolts, enabled a further advance of mining operations, without any impact on the personnel's safety level. Already in the seventh month an improvement of roof conditions was observed, which enabled a reduction of the number of used additional bolts.

Down-time and failures, occurring in the process of drivage, had an essential impact on daily advances achieved in the Bw-1n test roadway. Fluctuations of water pressure in the fire-extinguishing pipeline and down-time in the mine central haulage system belonged to the most often experienced problems. Besides, regular down-time of conveyors of regional haulage also contributed to a reduction of achieved advances. Without any doubts the potential of the technology under implementation was limited to a big extent by external factors and their elimination can make an increase of achieved advances possible and can also cause a reduction of costs.

Apart from the problems of technical-and-organizational nature, occurring in the drivage process, the COVID-19 pandemic had a significant impact on a realization of the project "Independent Bolting Support". In the sixth, seventh and ninth months of the project there was down-time connected with an introduction of the state of epidemic. Since the very beginning of the pandemic period the division has experienced an increased absence of workers. In the result a decrease of achieved daily advances happened and thus an increase in the project costs. In relation to the extraordinary situation the final analysis of drivage costs of the Bw-1n test roadway cannot have a decisive impact on taking a decision about a possibility of a further implementation of independent bolting support in the JSW S.A. mines.

Despite so complex mining-and-geological conditions a realization of the project in the JSW S.A. demonstrated that the technology of driving workings in independent bolting support was safe and that the driven working was stable. The operations realized by mining teams, directly in the driven working were less arduous than in the case of traditional LP arches (flexible arches).

The research-and-development character of the project enabled to conduct series of analyses, an elaboration of best practices for the working drivage and to gain valuable, operational experience in the scope of issues such as:

– a selection of optimum lay-out of machines and devices for the face equipment,
– a design of independent bolting support,
– a practical installation of bolting support in the conditions of deep gassy mines,
– a selection of optimum exploitational materials,
– monitoring of roadway workings in independent bolting support.

The conducted operations confirmed the analysis correctness of the tests results directions of maximum horizontal stresses at designing workings in independent bolting support, in particular at designing directions of block development of new panels. As design experience shows, the results of the objective tests and experts' opinions on possible effects of appropriate conducting operations, in relation to the direction of maximum horizontal stresses, were reflected during the drivage of the Bw-1n test roadway in the seam 401 of the Budryk mine. During the working drivage a local deterioration of the roof condition in the working corners was observed. The biggest impacts of horizontal stresses in the drivage direction of the Bw-1n test roadway, as regards horizontal stresses, were visible on the straight section after having driven the first bend of the working. It should be highlighted that there were no breaks in the project realization caused by excessive strengthening of support in the presence of zones situated within the impact of the main direction of horizontal stresses.

A realization of the roadway drivage was carried out at continuous monitoring of the rock mass. The rock mass monitoring, despite convergence measurements, was also performed with use of an introscopic camera. At determined sections of the working the stations of periodic control were situated. With use of instrumented bolts, hand displacement meters, hand displacement meter probes and two-and three-level separation meters, the changes of separations and acting forces were measured in in-situ conditions. The appropriate values of separations signalled a necessity of undertaking activities oriented onto strengthening the rock mass or an evacuation of personnel.

The experience from the project realization in the scope of the 12CM30 Bolter Miner cycles of operation indicated a repeatability of duration time of operational cycles. A proper work organization of mining and energy-mechanical brigades servicing the machines, a continuous improvement of qualifications and

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a personal development had a positive impact on a motivation of the workers and on a quality of their team work. A work organization on crossings of the workings, during a realization of the project, was subject to a continuous improvement, what is indicated by the time of driving individual sections of the roadway.

The project confirmed a possibility of using independent bolting support driven with use of cutting-and-bolting machine of the Bolter Miner type in the conditions of the JSW S.A. mines, which was its main objective.

The obtained measurable advantage for the project participants consisted in gaining experience in using state-of-the-art mining technologies in the Polish conditions.

A realization of the project was a source of new experience for all the consortium participants due to its unique character and type of conducted mining operations. It should be highlighted that despite many attempts of implementing independent bolting support in Polish mines in the past, the project can be treated as a pioneer solution in the Polish hard coal mining industry. An attempt of conducting a roadway working, supported with independent bolting support with use of specialistic 12CM30 Bolter Miner cutting-and-bolting machine, was undertaken for the first time.

Assessing the conducted implementation in the conditions of the Jastrzębie hard coal mines from the today's perspective, the grounds and efficiency of the accepted project realization path should be confirmed and international technological dialogue with a participation of the Polish research institutes of the hard coal mining sector and international companies should be recognized as the right decision. Since only such a formula of conducting the project was flexible enough and transparent at the same time to guarantee a successful realization of the project objectives.

The project was and it still is a reaction of the JSW S.A. to the challenges awaiting the whole Polish mining industry in the closest years and it should be a valuable inspiration in the scope of its further transformation.

A supplementary component of this Monograph is the Annex containing the photographic documentation of the realized project "Independent Bolting Support" at the JSW.

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Abstract

Bolter Miner Machine for Driving Roadway Workings - Polish Experience

The Monograph presents the experience gained by the research teams during an implementation process of commonly used in the world technology of roadway drivage with use of independent bolting support made with a cutting-and-bolting machine of Bolter Miner type in the conditions of deep cooking coal mines in the JSW S.A. This implementation was realized within the framework of the project "Independent Bolting Support" by the Jastrzębska Spółka Węglowa S.A. at the „Budryk" mine over the period of 13 months, during which 1.168 meters of the Bw-1n test roadway, situated at the depth of circa 900 m, were driven. It has been shown that the technology of driving workings in independent bolting technology is safe and the driven working is stable. Besides, the operations done by the mining teams, directly in the working under drivage, were less arduous than in the case of traditional support.

A research-and-development character of the project enabled to conduct a series of analyses, an elaboration of best practices for working drivage and gaining valuable operational experience in such issues as:

- a selection of an optimum system of machines and outfits of the face equipment,
- designing of independent bolting support,
- a practical installation of bolting support in the conditions of deep methane mines,
- a selection of optimum exploitational materials,
- monitoring of roadway workings in independent bolting support.

The conducted operations confirmed the grounds of analyzing test results of the directions of maximal horizontal stresses at designing of workings in independent bolting support, in particular at designing development directions of new panels.

A realization of the roadway drivage was conducted at continuous monitoring of the rock mass. The rock mass monitoring, apart from the convergence monitoring, was also realized with use of introscopic camera. On the determined sections of the working the stations of periodic control were located, on which in in-situ conditions the changes in stratifications and in acting forces were measured with use of instrumentation bolts, hand displacement meters and displacement meter probes as well as two-and three-level separation meters. The appropriate values of separations signalled a necessity of undertaking

activities oriented onto strengthening the rock mass or an evacuation of personnel.

The Monograph presents:

– an origin of the Bolter Miner machine,
– a history of using bolting support in the national mining industry and in the world,
– a description of the project of implementing independent bolting support in the roadway workings of the JSW S.A. mines,
– an analysis of a full spectrum of possibilities of applying the objective technology in the process of driving of opening and development workings an also an exploitation of residual parts of the seam,
– a description of mining-and-geological parameters as well as natural hazards occurring in the area of driven working,
– a description of making the bolting support for the Bw-1n test roadway (together with condition control and monitoring),
– an analysis of making bends during the roadway drivage by the Bolter Miner machine
– an analysis of advances and an assessment of efficiency of conducted mining operations.

ISBN 978-83-65593-26-9

Annex

The photographic documentation of the project “Independent Bolting Support” at the JSW

A view of the Bolter Miner 12CM30 assembled in a factory in USA

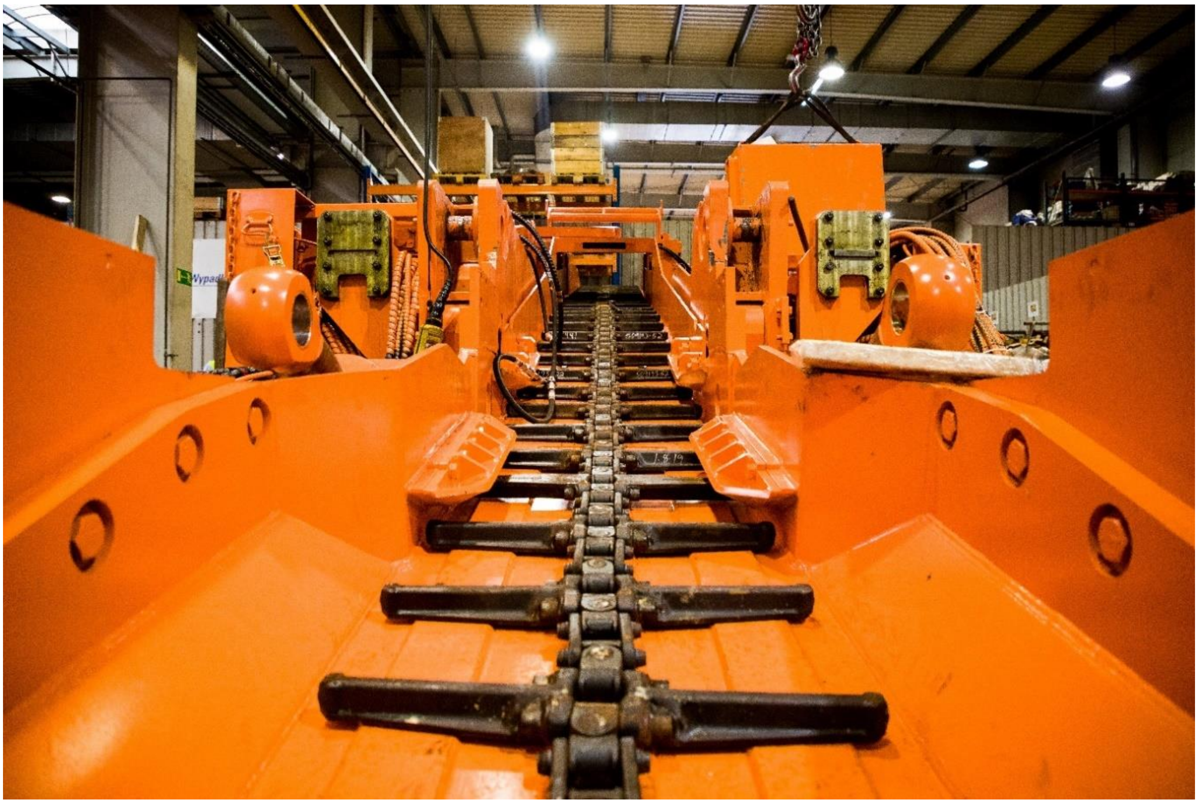
Assembling of the Komatsu Bolter Miner 12CM30 in a factory in USA

A view of the main frame of the Bolter Miner 12CM30 in a factory in USA

Bolter Miner 12CM30 during its assembly in US factory

Bolter Miner 12CM30 during its assembly in a factory in USA

Bolter Miner 12CM30 during its assembly in JOY plant in Tychy, Poland



A view of the Bolter Miner 12CM30 conveyor in JOY plant in Tychy, Poland

Bolter Miner 12CM30 during its assembly in JOY plant in Tychy, Poland

A cutting head of the Bolter Miner 12CM30 assembled in JOY plant in Tychy, Poland

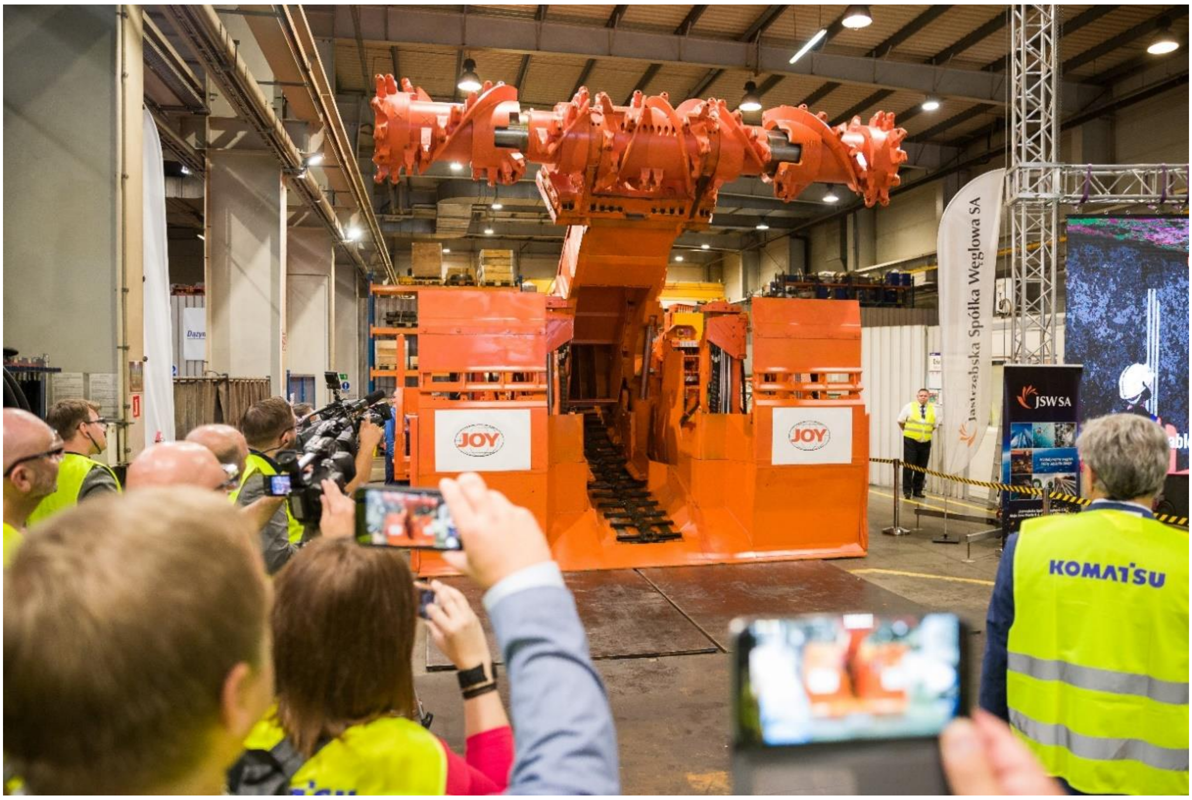
A cutting head and a loading table of the Bolter Miner 12CM30 assembled in JOY plant in Tychy, Poland

A conveyor end of the Bolter Miner 12CM30 in a Joy plant in Tychy, Poland

A view of the Bolter Miner 12CM30 assembled in JOY plant in Tychy, Poland

A view of the hydraulic system steering of the Bolter Miner 12CM30 in JOY plant in Tychy, Poland

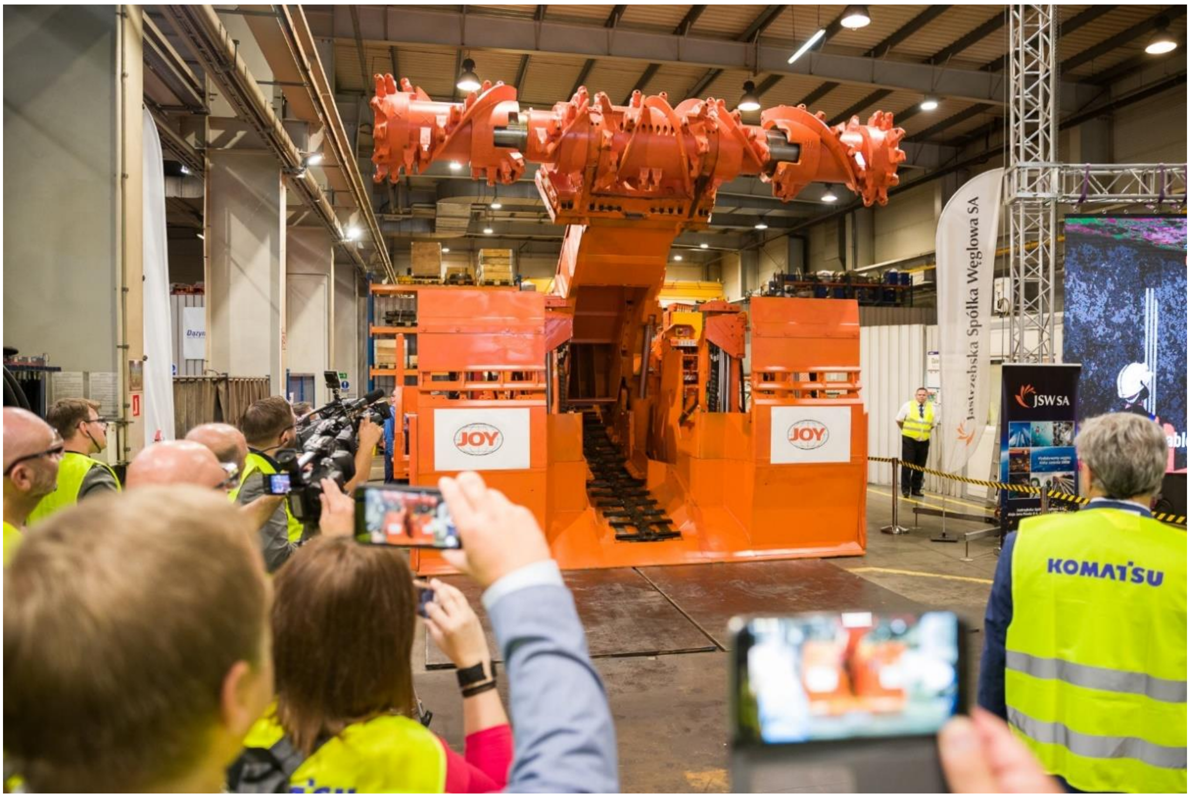
Bolter Miner 12CM during official opening of the project in JOY plant in Tychy, Poland



Bolter Miner 12CM during official opening of the project in JOY plant in Tychy, Poland

Bolter Miner 12CM during official opening of the project in JOY plant in Tychy, Poland

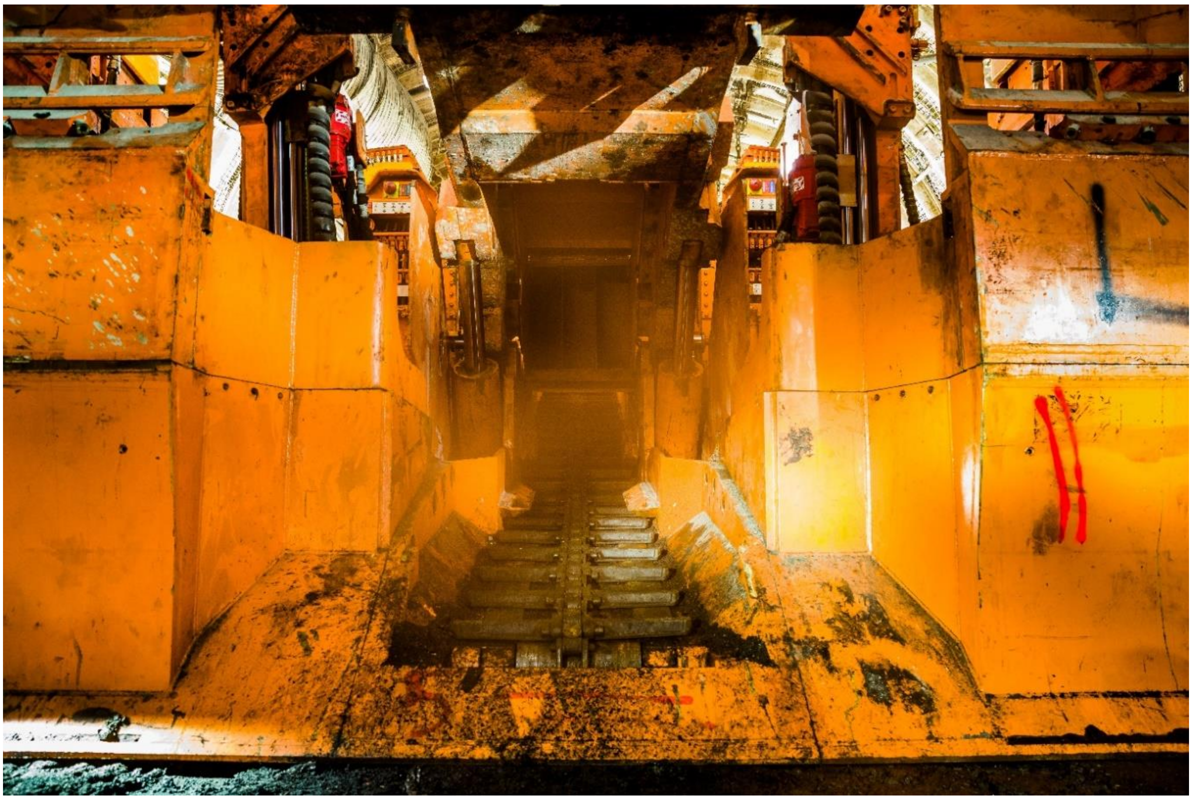
Official opening of the project in JOY plant in Tychy, Poland



Bolter Miner 12CM during official opening of the project in JOY plant in Tychy, Poland

A view of the remote controller of the Bolter Miner 12CM30 during the official opening of the project in JOY plant in Tychy, Poland

A view of the Bolter Miner 12CM30 assembled in the undergrounds of the Budryk mine – for the first time in history of Polish coal mining industry



A view of the Bolter Miner 12CM30 armed conveyor in the Budryk coal mine

A view of the Bolter Miner 12CM30 assembled in the Budryk coal mine

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A view of the Bolter Miner 12CM30 assembled in the Budryk coal mine

A view of the cutting head of the Bolter Miner 12CM30 in the face of Bw-1n roadway in Budryk coal mine

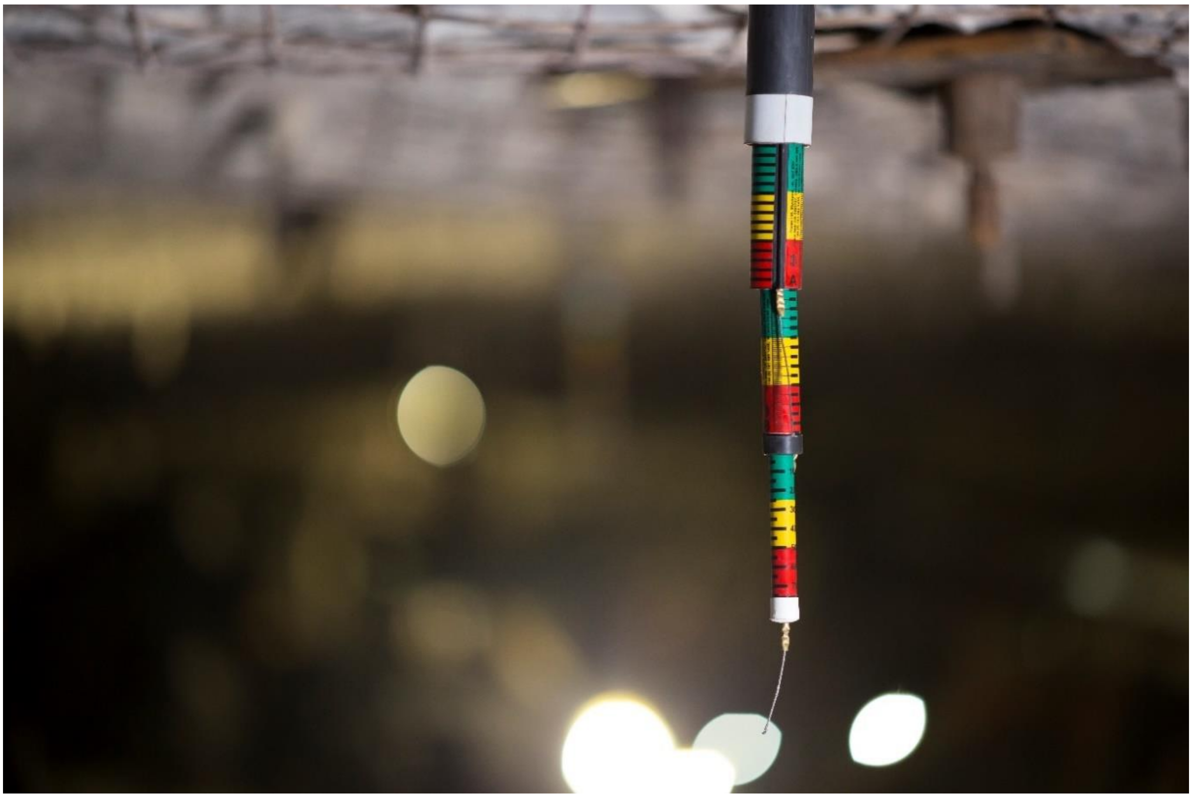
A view of the cutting head of the Bolter Miner 12CM30 in the face of Bw-1n roadway in Budryk coal mine

A view of the hydraulic system steering of the Bolter Miner 12CM30 in JOY plant in the face of Bw-1n roadway in Budryk coal mine

A view of the bolter of the Bolter Miner 12CM30 in JOY plant in the face of Bw-1n roadway in Budryk coal mine

A view of the equipment of Bw-1n roadway in Budryk coal mine

A view of the Bw-1n roadway bend in Budryk coal mine



A view of the extensometer installed in the roof of Bw-1n roadway in Budryk coal mine

A view of the extensometer installed in the roof of Bw-1n roadway in Budryk coal mine

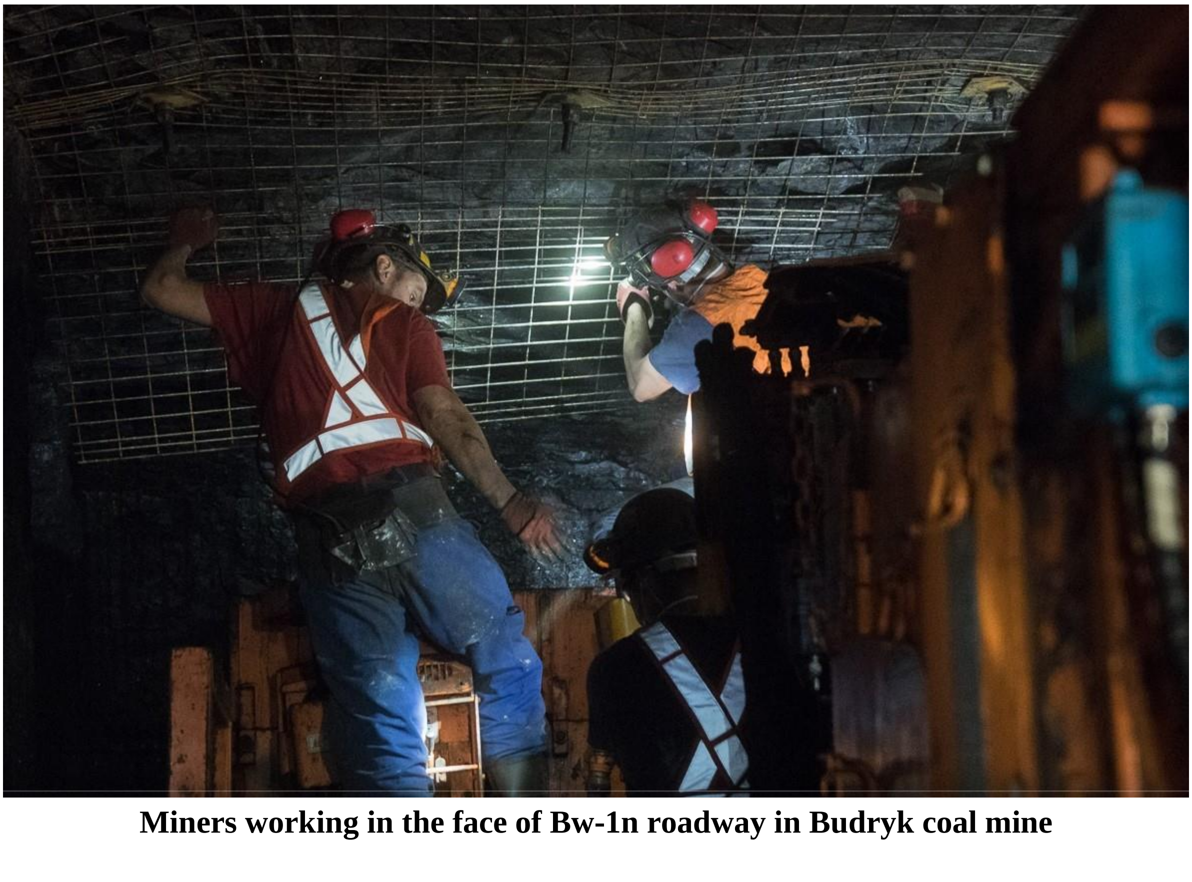
A view of cribbing in the area of Bw-1n roadway bend in Budryk coal mine

A view of the Bw-1n roadway bend in Budryk coal mine

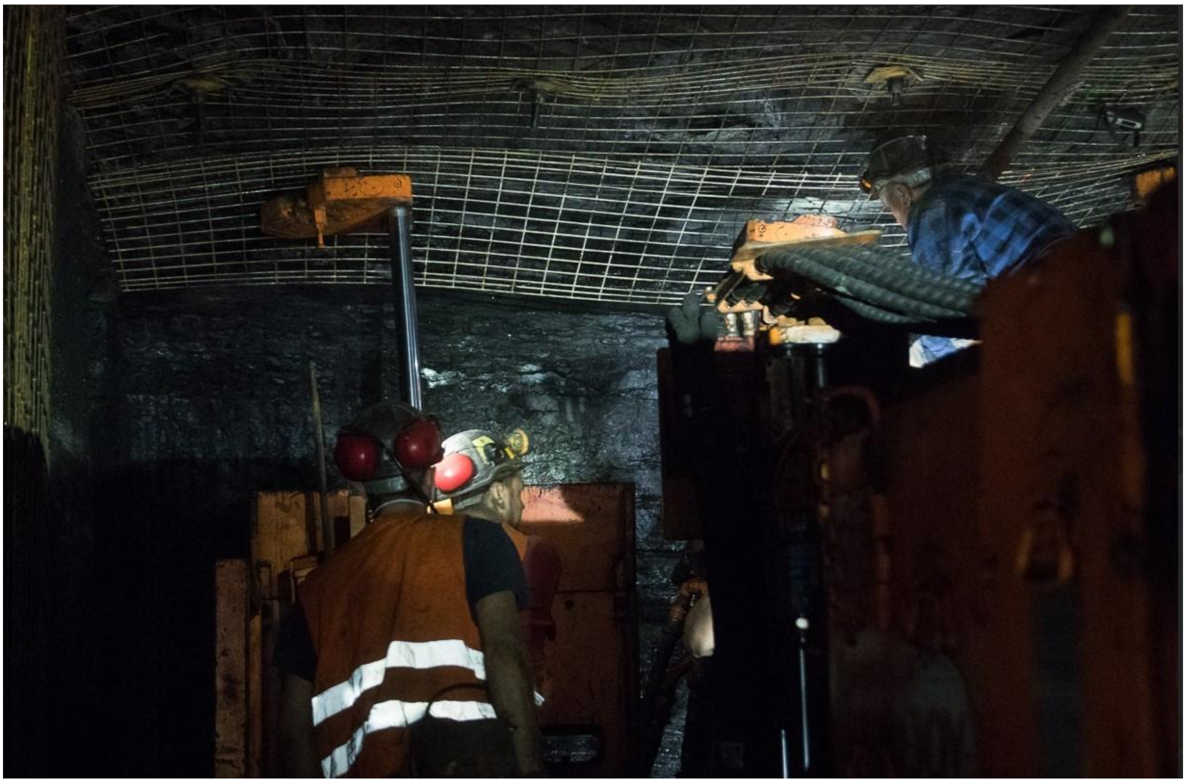
A view of bolted roof of Bw-1n roadway in Budryk coal mine

A view of bolted wall of Bw-1n roadway in Budryk coal mine

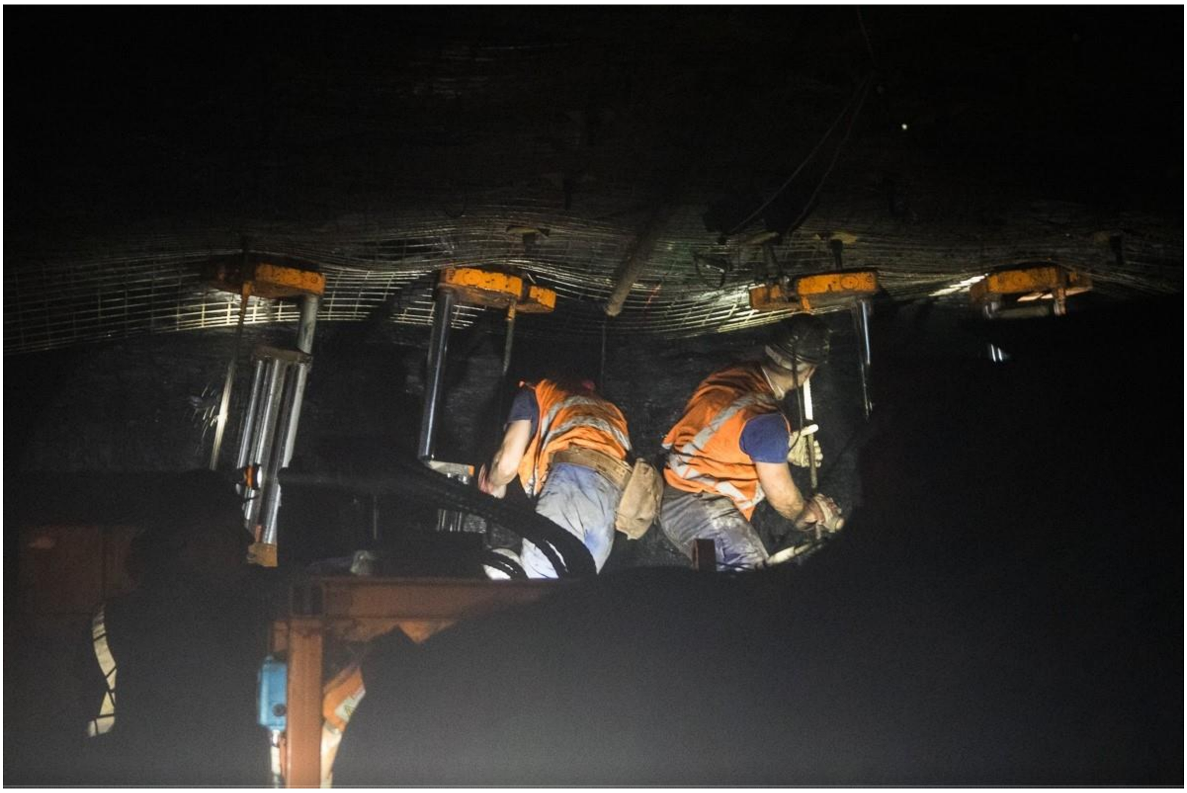
A view of bolted roof of Bw-1n roadway and a face of the roadway in Budryk coal mine



Miners working in the face of Bw-1n roadway in Budryk coal mine



Roof bolting in the face of Bw-1n roadway in Budryk coal mine



Roof bolting in the face of Bw-1n roadway in Budryk coal mine

A view of the Bolter Miner 12CM30 in the face of Bw-1n roadway in Budryk coal mine

Miner working in the face of Bw-1n roadway in Budryk coal mine

A view of bolted roof of Bw-1n roadway in Budryk coal mine

A view of the initial section of Bw-1n roadway supported with steel arches in Budryk coal mine

A view of the initial section of Bw-1n roadway in Budryk coal mine

A view of the initial section of Bw-1n roadway supported with independent rock bolting in Budryk coal mine

A view of the Bw-1n roadway bend in Budryk coal mine

A view of the Bolter Miner 12CM30 in the face of Bw-1n roadway in Budryk coal mine

A view of the conveyor end of the Bolter Miner 12CM30 in Bw-1n roadway in Budryk coal mine

A view of the suspended belt conveyor cooperating with Bolter Miner 12CM30 in Bw-1n roadway in Budryk coal mine

A view of the suspended belt conveyor cooperating with Bolter Miner 12CM30 in Bw-1n roadway in Budryk coal mine

A view of bolted roof of Bw-1n roadway in Budryk coal mine

Roof bolting in the face of Bw-1n roadway in Budryk coal mine

Widening of Bw-1n roadway supported with independent rock bolting support in Budryk coal mine

A view of the conveyor end of the Bolter Miner 12CM30 in Bw-1n roadway in Budryk coal mine

Roof bolting in the face of Bw-1n roadway in Budryk coal mine

Sidewalls bolting in the face of Bw-1n roadway in Budryk coal mine

Roof bolting in the face of Bw-1n roadway in Budryk coal mine

A view of the suspended belt conveyor cooperating with Bolter Miner 12CM30 in Bw-1n roadway in Budryk coal mine

Miners working in the face of Bw-1n roadway in Budryk coal mine

Roof bolting in the face of Bw-1n roadway in Budryk coal mine

Roof bolting in the face of Bw-1n roadway in Budryk coal mine

Miners working in the face of Bw-1n roadway in Budryk coal mine

Miners working in the face of Bw-1n roadway in Budryk coal mine

A view of bolted wall of Bw-1n roadway in Budryk coal mine

A view of the Bolter Miner 12CM30 in the face of Bw-1n roadway in Budryk coal mine

A view of Bw-1n roadway supported with independent rock bolting in Budryk coal mine

A view of the Bolter Miner 12CM30 in the face of Bw-1n roadway in Budryk coal mine

A view of the Bolter Miner 12CM30 in the face of Bw-1n roadway in Budryk coal mine

A view of the pull-out test jack during the test of the bolt in Bw-1n roadway in Budryk coal mine

A view of an extensometer installed in the roof of Bw-1n roadway in Budryk coal mine

A view of the Bolter Miner 12CM30 in the face of Bw-1n roadway in Budryk coal mine