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LABORATORY METHOD FOR EVALUATING THE CHARACTERISTICS OF EXPANSION ROCK BOLTS SUBJECTED TO AXIAL TENSION

LABORATORYJNA METODA BADANIA CHARAKTERYSTYK KOTEW ROZPRĘŻNYCH PODDANYCH ROZCIĄGANIU OSIOWEMU

Rock bolts have long been used in Poland, above all in the ore mining. Worldwide experience (Australia, Chile, Canada, South Africa, Sweden, and USA) provides evidence of rock bolt supports being used for loads under both static and dynamic conditions. There are new construction designs dedicated to the more extreme operating conditions, particularly in mining but also in tunneling. Appreciating the role and significance of the rock bolt support and its use in Polish conditions amounting to millions of units per year, this article describes a new laboratory test facility which enables rock bolt testing under static load conditions. Measuring equipment used as well as the possibilities of the test facility were characterized. Tests were conducted on expansion rock bolt supports installed inside a block simulating rock mass with compression strength of 80 MPa, which was loaded statically as determined by taking account of the load in order to maintain the desired axial tension, which was statically burdened in accordance with determined program load taking into consideration the maintenance of set axial tension strength at specified time intervals until capacity was exceeded. As an experiment the stress-strain characteristics of the rock bolt support were removed showing detailed dependence between its geometrical parameters as well as actual rock bolt deformation and its percentage share in total displacement and deformation resulting from the deformation of the bolt support elements (washer, thread). Two characteristic exchange parts with varying intensity of deformation /displacement per unit were highlighted with an increase in axial force static rock bolt supports installed in the rock mass.

Keywords: rock bolt support, laboratory test bed, stress-strain characteristics

Obudowa kotwowa jest już od dawna stosowana w Polsce, przede wszystkim w górnictwie rudnym. Światowe doświadczenia (Australia, Chile, Kanada, RPA, Szwecja,USA) świadczą o stosowaniu obudowy kotwowej zarówno w warunkach obciążeń o charakterze statycznym jak i dynamicznym. W podziemnych wyrobiskach górniczych wykonywanych na dużych głębokościach, szczególnie przy eksploatacji złóż rud miedzi w kopalniach LGOM, w których stosuje się samodzielną obudową kotwową istnieje niebezpieczeństwo nieprzewidzianego odpadania bloków skalnych do przestrzeni roboczej. Podstawowym zadaniem kotwienia wyrobisk górniczych jest zapewnienie ich stateczności, jako zasadniczy warunek

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bezpieczeństwa pracy. Powstają nowe konstrukcje przeznaczone do bardziej ekstremalnych warunków funkcjonowania, w szczególności w warunkach górniczych, ale również w tunelarstwie. Podstawowym rodzajem obudowy wyrobisk przygotowawczych i eksploatacyjnych w podziemnych kopalniach LGOM jest obudowa kotwowa rozprężna lub wklejana. Wybór sposobu utwierdzenia obudowy kotwowej zależy miedzy innymi od: czasu użytkowania, klasy stropu, wymiarów oraz przeznaczenia wyrobiska. W polach eksploatacyjnych, gdzie okres od wykonania wyrobiska do jego likwidacji jest stosunkowo krótki, częściej stosuje się kotwy rozprężne, które ze względu na mniejszą czasochłonność zabudowy, pozwalają na większą wydajność kotwienia. Doceniając rolę i znaczenie obudowy kotwowej oraz jej zużycie sięgające w warunkach polskich milionów sztuk rocznie, w niniejszym artykule opisano nowe stanowisko laboratoryjne umożliwiające badanie rzeczywistej obudowy kotwowej w warunkach obciążeń statycznych. Stanowisko laboratoryjne do badania wytrzymałości na rozciąganie obudowy kotwowej zbudowane w Katedrze Górnictwa Podziemnego AGH umożliwia badania obudów kotwowych przy różnych warunkach obciążeń. Składa się ono z kilku współpracujących ze sobą podzespołów: Hydraulicznego Układu Obciążającego Kotew (HUK), pulpitu sterującego I, pulpitu sterującego II, pulpitu rejestrującego oraz zespołu agregatu hydraulicznego (Rys. 1). W artykule scharakteryzowano zastosowaną aparaturę pomiarową oraz możliwości badawcze stanowiska badawczego. Pomiar siły na stanowisku laboratoryjnym był wykonywany za pomocą czterech tensometrycznych czujników siły. Czujniki były rozmieszczone co 90 stopni na tarczy pomiarowej (Rys. 4). Całkowita siła rejestrowana podczas badań rozciągania żerdzi kotwowej była sumą wartości sił uzyskiwanych na poszczególnych czujnikach siły. Pomiar przemieszczeń elementów obudowy oraz wydłużenia żerdzi kotwowej był wykonywany za pomocą enkodera linkowego inkrementalnego. Enkoder przymocowany był na stałe do bloku siłowników (Rys. 6), natomiast linka enkodera przemieszczała się wraz z wysuwem tarczy pomiarowej (Rys. 6). W celu określenia odkształcenia materiału badanego elementu (żerdzi kotwowej) w badaniach zastosowano tensometry elektrooporowe typu kratowego (Rys. 7). Czujniki siły, przemieszczenia oraz odkształcenia zostały podłączone do uniwersalnego wzmacniacza pomiarowego QuantumX MX840, za pomocą wtyczek 15-pinowych. Podczas procesu rozciągania kotwy wyniki pomiarów siły, przemieszczenia oraz odkształcenia były rejestrowane na bieżąco za pomocą specjalistycznego programu z dziedziny technik pomiarowych „CATMAN – EASY”. Wybór programu wynikał z możliwości współpracy z systemem operacyjnym MS Windows oraz połączenia komputera z uniwersalnym wzmacniaczem pomiarowym QuantumX MX840 poprzez kabel ethernetowy. Program umożliwiał bieżącą (on-line) wizualizację i ocenę pomiaru. Ponadto po zakończeniu testu, tworzone były raporty dokumentujące wyniki pomiarów, które były zapisywane w rozszerzeniu pliku ASCII. Następnie dane były przesyłane do programu Microsoft Excel w celu analizy uzyskanych wyników. W badaniach zastosowano obudowę kotwową rozprężną, zainstalowaną w bloku symulującym górotwór o wytrzymałości skał na ściskanie wynoszącej 80MPa (Rys. 3), która była obciążana statycznie według ustalonego programu obciążenia uwzględniającego utrzymywanie zadanej osiowej siły rozciągającej w określonych przedziałach czasowych, aż do przekroczenia nośności. W badaniach zastosowano obudowę kotwową rozprężną, która stanowi podstawową obudowę wyrobisk eksploatacyjnych w ZG „Polkowice – Sieroszowice”. Obudowa składała się z żerdzi kotwowej typu RS-2N (Tabl. 1). Żerdź kotwowa współpracowała z głowicą rozprężną typu KE3-2K (Tabl. 2, Rys. 9). W badaniach zastosowano profilowane podkładki kotwowe okrągłe o grubości 6 mm (Tabl. 3, Rys. 10). Eksperymentalnie zdjęto charakterystykę naprężeniowo-odkształceniową kotwy (Rys. 12) pokazując szczegółowo zależności pomiędzy jej parametrami geometrycznymi, odkształceniami właściwymi żerdzi kotwowej oraz ich procentowego udziału w całkowitych przemieszczeniach i odkształceniach wynikających z deformowania się elementów składowych obudowy kotwowej (podkładka, gwint) oraz przemieszczeń głowicy rozprężnej. Wyróżniono dwie znamienne części charakterystyki różniące się wielkością intensywności odkształceń/przemieszczeń przypadających na jednostkowy przyrost wartości siły osiowej obciążającej statycznie kotew zainstalowaną w górotworze.

Słowa kluczowe: obudowa kotwowa, laboratoryjne stanowisko badawcze, charakterystyka naprężeniowo-odkształceniowa

1. Introduction

Underground mining of mineral deposits, in particular at greater depths, is carried out under static and dynamic loads. Excavation mining is subject to deformation, pillars are gradually being destroyed, followed by separation of roof rocks or other symptoms of rock mass destruction. The lithological diversity, layer divisibility and the presence of weakened surface cohesion between the layers cause bending of floor layers, cracks and consequently the loss of stability of excavations, in which the primary means of protection is the rock bolt support (Kidybiński, 2005). In the future, we will have to deal with the worsening dynamic effects of the circumstances described above, whose causes are described by many authors (Nierobisz, 2012; Thompson et al., 2012; Cai, 2013). Modern rock bolt support structures are often based on a material with high tensile strength, which, together with preserving the required load capacity, also ensure acceptable parameters of elongation, axial displacement of the rock bolt or deflection of the supported ceiling (Korzeniowski & Skrzypkowski, 2011; Skrzypkowski, 2012). In mining excavations, in which individual rock bolt supports exist, there is a danger of unforeseen blocks of rock falling to the excavation. One of the conditions for safe mining is the ability to maintain certain geometrical parameters of the excavation within the required time period. In ore mining there are cases that in spans smaller than the limits, signs of local instability in the form of rock caving, landslides, or critical separation of roof occur due to the effects of structural disturbance (Korzeniowski, 1994, 1999). For many years, both in Poland and around the world, research has been carried out to improve the design of the rock bolt support structure and its interaction with the rock mass, as well as in scope of its validation or development of optimal design methods. The most reliable research results are obtained in positions where the size of the load and course of time is similar to actual "in situ" conditions (Stoiński, 1990). Research on the properties of individual components applies to both the individual rock bolts and support structures tested at full scale. Furthermore, individual elements of additional equipment, e.g. lagging, are examined as well (Radwańska, 2005; Complete, et al., 2013).

The paper presents research methodology and sample results obtained in the laboratory setting of a test facility enabling tests of rock bolt supports in their natural size, characteristic, above all, for underground metal ore mines.

2. Laboratory test facility for testing rock bolts

The laboratory test facility for testing the tensile strength of rock bolts in the AGH Department of Underground Mining (Katedra Górnictwa Podziemnego AGH) enables testing of rock bolt supports under various load conditions. It consists of several components interacting with each other: Hydraulic loading arrangement of rock bolts (HUK), control panel I, control panel II, recording panel and hydraulic power unit (Fig. 1).

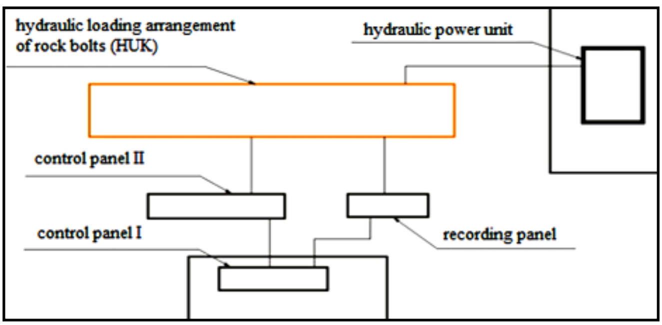


Fig. 1. A set of components in the laboratory test facility

Power is supplied to the system by a hydraulic power supply unit constructed on the basis of an axial piston pump with variable displacement, constant power regulators and constant pressure. The power supply is located in a separate room in order to eliminate excessive noise during its operation. The AC hydraulic reservoir is filled with HL46 hydraulic oil. The hydraulic oil tank capacity is 450 dm^3. Engine power of the hydraulic unit is 18 kW. Maximum supply pressure is 31 MPa. In addition, the system is equipped with a pressure switch (differential pressure sensor). Control panel I is located in the control room and is used to set the programs of the pull out machine. Control panel II is located in the main area, in direct vicinity of the pull out machine. It is used during the construction of the rock bolt support as well as during static testing. A measurement amplifier (QUANTUM MX840), to which power, displacement and deformation sensors are connected, is found on the recording panel. The results are recorded on a computer using specialized software (CATMAN-EASY).

Tests on the rock bolt support in the laboratory test facility can be carried out using the appropriate operating mode of the HUK system. Rock bolts supports should be installed as a first step. The rock bolt support installation mode is used to perform the preparatory activities prior to the trial. It served to set all working components in their required positions. With regard to tests on the expansion rock bolt support, it is treated as initial tension of the rock bolt. The load method is established after constructing the rock bolt support: this can be static or dynamic. Power is gradually increased in the static mode. The rock bolt may remain under force at a fixed level. The load increase of the rock bolt tested as well as its retention time is carried out with the help of a pressure regulator until complete breakage of the rock bolt. In the dynamic load mode, the rock bolt is stretched and broken by the use of maximum force in the shortest possible time resulting from the power of the pump.

2.1. Hydraulic loading arrangement of rock bolts – HUK

The HUK system (Fig. 2) consists of a support frame (1), on which block 8 of the hydraulic cylinders (2), (3) is placed. They may be combined in pairs of 2, 4 and 8 of servo motors operating together, depending on the strength of the test material. When testing long components on the support frame, additional dual bushings (4) are placed behind the cylinder block (4). The crossbeam of the hydraulic motor moves in the guides of the support frame (5) moved by cylinders (6).The installed rotary cam engine (7) allows the rock bolt to rotate around its own axis. There is the possibility of three areas of maximum axial tensile force: 400 kN, 800 kN and 1,600 kN. Maximum piston displacement is 800 mm. The maximum rate of extension of the measuring disk is 0.03 m/s. The design of dual bushings enables the release of upper and lower parts (Fig. 3). The dual bushings were filled with a concrete compressive strength R_c = 80 MPa (Fig. 3).

2.2. Instrumentation

Force measurement at the laboratory test facility was carried out using four strain gauge force sensors. The sensors were placed every 90 degrees on the measurement disk (Fig. 4). The total force recorded during the bolt rod tension tests is the sum of the forces generated at the various sensor strengths. Type FFM 50 sensors were used in the test. Each of the sensors mounted is characterized by a nominal load equal to 500 kN.

Each sensor was calibrated on the AG Walter + Bai DB 3000/200 pull out machine, by controlled compression. As part of the test, a load scheme, which consisted in determining the

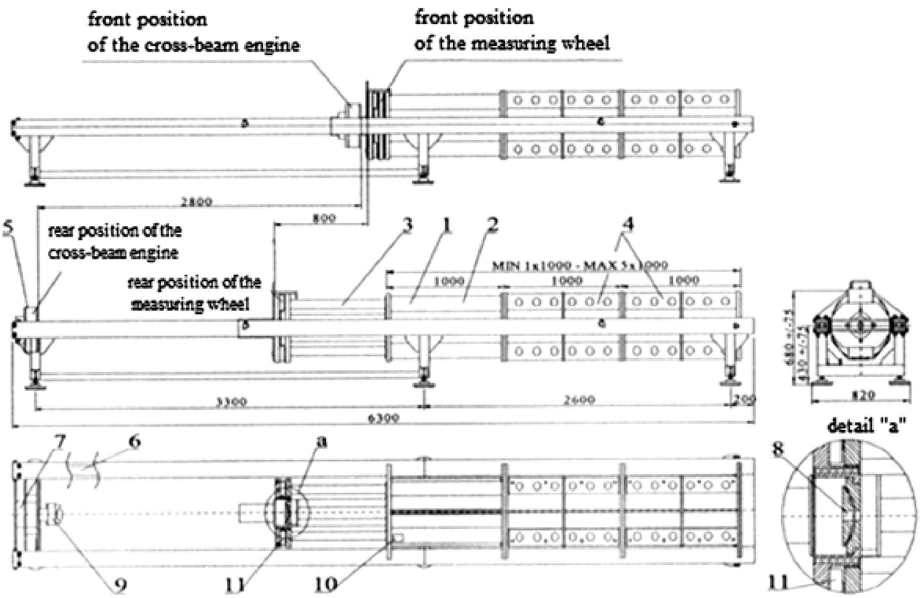


Fig. 2. Cylinder blocks. 1 – support frame, 2 – cylinder block, 3 – cylinder, 4 – dual bushings, 5 – hydraulic motor crossbeam, 6 – rock bolt support cylinders, 7 – rotary cam engine, 8 – support plate, 9 – three jawed clamp, 10 – displacement registration sensor, 11 – force registration sensor

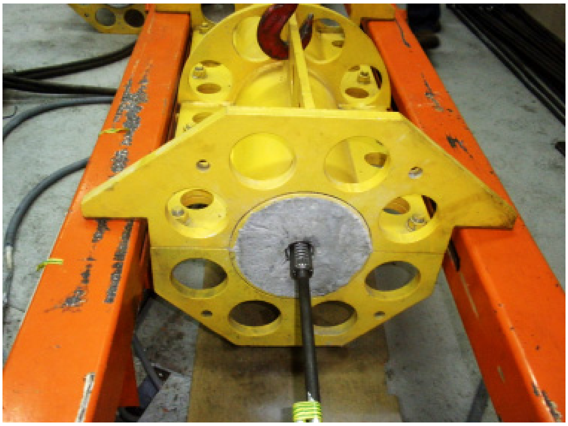


Fig. 3. Dual bushings – rock bolt mounted in a concrete block (*photo: K. Skrzypkowski*)

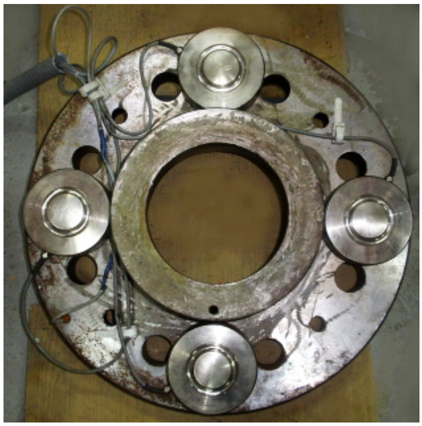


Fig. 4. Measuring disk with distributed force sensors (*photo: K. Skrzypkowski*)

measure of increase in force at every 5 kN, was established. The crosshead speed was 0.5 kN/s, while the retention time was 10s. Control sensor calibration was carried out by comparing two independent indicators: the pull out machine sensor and the measuring amplifier. The sensors were tested to 50% of nominal value (250 kN).

The second test validating the correctness of reading of acting forces consisted of resistance tests on the tensile strength of the bolt rod and specially prepared samples made of the same material (Fig. 5). The middle section of each sample (at a length of 100 mm) was rolled to a diameter of 10 mm. The study involved comparing the forces required to break off the sample on the pull out machine and the HUK machine.

a) b) c)

Fig. 5. Comparative studies of rolled samples. a) tenfold proportional samples, b) VEB Thüringer Industriewerk Rauenstein type: CRB 2214 pull out machine, c) rolled bolt rod (*photo: K. Skrzypkowski*)

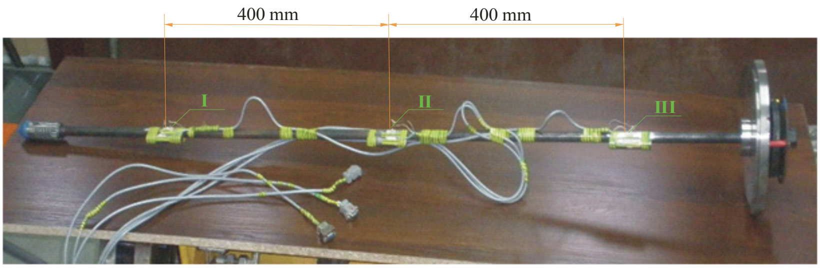
The results obtained in the test are in the range of 699 MPa to 703 MPa and overlap in terms of the tensile strength of ATLAS III steel grade (R_m = 697÷705) (Certificate of receipt, 2012).

<u>Displacement measurements of support elements</u> (in the absence of sufficient dilation of the expansion head in the block) and the tension of the bolt rod was carried out using a HLS – S – 10 – 01 type incremental cable encoder. The encoder has a 1000 mm cable length in which the maximum displacement speed is 1000 mm/s, while the encoder resolution is 0.1 mm/impulse. The encoder is permanently attached to the cylinder block (2), while the encoder cable moves together with the extendable measuring disk (1) (Fig. 6).

<u>In order to determine the material deformation of the components tested</u> (bolt rod), a lattice type electro resistant strain gauge (Fig. 7) made of constantan wire (60% Cu + 40% Ni) with a diameter of 0.025 mm was used in the tests. The resistance wire was placed on a paper support pad (100% tissue paper). The tips supplying current were made of coil wire (tin-plated copper tape with a width of 0.55 mm and a thickness of 0.15 mm). Strain gauges were characterized by resistance of 120 Ω, measurement base length of 10 mm and sensitivity factor of deformation of $k = 2.15$. The choice of a lattice strain gauge resulted from the following technical characteristics: lattice type strain gauges are not sensitive to lateral deformation; however, the hose-type strain gauges are sensitive to deformation perpendicular to their length. The rod surface was thoroughly

Fig. 6. Cable encoder installed on the HUK machine (*photo: K. Skrzypkowski*)

cleaned with fine sandpaper so that the strain gauge could be attached to the bolt rod. After the surface was mechanically cleaned it was subjected to chemical purification, so that degreasing with acetone could be carried out. The cleaned metal surface was coated with a thin layer of a special quick-drying adhesive (HBM Z70). Alongside axial tensioning of the bolt rod, the strain gauges were connected to the Quantum X 840 measuring amplifier in such a manner that the active t_1 strain gage was attached alongside the axis of the stretching rod and the compensatory strain gauge t_2 was attached to an idle piece of material identical to the tested material and placed near to the active strain gauge t_1.



I - first measurnig base of strain gauges, ($\varepsilon 1$),
II - second measuring base of strain gauges, ($\varepsilon 2$), III - third measuring base of strain gauges, ($\varepsilon 3$)

Fig. 7. Temperature compensation – rock bolt with three measurement bases (*photo: K. Skrzypkowski*)

Displacement and deformation force sensors were connected to a <u>QuantumX MX840 universal measuring amplifier</u> via 15-pin plugs. The individual transducers were connected to the recorder according to the following scheme:

- full bridge – strain gauge force sensors,
- half-bridge – strain gauge strain sensors,
- measuring the signal frequency of the rotation direction (unipolar system) – displacement sensor (incremental cable encoder).

During the process of rock bolt expansion, the results of force measurement of displacement and deformation were recorded continuously by a program specialized in the field of measurement technology – known as "CATMAN – EASY". The choice of program resulted from the possibility of co-operating with MS Windows operating system and connecting the computer with the QuantumX MX840 universal measuring amplifier via an Ethernet cable. The program enabled on-going (on-line) visualization and evaluation of the measurement. In addition, after the tests were completed, reports documenting the results of the measurements were created and then stored as an ASCII file extension. Following this, the data was transferred to Microsoft Excel for results analysis.

3. Load and deformation characteristics – deformation of the expansion bolt support tensioned statically

Static tests of expansion rock bolts were carried out at the laboratory test facility in simulated mining conditions, in particular for the KGHM Polska Miedź S.A. mine. The aim of the study was to obtain stress and deformation characteristics. Studies were carried out using a consistent methodology. Expansion rock bolts used in the study form the basis of mining excavation supports at ZS "Polkowice – Sieroszowice". The support consisted of an RS-2N type rock bolt with a length (l) from 1823 mm to 1834 mm and a diameter (d) from 18.24 mm to 18.29 mm (Table 1).

TABLE 1

Technical parameters of RS-2N rock bolts

Parameter	Characteristics
Steel grade	ATLAS III
Cross-section	Round, smooth
Rod diameter d (actual), mm	18,24÷18,29
Rod length l (actual), mm	1823÷1834
Thread*	M20, cold rolled
Thread length L, mm	170
Active rod length subject to extension l_0, mm	1670
Core diameter of bolt rod d_3 (actual), mm	16,45÷16,53
Outer diameter of bolt rod d_z (actual), mm	19,70÷19,86
Thread length (actual), mm	168÷170
Tensile strength, R_m, MPa	770
Yield strength $Re_{(R0,2)}$, MPa	440÷448
Elongation after rupture** A_5, %	21,2÷21,4
Mass 1 mb, kg	2,043÷2,050

Note:
* A crumpled zone is formed as a result of cold rolling – thanks to which the material is hardened (plastic deformation causes an increase in the density of the crystallographic network of defects). An increase in the dislocation density causes metal strengthening since a reduction in the distance between dislocation rises, and hence an increase of strength as a result of interaction between them (Wendorff, 1968; Przybyłowicz, 1992).
** Values specified by the manufacturer refer to steel mill smelting.

The bolt rod has an M20 cold rolled thread on one end, while the other end the rod is swollen (has a square profile) and adapted for transmitting torque (Fig. 8). The rod cooperates with the KE3-2K type expansion head which has a length of 136 mm and a diameter of 36 mm (Table 2,

Fig. 9), and is composed of three jaws, two K-2143-2-2 type expanders, 31 mm φ springs and a conical cap. Round profiled bolt washers with a thickness of 6 mm were used in the studies (Table 3, Fig. 10). The choice of this shape of washer with a rock bolt support resulted from its common use in the ore mining.

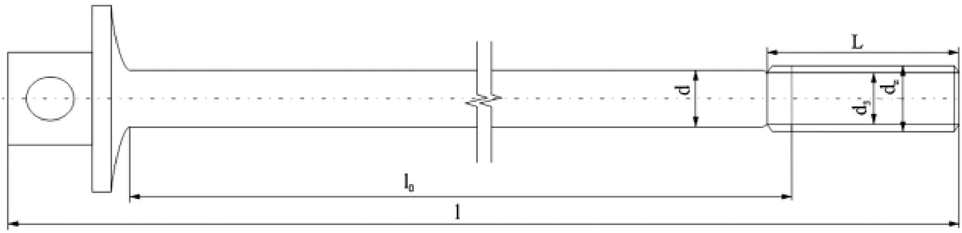


Fig. 8. RS-2N bolt rod

TABLE 2

Technical characteristics of the KE3-2K expansion head

Parameter	Characteristics
Length of head, mm	136
Diameter of head, mm	36
a) Mechanical parameters of KE3-2K GG81/95 rock bolt jaws – resistance properties of the Φ12 mm tested samples	
Type of material	White malleable cast iron
Tensile strength R_m, MPa	521÷534
Elongation,%	7,2÷7,8
Hardness, HB	167÷176
b) Mechanical camshaft parameters K-2143-2-2a – resistance properties of the □12 mm tested samples	
Type of materials	White malleable cast iron EN-GJMW-400-5 wg PN-EN 1562:2000
Working length of thread, mm	26,50
Tensile strength R_m, MPa	521÷539
Elongation,%	7,2÷7,9
Hardness, HB	163÷176

TABLE 3

Technical characteristics of rock bolt washer

Parameter	Technical characteristics	
	Round washer	Square washer
1	2	3
Steel grade	S235JR	
Washer thickness t, mm	6,00	
Free height of uncompressed washer H, mm	16,60	18,80
Inner diameter d, mm	22,00	22,00

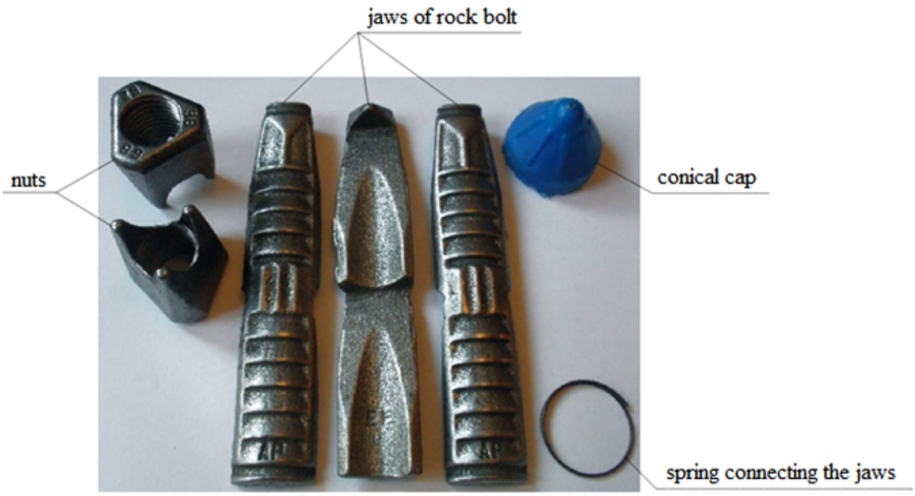


Fig. 9. Head expansion components (*photo. K. Skrzypkowski*)

1	2	3
Outer diameter D, mm	141,00	–
Side length a, mm	–	145
Collaborative diameter of disk spring d_t, mm	93,30	–
Tensile strength, R_m, MPa	403÷442	
Yield strength R_e, MPa	314÷338	
Elongation of samples after rupture A_5, %	36,2÷39,0	

In order for laboratory studies to best satisfy mining conditions, dual bushings (Fig. 3) filled with concrete with a compressive strength of at least 80 MPa were used at the laboratory test facility. Openings with a diameter of 37 mm to 38 mm were made in the 220 mm diameter cylindrical concrete blocks. The axially arranged openings had the same diameter as the openings in the mining chamber roof. PVC tubes were used to make the opening. These tubes were placed axially in the dual bushings while being filled with a concrete mix. The tubes were pulled out after a period of "setting" of the concrete mix. The procedure for installing the rock bolt support commenced after a period of 28 days from the time of filling the dual bushings. The sensored rock bolts were fed manually into the concrete block by a measuring disk and an opening in the hydraulic rock bolt load support arrangement (HUK). Afterwards, the rock bolts were installed in the openings using a torque of 250 Nm, in accordance with the building requirements for expansion rock bolts used in KGHM Polska Miedź S.A. The differences between the diameters of the rock bolt borehole and the diameter of the head did not exceed 2 mm. A difference greater than 2 mm caused broaching of the rock bolt by jaw head expansion (chamfering of threads). The advantage of the jaw head expansion is arbitrary of its dilation in the opening without giving it initial tension, though this applies only to the possibility of tests in the laboratory. This is due to the structure of three jaw heads which block axial movement of the rod. The head installed in the concrete block caused dilation, whereas the second point of dilation was the profiled bolt rod

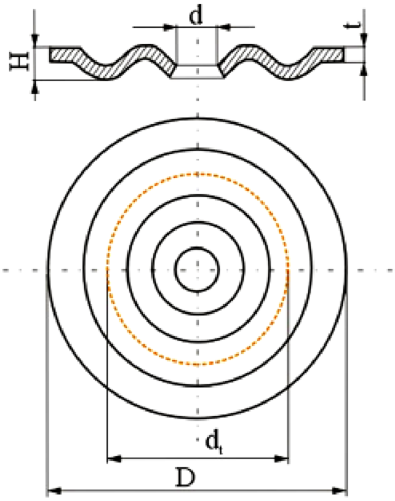


Fig. 10. Profiled rock bolt washer

head preceded by a profiled washer resting on a retractable disk of the hydraulic equipment for rock bolt load support arrangement (HUK). After dilation of the expansion head in the opening, initial tension was given at 30 kN. Next, the value of the static tension value of breaking strength of 400 kN was set on control panel I. Static tensile tests of the rock bolt support consisted of intermittent increases in pressure of 10 bar, which were performed using a pressure reducing valve knob located on control panel II. Each increase in pressure was preceded by a 10 second pause. The duration of the test up to the breaking point was an average of 120 seconds and was within the range of both the requirements imposed by the PN-EN 10002-1 (2004) standard for tensile tests as well as the corresponding growth of static load in the excavation chamber. During the entire load cycle the expansion force value was continually measured using four force KKM 50 type sensors mounted on the retractable disk of the hydraulic unit (HUK). Furthermore, the distortion value of rod was recorded with the help of three electro-resistant strain gauges attached to its rolled part at equal distances of 400 mm between each other. In addition, the station was equipped with one encoder cable, recording the total elongation to rupture. For the entire duration of the static stretching process all the data from the various sensors was recorded using the "CATMAN – EASY" specialized software in the field of measurement technology.

The following definitions were introduced to facilitate analysis of the results:

Bolt rod diameter, d – is the diameter of the rolled rod,

Bolt rod diameter, D_3 – is the smallest diameter of the rod; in the case of a rod with a M20 thread, it is the thread root diameter,

Active length of bolt rod, $l0_0$ – is the length subjected to tensile stress, the section of dilation of the concrete block head to the rod flange from the side of the washers) (Fig. 8),

Breaking force, F – is the transgression force which was followed by breaking the continuity of the material (load limit value)

Displacement, Δl – this is the value of the maximum breaking elongation of the rock bolt. The rock bolt secured on a concrete clock which has one degree of freedom – is subjected to axial tensile force. Dilation is so sufficient that during the study there was no slide of the rock bolt from the borehole and a rupture of the thread followed (Fig. 11).

Fig. 11. View of rod surface after rupture (*photo: K. Skrzypkowski*)

Load speed, v – is the ratio of the displacement Δl or Δlu recorded during the test period:

$$v = \frac{\Delta l}{t}, \text{ m/s} \tag{1}$$

Maximum tensile stress, σr – is the ratio of maximum breaking force F, impacting the smallest cross sectional area of the bolt rod (Osinski et al., 1975):

$$\sigma_r = \frac{F}{\frac{\pi \cdot d_3^2}{4}}, \text{ MPa} \tag{2}$$

Relative deformation of rock bolt, ε – is the ratio value of total displacement (breaking elongation) of bolt rod Δl in relation to the active length of the rod bolt l_0. Relative deformation is the sum of deformation of the rolled part and the thread of the rod:

$$\varepsilon = \frac{\Delta l}{l_0} \cdot 100, \text{ \%} \tag{3}$$

Relative deformation of the bolt rod, $\varepsilon 1$, $\varepsilon 2$, $\varepsilon 3$ (%) – is the value of bolt rod deformation in the scope of resilience and linear interpreted from an electro-resistant strain gauge attached to the rolled surface (d) of the rod.

Examples of the characteristic parameters obtained in the tests are shown in Table 4 and in Figure 12. The variation scope of breaking force F, ranged from 160.3 to 164.5 kN. Relative deformation ε, ranged from 5.7% to 6.7%. Static load speed was in the range of 0.00089 m/s to 0.00094 m/s

TABLE 4

Technical characteristics of the rock bolt support expanded statically

Line	Parameter		Unit	Value
1.	Bolt rod length, l		mm	1825
2.	Active bolt rod length, l_0		mm	1670
3.	Bolt rod diameter, d		mm	18,23
4.	Bolt rod diameter, d_3		mm	16,43
5.	Breaking force, F		kN	160,382
6.	Displacement, Δl		mm	109,728
7.	Tensile stress time to moment of rock bolt rupture, t		s	117,14
8.	Load speed, v		m/s	0,00094
9.	Maximum tensile stress, σ_r		MPa	756, 468
10.	Relative deformation, ε		%	6,571
13.	Force corresponding to the deformation of the electro-resistant strain gauge $\varepsilon 1$, $\varepsilon 2$, $\varepsilon 3$, at the level of 2‰, F_s	$F_{s1} = F_{s2}$	kN	98,040
		F_{s3}		100,003
14.	Displacement corresponding to electro-resistant strain gauge $\varepsilon 1$, $\varepsilon 2$, $\varepsilon 3$, at 2‰ level, Δl	$\Delta l_1 = \Delta l_2$	mm	15,820
		Δl_3		16,020
15.	Tensile stress time in the scope of elasticity, t_s	$t_{s1} = t_{s2}$	s	55,80
		t_{s3}		56,86

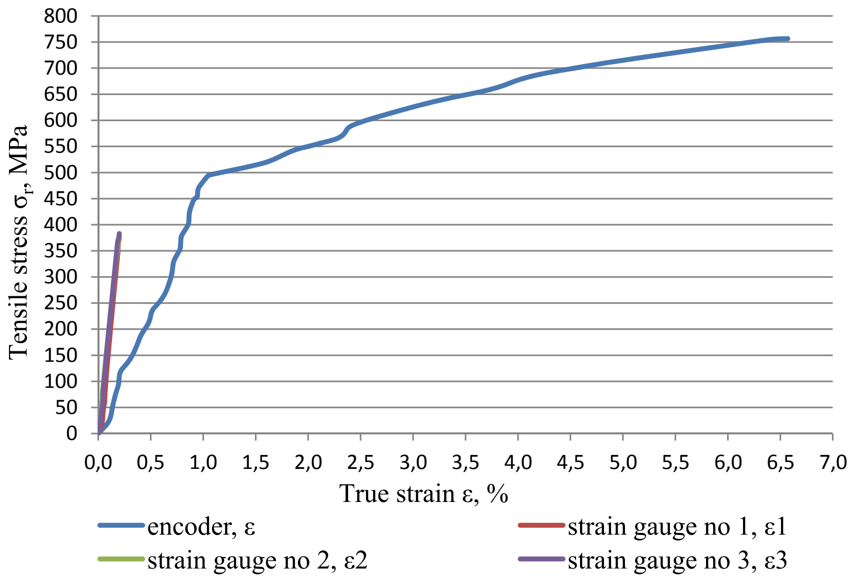


Fig. 12. Stress and deformation characteristics of statically tensioned rock bolts

5. Summary

The modern laboratory test facility at which the rock bolt support study was carried out, has many advantages, which include:

- the opportunity to study rock bolts in a 1:1 scale with rods of up to 6m in length, which enables the study of not only bar rods, but also cable, string pipe type of bolts, made of any given material and different ways of fixing within the rock mass,
- the opportunity to study mining rock bolt, as well as geotechnical and construction types, through variable values as well as static and dynamic load programs;
- the opportunity to apply loads in three ranges: 400 kN, 800 kN and 1,600 kN.

In analyzing the course of characteristics shown in Fig. 12, it can be concluded that they differ from each other in both the current values and the shape of the curves. Deformations: $\varepsilon 1$, $\varepsilon 2$ and $\varepsilon 3$ are based on measurements made using the electro-resistant strain gauges attached directly to the surface of the steel rod (properly prepared beforehand) and a relatively short measurement database, which in this case was 10 mm, placed centrally along the entire length of the rod at distances of every 400 mm. Deformations of individual strain gauges are therefore accepted as equivalent of the deformation of the steel rod (bolt rod). From the graph it appears that all three courses practically coincide, with uniformly increasing deformation to a value of less than 0.2% of the base length, and load range to the tension value of about 370 MPa, which corresponds to the axial force value of 100 kN influencing the rock bolt. In certain experiments, the characteristic limits of stress and deformation of the bolt rods have a linear course.

The stress-strain characteristics, based on measurements carried out on the basis of a longer measurement base using a precise cable encoder, have a separate course because this measurement, apart from the deformation of the rod itself, takes into account the deformation of the remaining elements of the bolt rods. In this case we are talking about the characteristics of the rock bolt and not the bolt rod. In the course of deformation function relative to the force in operation, two distinctive parts can be distinguished: the first is in the range of up to 490 MPa stress and a strain of up to about 1%, and the second, at a stress of 490 to 756 MPa and a much larger deformation of up to about 6.5%. Furthermore, the second phase is characterized by a faster growth of deformation attributable to a specific force (or stress) in relation to the initial part of the characteristics. Rock bolt loads greater than 100 kN cause increasingly rapid deformation of the structural elements of the rock bolt, in particular the bolt washers, but also the threaded part together with the nut. Certain parts of displacement recorded by the encoder also result from slight displacement of the rock bolt head.

Referring to the results of actual operating conditions of the rock bolt, it can be concluded that only about 3.5% of the total deformation of all rock bolt components and displacement is attributable to deformation of the rock bolt bar itself.

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