ENGINEERING TRANSACTIONS • *Engng. Trans.* • **64**, 1, 89–103, 2016
Polish Academy of Sciences • Institute of Fundamental Technological Research (IPPT PAN)
National Engineering School of Metz (ENIM) • Poznan University of Technology

Anchoring Principles of a New Energy-Absorbing Expandable Rock Bolt

Zhijie WEN[1,2], Yujing JIANG[2], Zuozhen HAN[2],
Sitong YANG[2], Xiao WANG[2]

[1] *Research Center of Geotechnical and Structural Engineering*
Shandong University
Jinan, Shandong 250061, P R China
e-mail: sdust0532@gmail.com

[2] *State Key Laboratory of Mining Disaster Prevention and Control Co-founded by Shandong Province and the Ministry of Science and Technology*
Shandong University of Science and Technology
Qingdao, Shandong 266590, P R China

The greater the mine, the harder stability control will be. And the conventional rock bolts do not adapt well to the severe rock stress conditions. An ideal bolt having a high resistance and large deformation should be developed. Based on the test results and theoretical study, this paper proposes an energy-absorbing expandable rock bolt, which consists of the bar, sleeve, bolt plate, nut, and bolt end. The anchoring mechanism and its efficiency were systematically analysed in the laboratory and in practice: the anchoring mechanism and supporting density, especially the quantitative relationship, were deduced under the Energy Balance Theory, that is, $E_B = \frac{1}{2} \cdot n \cdot F_0\ (u_0 + 2\Delta u)$. As Compared with the conventional bolt and large deformation bolts, the new type of bolt could provide a larger constant resistance, even in the soft rock roadway with large squeezing deformation, the pulling force can be achieved by $F = A \cdot \sigma \cdot f_2$, it mainly being generated by a normal stress acting on the pore surface. These characteristics are helpful in making the supported roadway safe. The amount of released energy during the large deforming process of the surrounding rock is expressed through conservation of energy, which can provide reference to the quantitative calculation of the bolt supporting system.

Key words: rock bolt, energy-absorbing, large deformation, constant resistance, action mechanism.

1. Introduction

With the increase of mining depth, the mining conditions become complicated, and the in-situ stress becomes greater, which changes the mechanical properties of rock mass (e.g., structure, strength) [1–4]. Under these conditions, the surrounding rock of the roadway is characteristic of obvious soft rock properties, more broken structures, and larger deformations [5]. So it is faced with some engineering problems, including immense repair work, high maintenance costs, and some safety problems, which threaten the safety and high efficiency of deep resources [6, 7].

In China, the total length of the mine roadway is up to 50 km, 60% of which is weak or broken roadway. The deformation of surrounding rock is severe, up to 500–1000 mm totally, under the condition of high in-situ stress and the effects of underground coal mining [8]. The big deformation seriously affects the efficiency of the conventional rock bolts. Aimed at the stability control problem of the roadway, the first author invented a new energy-absorbing expandable rock bolt in the year 2012. The new bolt can provide large constant resistance by the way of anchoring the action of the bolt end instead of resin. And the resistances provided by the new bolt can avoid the conventional bolt failure induced by the non-uniform deformation in the roadway. It can provide a steady roadway.

2. Conventional rock bolts and their performance

As compared to the other supporting devices (e.g., wooden support, shotcrete support, hydraulic prop support, and steel arch support), rock bolt has its own obviously technical and economic advantages in the engineering field of roadway supports [9, 10]. It has become one of the main supporting ways in the mine roadway, geotechnical engineering, and other underground engineering. There are many kinds of rock bolts that have their own advantages and disadvantages. For the cement grouted rock bolts, the cement cartridge must be immersed in water before use [11]. And its anchoring strength depends on the duration of it being immersed in water. Its construction technology is complex. A normal expansion shell bolt has the disadvantages of a difficult installation, easy corrosion for the bolt body, unreliable anchoring force, and a matching problem between the steel tensile stress and the anchoring force [12–14]. The resin grouted bolts have the disadvantages of a complicated processing technology for the bolt body, high price, and not being suitable for surrounding rock with a high stress [15, 16].

The conventional bolts in a coal mine roadway will fail when the elongation is up to 18%. The relationship between the working resistance and roadway deformation in the deformation process of surrounding rock for conventional bolts and ideal bolts is shown in Fig. 1.

The large deformation in underground roadway supporting is good. It can release the deformation energy of the roadway surrounding rock that is helpful to the stability of roadway. So the researchers design the bolts with a large deformation. When the load on the bolts reaches the tolerance limit, the bolts will extend through the coupling device that controls the tensile elongation,

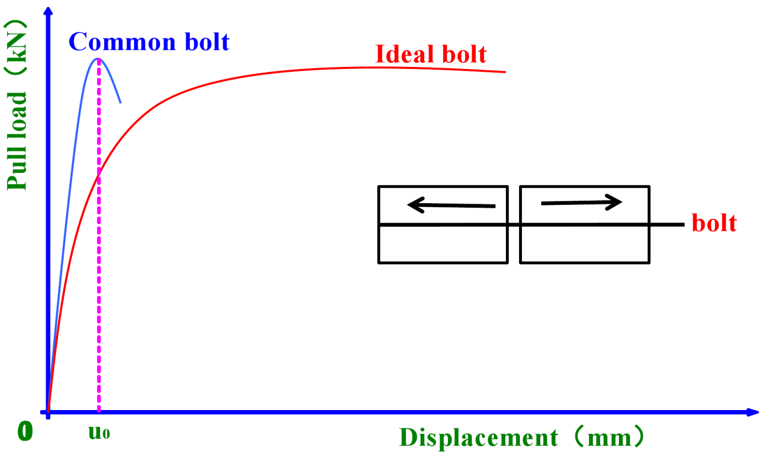


FIG. 1. Load-displacement curves of the common bolt and ideal bolt under pull loading.

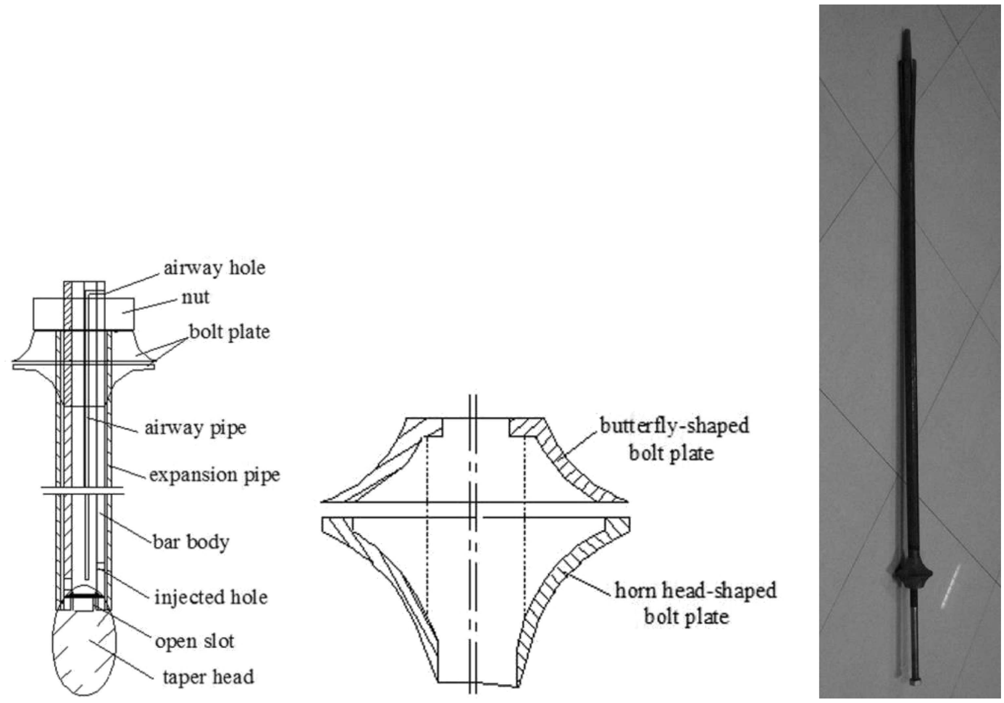
allow the surrounding rock to move slightly, decrease the surrounding rock stress, and establish a new equilibrium state in the roadway. An energy-absorbing bolt invented in the United States consists of a steel bar with threads, sleeve, and a special connection. The outer diameter of steel bar is much bigger than that of inner diameter of the steel bar. So the prestressing force can be loaded by tightening the nut. When the bolt load reaches the predetermined value, and the steel bar moves into the sleeve to a certain distance, the threads on the bolt can partly fail. A MCB33-type of cone bolt, first used in Canada, is a non-constant resistance bolt and can only accommodate maximum deformation of 120 mm, which is not large enough for rock masses with a large deformation condition. A bolt called Roofex, which first appeared in Australia, is made of steel bar. The steel bar is coated with a layer of smooth plastic and fixed into the borehole by using either cement or epoxy resin. This bolt could bear the constant load of 80 kN and withstand the maximum deformation of 300 mm, so it is suitable for a soft-rock roadway.

3. Structure and anchoring principle of the new bolt

3.1. Mechanical properties of the new bolt structure

The new energy-absorbing expandable rock bolt consists of bar body, sleeve, bolt plate, nut, and bolt end. The bar body is connected to the cone-shaped bolt end with two injected holes. The new bolt is shown in Fig. 2.

The bolt discussed here, as compared to the common pre-stressed bolt, has some main characteristics of yielding with resistance, resistance with yielding, and constant resistance with no failure. According to the wedged mechanics



a) sketch map of rock bolt b) sketch map of bolt plate c) photo of the new bolt

FIG. 2. Layout of the new type of energy absorbing rock blot.

principle, the bolt end and the bolt plate is connected by the pre-stressing steel strand. The bolt is fixed in the borehole using the bolt end. When the relative movement between the bolt end and the sleeve occurs, the bolt end exerts compressive stress and friction force on the sleeve and the sleeve expands. The bearing capacity of the bolt is from the bond shear stress induced by the action between the sleeve and the wall of the hole. Due to the expansion of the sleeve, the radial pressure on the hole's wall is generated. When the bar moves along the sleeve, the action of the radial pressure will generate a great frictional resistance. Then, the bond shear stress will firmly fix the bolt into the surrounding rock, which increases the interaction between the bolt and the surrounding rock.

After roadway excavation and installation of the new bolt, it will bear the lateral force induced by the deformation of surrounding rock under the mechanical conditions of gravity and tectonic stress, and will be squeezed by this lateral force. During the deformation, the lateral force on the bolt is transferred to the bolt end as friction force, meanwhile, the reactive force generated by the bolt end is applied on the surrounding rock. It is called bond shear stress on the surrounding rock. If the tensile force supplied by the surrounding rock is greater than the friction force between the bolt end and the sleeve, a compatible deformation between the bolt end and the surrounding rock will occur with the constant compressive stress between the bolt end and the sleeve. Then, the bolt could continuously supply constant working resistance with a large deformation. So, the bolt has some characteristics of increasing the anchoring force with the increase of the gravity and tectonic stress, high prestress, constant anchoring force with the same geological conditions, yielding with the bar slippage, which is helpful to be applied to a broad range, especially in the roadway, tunnels, and rock slope with large deformation. Meanwhile, for the fractured rock mass, the surrounding rock could be grouted by an injected hole on the new bolt.

3.2. Installation of the new bolt

The installation process is as follows. After drilling in the surrounding rock, the bar is present in the sleeve. Then the bolt plate is set on the bar and temporarily fixed by the nut. Finally, this composite structure is put into the drilling hole, then the nut is tightened. The expansive force generated in the relative displacement between the bolt end and the sleeve generates the friction force between the sleeve and the hole wall, which is called the tensile force. At this moment, the supporting action of the bolt starts to work.

3.3. Anchoring principle and its performance

The theory of elastic mechanics, developed on the basis of small deformations, is not appropriate to solve large discontinuous deformation problems in the geotechnical engineering. For the convenient calculation analysis, Newton's Third Law of Motion is employed to analyse the anchoring principle of the new bolt. The analysis is based on the following three assumptions:

- the sleeve and the surrounding rock mass are considered as linear elastic materials,
- the interaction between the sleeve and the surrounding rock mass satisfies Mohr-Coulomb criteria,
- the distribution of the axial stress along the sleeve is uniform.

3.3.1. Mechanical balance equation. According to the anchoring principle of the bolt, before its installation, the hole should be set into the appropriate position, then the bolt is put into the hole. The bolt end should be very close to the sleeve. Then, the sleeve is also tightly bound to the borehole, which could be considered as an anchorage section. The pre-designed friction force between the bolt end and the sleeve supplies the function of constant supporting resistance. This bolt deformation sketch is shown in Fig. 3. The anchorage section is simplified into the mechanical model shown in Fig. 4. According to the limit equilibrium theory, the tensile force (F) is calculated from Eq. (3.1),

$$F = F_2 \cdot S = \sigma \cdot f_2 \cdot S = \pi D l_0 \cdot \sigma \cdot f_2 = A \cdot \sigma \cdot f_2, \tag{3.1}$$

where D (m) is the diameter of the bolt end, f_2 is the friction factor between the steel bar and the sleeve, l_0 (m) is the contact length between the sleeve and the bolt end, σ is the normal stress between the steel bar and the sleeve.

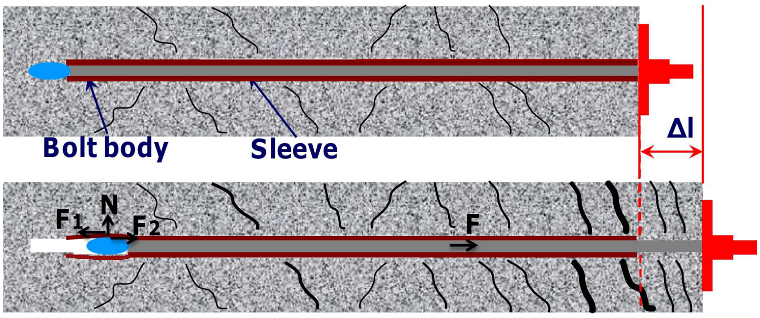


FIG. 3. Sketch illustrating the loads induced in the bolt when it is subjected to a large deformation of the surrounding rock.

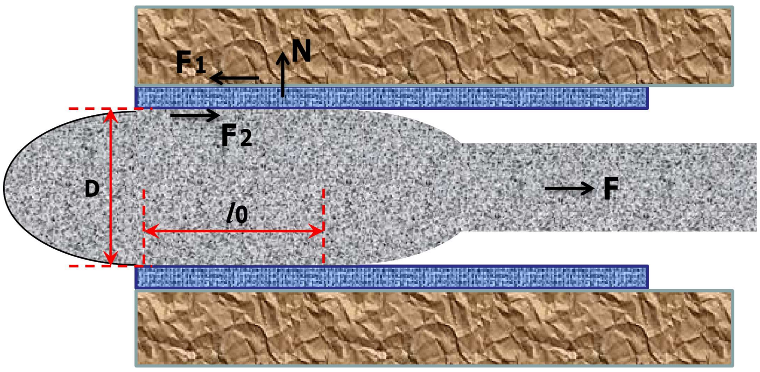


FIG. 4. Diagram of the mechanical action of the bolt end.

The friction force (F_1), the normal stress (σ), and the shear stress between the sleeve and borehole (τ) are calculated from Eqs. (3.2)–(3.4), respectively:

$$F_1 = \sigma \cdot f_1 \cdot S = \pi D l_0 \cdot \sigma \cdot f_1 = A \cdot \sigma \cdot f_1 = \frac{F \cdot f_1}{f_2} = \tau \cdot S = \pi D l_0 \cdot \tau, \tag{3.2}$$

$$\sigma = \frac{F}{S \cdot f_2}, \tag{3.3}$$

$$\tau = \frac{F \cdot f_1}{f_2 \cdot \pi D l_0}. \tag{3.4}$$

where τ is the shear stress between the borehole and sleeve.

When the axial force of the bolt is constant, the strain on the bolt end is calculated from Eq. (3.5),

$$\varepsilon = \frac{\sigma}{E} = \frac{F \cdot f_1}{f_2 \cdot \pi D l_0}. \tag{3.5}$$

3.3.2. Supporting system From the viewpoint of energy, the load from the external force to the rock bolt causes energy transfer. The amount of energy is equal to the product of the load multiplied by the displacement of the bolt end. The load to the rock bolt is eventually transferred to the anchor head, and becomes pressure to the sleeve. The sleeve then bears pressure, and frictional work appears between the anchor head and the sleeve. Furthermore, there is displacement between the sleeve and host rock, so a part of the load is transferred to the host rock and produces frictional work between the sleeve and the host rock. Meanwhile, parts of energy are transferred and accumulated to the host rock, and large deformation of the host rock takes place. In the equilibrium state of the rock bolt under external force, the energy transfer includes deformation energy in the sleeve, host rock, and the bolt supporting the structures. The energy E is calculated from Eq. (3.6),

$$E = E^{\text{bolt}} + E^{\text{rock-coal}} + E^{\text{deformation}}. \tag{3.6}$$

The bolt is in the state of equilibrium, so the work done by the applied load is totally transferred into the deformation energy of the bolt and other supporting structures, as shown in Fig. 5.

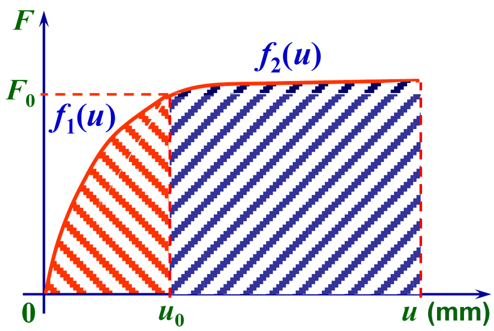


Fig. 5. Load-displacement curves of rock bolts.

n is the supporting density of the bolt that means the amount of bolts per square meter, then the work done by the applied load is calculated from Eq. (3.7)

$$E_B = E^{\text{bolt}} + E^{\text{deformation}} = n \cdot \int_0^{u_0} f_1(u)du + n \cdot \int_{u_0}^{u} f_2(u)du = \frac{1}{2} \cdot n \cdot F_0(u_0 + 2\Delta u). \tag{3.7}$$

4. Laboratory tests

To test the maximum constant resistance of the new energy-absorbing expandable rock bolt, a testing system of bolt static mechanical properties was designed in the Geotechnical centre of Shandong University, see Fig. 6. The basic parameters of the testing system were: the maximum load of 460 kN, maximum displacement range of 800 mm, loading rate of 0.1–16 kN/min, displacement rate of 0.5–80 mm/min.

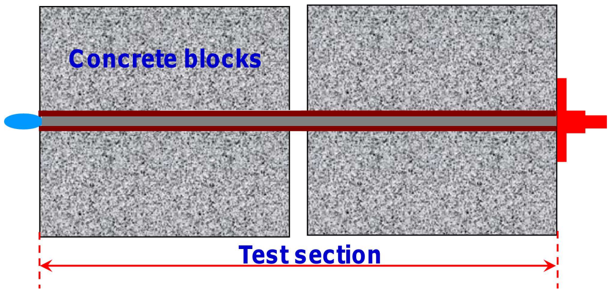


Fig. 6. Bolt test rig for a full-scale pull test of rock bolts.

With the displacement control method, the tensile test of the large deformation bolt was tested by using the testing system. The parameters of the bolt were the bolt length of 2200 mm, sleeve diameter of 32 mm, bolt end diameter of 32 mm, steel bar diameter of 20 mm. And it had two injected holes on the cone-shaped bolt end. The detailed parameters are shown in Table 1. The test results for three samples are summarised in Fig. 7.

Table 1. Samples for the pull tests.

Sample	Ultimate load [kN]	Diameter of sleeve [mm]	Diameter of cone-shaped body [mm]	Diameter of bolt	Grouting
S 1	278.2	32	32	20	Yes
S 2	267.6	32	32	20	No
S 3	272.7	32	32	20	No

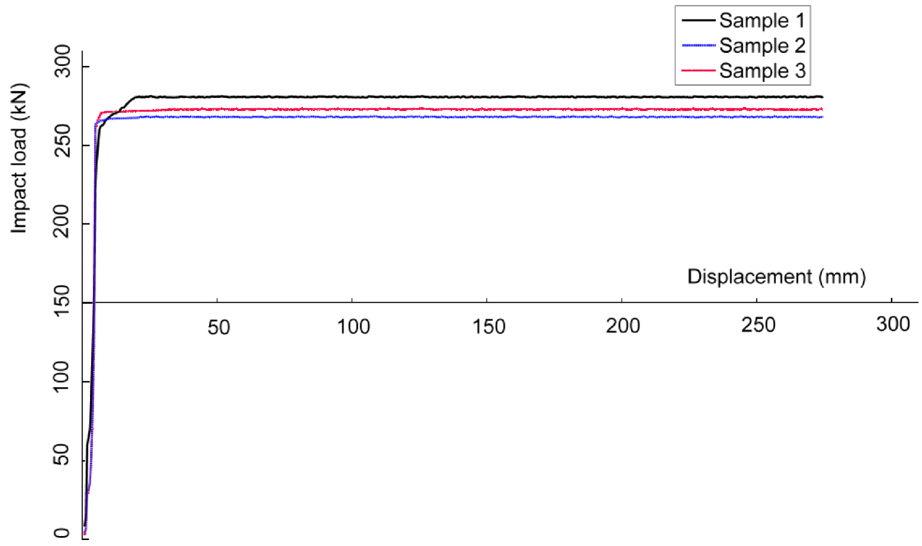


FIG. 7. Pull test results of three samples.

As it can be seen from the Fig. 8, the anchoring process of the new type of energy-absorbing bolt can be divided into three stages, which are: the elastic, elastic-plastic, and slip ones. Especially at the third stage, the conventional rock bolt could be failure due to a large deformation, which is the point of innovation of this bolt. It can keep the same anchoring force when a large deformation happens.

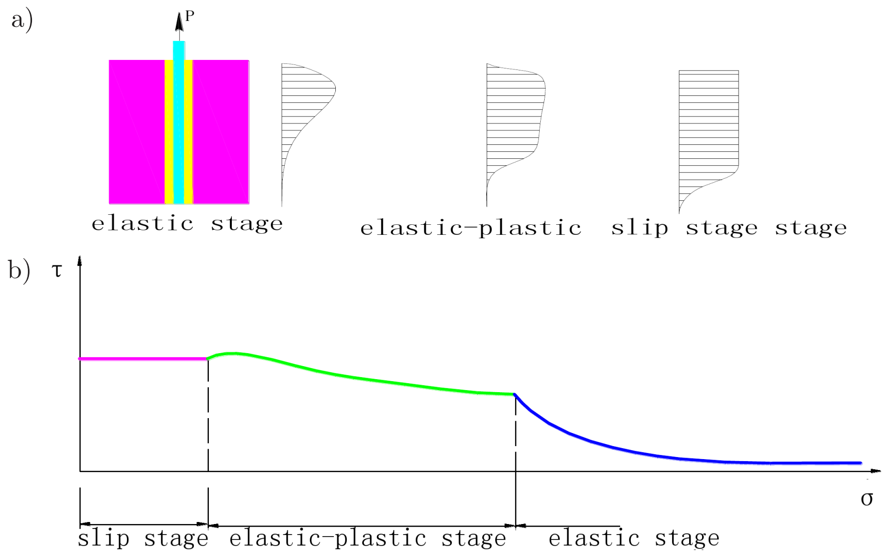


FIG. 8. Shear stress distribution under a different pulling stage.

5. Field texts

5.1. Comparison of the supporting effect

The field tests were carried out to examine the new energy-absorbing expandable rock bolt installed in a high-stress and broken roadway of a deep mine in the Eastern China. To compare the supporting effects between the supporting scheme with common bolts and a similar supporting scheme with the new bolts, both schemes were used in the same roadway with the length of 50 m, respectively. The Canadian SSI prone equipment [5] and Pulse Ekko100 [5] geological radar were used to detect the broken rock zone in the roadway. The detection results are shown in Table 2.

Table 2. Size of the broken rock zone in the surrounding rock (All data are derived from reference [5]).

Common bolt			New bolt		
Measuring zone No.	Test No.	Failure zone [m]	Measuring zone No.	Test No.	Failure zone [m]
A	1	2.2	A	1	0.3
	2	2.0		2	0.3
	3	2.1		3	0.4
	average	2.06		average	0.33
B	1	2.0	B	1	0.1
	2	2.1		2	0.2
	3	2.2		3	0.3
	average	2.06		average	0.2
C	1	2.2	C	1	0.3
	2	2.3		2	0.3
	3	2.3		3	0.2
	average	2.26		average	0.27
D	1	2.3	D	1	0.4
	2	2.2		2	0.3
	3	2.3		3	0.2
	average	2.26		average	0.25
Average surrounding rock deformation: 150 mm Bolt parameters: length: 2.2 m, ultimate load: 60 kN Diameter of the bolt: 20 mm			Average surrounding deformation: 30 mm Bolt parameters: length: 2.4 m, ultimate load: 118 kN Diameter of the cone-shaped body: 32 mm Diameter of the bolt: 20 mm		

The initial supporting parameters for the roadway were 2200 mm length rebar bolt with a diameter of 20 mm and two resin cartridges with the bolt spacing of 800×900 mm. The size of the broken rock zone was 2.26 m at the initial stage of the rock support. Eight days later, the working load was up to 60 kN, with a deformation of 30 mm. The conventional grouted or end anchored rock bolts were broken at this stage. The roadway deformed quickly until failure.

The modified supporting parameters for the same roadway, mentioned above, were the new 2200 mm bolt with a diameter of 20 mm and the bolt spacing of 800×800 mm. The size of the broken rock zone was 0.03 m. Three days later, the load was 118 kN. The stability of the roadway has significantly improved.

5.2. Releasing energy in the deformation process of the surrounding rock

During the loading process, the relative movement between the sleeve and the borehole is quite small, so the deformation energy induced by the relative movement is ignored. So, the change in energy mainly includes the potential energy increment of the supporting structure E^{rock} and work done by the tensile force on the bolt E^{bolt}.

The total potential energy of the supporting structures is given by $E = E^{\text{rock}} + E^{\text{bolt}}$, where E^{rock} is the potential energy increment induced by the deformation of the structure, E^{bolt} is the potential energy increment from the work done by the support body.

E_1 stands for the releasing energy in the deformation process for the initial supporting scheme. E_2 is the energy equation of the supporting system during the deformation process.

For the modified scheme, the density of the bolt is $n = 1.6 \frac{\text{bolts}}{\text{m}^2}$, $\Delta u = 0.03$ m, $u_0 = 0.01$ m.

The deformation energy per unit area from the releasing of rock mass ΔE is given by

$$\Delta E = E^{\text{rock}} + E^{\text{bolt}} = E_1 - E_2 = \frac{1}{2} \cdot n \cdot F_0(u_0 + 2\Delta u) = 6608 \text{ J}.$$

According to Eq. (3.6), the working resistance of the bolt is proportional to the density of bolts and deformation of the surrounding rock, namely, $\propto (nu)$. So, the new bolt with grouting could reduce the deformation of the surrounding rock and reinforce the stability of the roadway, then, it is helpful to decrease the supporting density of bolts and improve the stability of the roadway.

6. Discussion

6.1. Laboratory test

To compare the mechanical properties of the new bolt, with or without grouting, three test samples were performed. For the No. 1 sample with grouting on the bolt end, its working resistance was up to 278.2 kN, when the deformation of the bolt end was 20 mm. For the No. 2 and No. 3 samples without grouting, their working resistances were 267.6 kN and 272.7 kN, respectively, when the deformation of the bolt end was 10 mm, as shown in Fig. 7.

A comparative analysis of the working performance among several bolts is shown in Fig. 9.

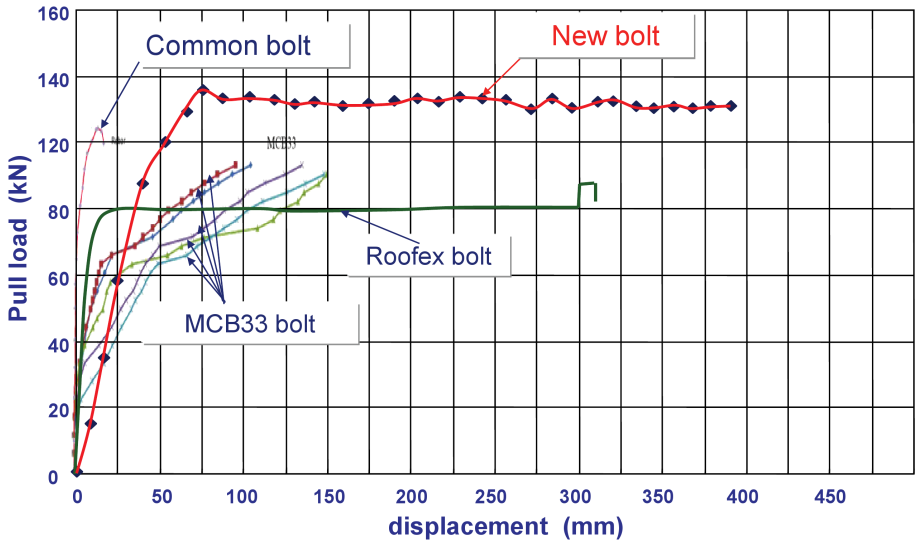


Fig. 9. Load-displacement curves of different rock bolts under pull loading (All data, except for those of the new bolt, are redrawn from [6]).

It should be noted from the load-displacement curves of different rock bolts that the new bolt has a stronger energy-absorption capacity. As compared with the common bolts, the new bolt could withstand a larger deformation without the supporting failure problem. As compared with the Roofex bolt and the MCB33 bolt, the new bolt has a 130 kN working resistance. The Roofex bolt and MCB33 bolt are only 80 kN and 110 kN, respectively. And the appropriate bolt end should satisfy a larger deformation. There is a small fluctuation of curve, because of the discontinuous relative displacement between the bolt end and the sleeve.

6.2. Field test

The field observations are shown in Fig. 10. The supporting resistance increment of the common bolts is good. Three days after installation, it reaches 58 kN. It means, they can reach working resistance quickly. But, eight days after installation, the roadway is destructed because of the small resistance.

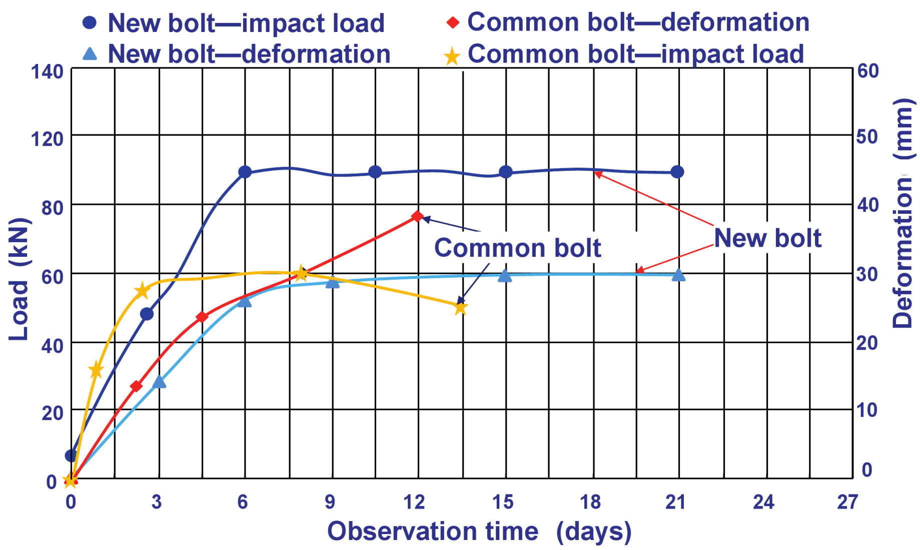


Fig. 10. Load-time curve of bolts from field test (All data, except those of the new bolt, are redrawn from [6]).

The supporting resistance increment of the new bolt is not as good as the common bolts. But, six days after installation, it reaches 118 kN. The deformation of the surrounding rock is only 30 mm, and the stability of the roadway is good. This means that the new bolt can adopt more energy in the deformation process of the surrounding rock, which is helpful in solving the large deformation problem of roadways.

7. Conclusions

As compared with the conventional bolt, the new bolt that consists of a bar body, sleeve, bolt plate, nut, and bolt end, could provide a larger constant resistance in the soft rock roadway even with a large squeezing deformation through the anchoring way of the bolt end itself rather than using the cement grout or resin. The test results showed that the new bolt could bear 278.2 kN with the ultimate elongation of 400 mm. And the field tests showed, in the same condition of the roadway, that the supporting scheme with the conventional bolts failed but the roadway supported by the new bolts was steady and deformed only 30 mm. At the same time, based on the principle of energy balance, this paper proposed the calculation of the releasing energy during a large deformation of the surrounding rock, which could provide a reference for calculating the roadway supporting from the qualitative calculation stage to the quantitative calculation stage.

Acknowledgment

This work is supported by the National Basic Research Program of China under Grant No. 012CB72310402, the National Natural Science Foundation of China under Grant No. 51244010 and No. 51304126, and the Science Research Innovative Group of College of Resources and Environmental Engineering of SDUST No. 2012ZHTD06; New Teachers' Fund for Doctor Stations of Ministry of Education under Grant No. 20123718120009; Open Project of State Key Laboratory Breeding Base for Mining Disaster Prevention and Control (Shandong University of Science and Technology) (MDPC2012ZR01); the research fund for excellent young and middle-aged scientists of Shandong Province under Grant No. BS2013NJ007; Fok Ying Tung Education Foundation under Grant No. 141046; China Postdoctoral Science Foundation under Grant No. 2013M541918. Shan Dong University of Science and Technology Outstanding Young Investigator Award No. 2014JQJH105.

References

1. Li C.C., *A new energy-absorbing bolt for rock support in high stress rock masses*, International Journal of Rock Mechanics and Mining Sciences, **47**(3): 396–404, 2010.
2. Li C.C., *Performance of D-bolts under static loading*, Rock Mechanics and Rock Engineering **45**(2): 183–192, 2012.
3. Wen Z.J., Qu G.L., Wen J.H., Shi Y.K., Jia C.Y., *Deformation failure characteristics of coal body and mining induced stress evolution law*, The Scientific World Journal **2014** (5), Article ID 714507, 2014.
4. Ansell A., *Laboratory testing of a new type of energy absorbing rock bolt*, Tunnelling and Underground Space Technology **20**(4): 291–300, 2005.
5. Wen Z.J., *Study on the stability of roadway in Dongda coal mine* [in Chinese], Shandong University of Science and Technology Report, pp. 56–89, 2012.
6. Wang J., *Large deformation with constant-resistance supporting mechanism and its application of deep rock tunnel in Tangkou Coal Mine* [in Chinese], Doctoral Thesis, China University of Mining and Technology, 2010.
7. Wen Z.J., *Study of stress features of fully grouted prestressed anchors*, Rock and Soil Mechanics, **31**: 177–181, 2010.
8. Ansell A., *Dynamic loaded rock reinforcement*, Doctoral Thesis, Bulletin 52, Dept. of Structural Engineering, Royal Institute of Technology, Stockholm, 1999.
9. Gaudreau D., *Performance assessment of tendon support systems submitted to dynamic loading*, Master Thesis, Ecole Polytechnique de Montreal, 2004.
10. StacEY T.R., Ortlepp W.D., *Support appropriate for dynamic loading and large static loading in block café mining openings*, Procc. MassMin 2000, pp. 783–789, Brisbane, Qld, 2000.

11. Wu R., *Development of a new yielding rock bolt-Yield-Lock bolt*, [in:] The 44th US rock mechanics symposium, pp.10–197, Salt Lake City, USA, ARMA, 2010.

12. Amusin B., *Ground control and design of deep mine openings*, International Journal of Rock Mechanics and Mining Sciences **35**(4–5): 1365–1609, 1998.

13. Ferrero A.M., *The shear strength of reinforced rock joints*, International Journal of Rock Mechanics and Mining Sciences & Geomechanics Abstracts **32**(6): 595–605, 1995.

14. Li C., Stillborg B., *Analytical models for rock bolts*, International Journal of Rock Mechanics and Mining Sciences & Geomechanics Abstracts, **36**(8): 1013–1029, 1999.

15. Serrano A., Olalla C., *Tensile resistance of rock anchors*, International Journal of Rock Mechanics and Mining Sciences & Geomechanics Abstracts **36**(4): 449–474, 1999.

16. Gálvez J.C., Elices M., Olivares A., *Damage tolerance of an anchor head in a post-tensioning anchorage system*, Engineering Failure Analysis, **13**(2): 235–246, 2006.

Received January 8, 2015; accepted version July 5, 2015.