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NATURAL FILLING AND SYSTEMATIC ROOF CONTROL TECHNOLOGY FOR GOB-SIDE ENTRY RETAINING IN STEEP COAL SEAMS

NATURALNE PODSADZANIE STROPU I METODA SYSTEMOWEGO PROWADZENIA STROPU PRZY UTRZYMANIU CHODNIKÓW PRZEWOZOWYCH OD STRONY ZROBÓW W NACHYLONYCH POKŁADACH WĘGLA

The technology for gob-side entry retaining in steep coal seams is still in the development stage. The analysis results of the caving structure of main roof, low influence of gateway's stability because of long filling distance and weak dynamic effect of the gateway, and the low stress redistribution environment indicate that using this technology in steep coal seams has significant advantages. Moreover, to reinforce the waste rock and the soft floor and to better guard against the impact of the waste rock during natural filling, a rock blocking device and grouting reinforcement method were invented, and theoretical calculations result show that the blocking device has high safety factor. In addition, we also developed a set of hydraulic support devices for use in the strengthening support zone. Furthermore, because the retaining gateway was a systematic project, the selection of the size and shape of the gateway cross section and its support method during the initial driving stage is a key step. Thus, first, a section the size of bottom width and roof height of a new gateway was determined to meet any related requirements. Then, according to the cross sections of 75 statistical gateways and the support technique, it chosen a trapezoidal cross section when the dip of the coal seam is $35° < \alpha \leq 45°$, a special and an inclined arch cross section when $45° < \alpha \leq 55°$. Eventually, a support system of bolts and cables combined with steel mesh and steel belts was provided. The support system used optimized material and improved parameters, can enhanced the self-bearing ability of the surrounding coal and rock masses.

Keywords: gob-side entry retaining; steep coal seams; natural filling; systematic project

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Technologia utrzymywania chodników w obszarze zrobów w nachylonych pokładach węgla jest nadal rozwijana i udoskonalana. Jej zastosowanie prowadzi do zawału głównego stropu, który jednak w nieznacznym tylko stopniu wpływa na stabilność chodników z uwagi na odległość obszaru podsadzania, podczas gdy oddziaływania dynamiczne na chodniki przewozowe będą niewielkie. Powstały rozkład naprężeń wskazuje, że zastosowanie tej technologii w stromych pokładach przyniesie znaczne korzyści. Ponadto, w celu wzmocnienia warstw skał płonnych i miękkich warstw spągowych, a także dla lepszego zabezpieczenia przed skutkami ruchów skał płonnych w trakcie podsadzania, opracowano urządzenia blokujące ruch skał wraz ze wzmocnieniem cementowym. Obliczenia teoretyczne wskazują że zastosowana blokada ruchów skał charakteryzuje się wysokim wskaźnikiem bezpieczeństwa. Ponadto, opracowano także zestaw wsporników hydraulicznych dla dodatkowego wzmocnienia strefy podsadzania. Z uwagi na to, że zachowanie chodnika przewozowego jest działaniem stałym i systematycznym, dobór wymiarów i kształtu przekroju chodnika oraz metody jego stabilizacji jest sprawą kluczową już na etapie drążenia chodnika. W kroku pierwszym określono więc szerokość chodnika w jego dolnej części oraz wysokość stropu zgodnie z odpowiednimi wymogami. Następnie w oparciu o wymiary przekrojów 75 statystycznych chodników oraz uwzględniając dostępne techniki stabilizacji stropu wybrano przekrój trapezoidalny gdy nachylenie pokładu węgla mieści się w przedziale $35° < \alpha \le 45°$, zaś dla kątów nachylenia w przedziale $34° < \alpha \le 55°$ wybrano nachylony profil łukowy. W etapie końcowym zastosowano układ stabilizujący oparty na kotwach i kablach połączonych siatka stalową i stalowymi taśmami. W systemie stabilizującym wykorzystano zoptymalizowane materiały zapewniając lepsze parametry pracy, co korzystanie wpłynie na nośność warstw górotworu w otoczeniu pokładu węgla.

Słowa kluczowe: utrzymanie chodnika przewozowego, nachylone pokłady węgla, naturalne podsadzanie, projekt systemowy

1. Introduction

In gob-side entry retaining in underground coal seams, the headgate of the current mining panel is retained to serve as the tailgate of the subsequent adjacent panel. Due to its many advantages, such as effectively increasing the coal recovery rate, reducing the roadway development rate, and mitigating the risk of outbursts (Wang et al., 2015b; Yang et al., 2015), gob-side entry retaining (GER) technology has been widely used in coal mines with complex geological conditions. In regards to this technology, previous studies showed that the dip of the coal seam was the most important geological factor (Yang et al., 2016a; Zhang et al., 2015) because the dip angle affects the difficulty of implementing GER. Thus, because of the limitation of the technique for controlling the surrounding rock, the vast majority of GER cases have focused on flat and gently inclined coal seams with dips of less than 25° (Cao et al., 2012; Li et al., 2016; Tan et al., 2015; Yang et al., 2013). In addition, no pillar is needed for the retained entry, but rather a filled wall on the gob side is constructed, usually by one of five materials: high-water material (HWM), concrete, paste, blocks, or waste pack (Tan et al., 2015).

Gradually, people found that caved in waste rock will automatically slide or roll to the low end of the working face in the gob of a steep coal seam with a dip of 35° to 55° according to the natural repose angle of the caved waste rock (Zhang et al., 2015). Then, they realized that the caved waste rock could be used as filling material in steep coal seams. In addition, if the caved waste rock can be used up gob side filling material, it will not only decrease the cost and labor intensity, but will also meet the requirements of green coal mining (Zhang et al., 2009). In addition, experiments conducted on the use of broken gangue as the gob filling material (Hu & Guo, 2009; Jiang et al., 2001), found it had good physical properties, such as a high compressive strength and a greater residual strength than that of broken shale and sandstone, and a high compressibility, so it fully meets the supportive resistance, rotation, and sinking adaptation requirements of large roof structures, making it an ideal gob side filling material (Miao & Zhang, 2007; Su & Hao, 2002).

However, GER in steep coal seams is closely connected to the movement of the roof. Many studies have been conducted on the mechanical structure, stress distribution, deformation, and failure process of steep rooves by "clamped beam" and "simply support beam" theory based on various conditions and mining characteristics (Cao, 1992). It was concluded that the main roof will form "voussoir beam" structures, and an immediate roof located at the middle-lower part of the working face will probably form a small scale "voussoir arch" structure due to the waste filling effect (Huang, 2002). In addition, it was found that a steep coal seam has asymmetric mechanical characteristics along the dip direction. Roof caving began in the middle-upper part of the working face and continued into the upper strata and the lower part of the working face. Finally, the caving range can extend to the tailgate with the working face advancing (Wu et al., 2010a; 2010b). In addition, based on previous studies, the failure mechanism of the gateway in a steep coal seam was analyzed. As the dip angle increases, the gateway's failure location will mainly be concentrated on the ribs, the roof's upper corner, and the floor's lower corner due to roof deformation and floor sliding (Gou & Xin, 2011; Huang et al., 2006; Xin et al., 2012).

Though we have a great deal of knowledge concerning GER, the utilization of GER in steep coal seams has not been extensively developed to date. In recent years, as mining science progressed, GER technology in steep coal seams has begun to be considered and accepted. The use of GER has been tested in several steep coal seams, and a certain amount of success has been obtained (Deng & Wang, 2014; Zhang et al., 2014; 2015; Zhou et al., 2012). But even so, many technical problems still exist and have not been systematically analyzed. Therefore, we conducted our study to further develop our understanding. First, the steep coal seams were investigated. Second, the characteristics of the caved in rooves and the stress evolution were determined using three-dimensional distinct element code (3DEC) (Itasca, 2013). Third, the rock blocking device and grouting reinforcement method of the natural filling technology and the hydraulic support for strengthen and supporting the gateway were developed. Finally, the gateway cross section and its driving support were designed taking into consideration the earlier findings.

2. Investigation of steep coal seams

The world's major coal-producing countries, e.g., Russia, the UK, Germany, and Poland, have a large proportion of their reserves in steep coal seams. In China, these reserves account for 14.05% of the total coal reserves, and in some provinces, it is an even larger proportion (Wu et al., 2000). There are 4 main coalfields (1, 2, 3, and 4) in the southwestern part of Sichuan Province, China, as shown in Fig. 1 (Yang et al., 2016b).

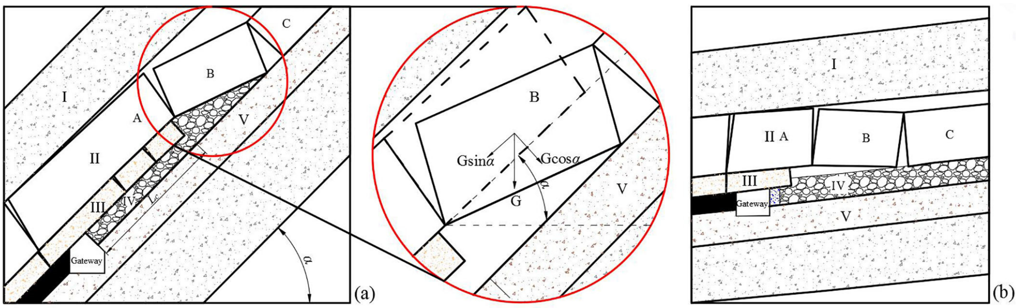
As can be seen from Fig. 1, coalfields 1, 2, and 3 located around the Sichuan Basin, while coalfield 4 is located in southern Sichuan. The coal-bearing strata contain a large number of tectonic structures, resulting in diverse and complex geological conditions. To obtain detailed dip angles for this area, 45 workable coal seams within the 4 coalfields were analyzed. It was found that 56% of the investigated coal seams (25 coal seams) have dips of more than 35°. In addition, these steep coal seams accounts for about 20% of the proven reserves and 10% of the coal output of China's coal seams, and more than 50% of these contain high quality coking coal and anthracite (Wu et al., 2014). Thus, the coal recovery rate must be improved, and the use of GER technology in steep coal seams is critical and necessary.

Fig. 1. Locations of coalfields in Sichuan Province

3. Analysis of the advantage of the use of GER in steep coal seams

3.1. Analysis of the characteristics of roof cave-ins

It is well-known that after the longwall mining working face advances, the roof will cave-in. The characteristics of this caving in steep coal seams are shown in Fig. 2a, assuming the caved waste rock is blocked on the gob side (Huang, 2002; Tu et al., 2015; Wu et al., 2010b; Zhang et al., 2015).



I: Overburden strata; II: Main roof; III: Immediate roof; IV: Caved waste rock; V: Immediate floor.

Fig. 2. Characteristics of roof cave-ins. (a) Characteristics of roof cave-ins in steep coal seams; (b). characteristics of roof cave-ins in flat and gently inclined coal seams

The characteristics of roof cave-ins, mainly involve the main roof caving structure, the filling characteristic, and the dynamic effect. In this study, they were analyzed as follows:

(1) The main roof caving structure. The characteristics of the roof's failure and movement exhibit an obvious asymmetry. In the inclined direction, the middle-upper part of the

roof caved first and periodic weighting, thus resulting in an enlarged caving space, and greatly rotated deformation of the main roof, which fractured and formed "voussoir beam" structures.

(2) The filling characteristic. After the middle-upper part of the immediate roof caved, the caved waste rock rolled or slid toward the lower working face, and finally amassed there. The inclination length of the waste rock filling and supporting part is L_c, as seen in Equation 1 (Huang, 2002).

$$L_c = \frac{\bar{h} k_i}{\bar{h} + h} L - 0.5(\bar{h} + h)\cot(\alpha - \beta) \tag{1}$$

L is the inclination length of the working face (in m); k_i is the initial bulking factor of the immediate roof; $\bar{h}$ is the average caving height of the immediate roof (in m); h is the mining height (in m); α is the dip angle of the coal seam (in degrees); and β is the natural repose angle of the caved waste rock (35°).

Because of the filling effect of the long part, it is a long distance from the bending, rotating, and breaking location of the main roof to the gateway, which lowers the influence of the cave-in on the gateway's stability compared to that of flat and gently inclined coal seams (as shown in Fig. 2b), making steep coal seams more advantageous to the stability of the gateway.

(3) The dynamic effect. The caving of the main roof has a dynamic effect on the gateway, and the long filling distance of the waste rock greatly weakens its dynamic damage effect compared to that of the flat and gently inclined coal seams (as shown in Fig. 2b). In addition, with increasing dip angle, the gravity component ($G\cos\alpha$) of the key block (B) (Fig. 2a) will decrease after it loses stability, which can also reduce the dynamic effect.

This demonstrates that the characteristics of roof cave-ins in steep coal seams are advantageous to GER.

3.2. Analysis of the stress redistribution

In the case of stress redistribution caused by roof cave-ins, the secondary stress has a large impact on the stability and integrity of the gateway surrounding the coal and rock masses.

3.2.1. Numerical model

To study the stress redistribution, one coal mine with steep coal seams was investigated, and a 3DEC (Itasca, 2013) model was employed for plane-strain conditions. The dimension of the model was 170×5×175 m, in the x, y and z direction, respectively, and it included the coal seams and rock strata, with a total of 7 layers, as shown in Fig. 3a, in accordance with the geological conditions of the investigated mine. The Mohr-Coulomb yield criterion was used for the strata and the Coulomb slip model was used for the contacts. The mechanical and physical properties of all the layers and the contacts between the layers are described in Deng and Wang (2014), Gao et al. (2014) and Verma and Singh (2010). All of the side boundaries were roller-constrained, and the bottom was fixed. The upper boundary was subjected to a uniformly distributed vertical stress, for detailed see Deng and Wang (2014).

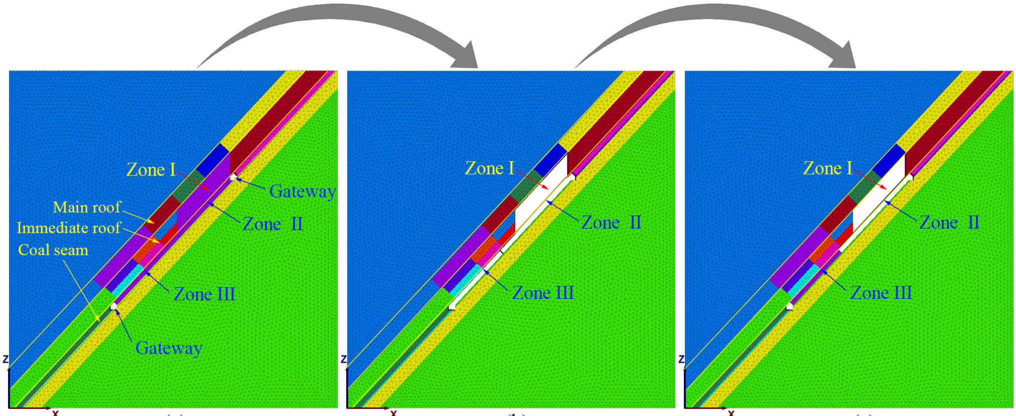


Fig. 3. Numerical model of the roof cave-in of a steep coal seam. (a) Numerical model; (b) "excavated" step; and (c) "fill" step

First, we used the model to perform an initial equilibrium calculation. Then, the headgate and tailgate were excavated and the model was run until it reached equilibrium again. Finally, the roof was removed in "Zone I", "Zone II", and "Zone III" of the coal seam using the "excavate" command (as shown in Fig. 3b). Then the gob backfill zone of "Zone III" was filled with gob filling material using the "fill" command (as shown in Fig. 3c), and the model continued to calculate.

3.2.2. Gob filling material

The gob filling material (caved rock) is a strain-stiffening material. After the initial high compaction, the material will be stiffer and the modulus of the compacted aggregate will increase. The double-yield model, which is intended to represent the materials in which there may be significant irreversible compaction in addition to shear yielding, is available in 3DEC. Hence, the double-yield (DY) model has been widely used to simulate gob material in many studies (Shabanimashcool & Li, 2012; 2013; Wang et al., 2015b). Salamon's model is valid to simulate the stress-strain relationship of cave-in materials, as shown in Equation (2).

$$\sigma = \frac{E_0 \varepsilon}{1 - \varepsilon / \varepsilon_m} \tag{2}$$

where σ is the uniaxial stress applied to the gob material while the material is rigidly confined laterally; E_0 is the initial tangential modulus of the material; ε is the strain occurring under the applied stress; ε_m is the maximum possible strain; and E_0 can be obtained from Equation (3) (Yavuz, 2004).

$$E_0 = \frac{10.39 \sigma_c^{\ 1.042}}{k_i^{\ 7.7}} \tag{3}$$

where σ_c is the compressive strength of the caved rock piece (30 MPa); and k_i is the initial bulking factor of the caved rock.

For the determination of ε_m, some studies have used $\varepsilon_m = (k_i - 1)/k_i$ and $k_i = (h_m + h_n)/h_n$ to calculate in flat seam. h_m is the mining height, and h_n is the roof caving height. However, the situation is more complex in a steep coal seam, as the caved rock will roll or slide to the lower part of the working face where only existed the mining height and no caving height. In addition, the equation for ε_m was obtained assuming only that the maximum displacement of the caved rock is the mining height, which cannot be reached. Thus, the equations for ε_m and k_i cannot be used in the case of steep coal seams. In reality, the caved rock has a compression limit, which can be expressed by the residual bulking factor (k_r), and the maximum displacement of the caved rock can be controlled by k_i and k_r. Hence, ε_m can be determined from Equation (4).

$$\varepsilon_m = \frac{h - \dfrac{h}{k_i} \times k_r}{h} = 1 - \frac{k_r}{k_i} \tag{4}$$

Site investigations in coal mines show that the bulking factor of coal rocks is1.1 to1.5 (Shabanimashcool & Li, 2012). During the rolling or sliding process of waste rock, the initial bulking factor can be defined as $k_i = 1.5$, and the residual bulking factor can be defined as $k_r = 1.1$. Thus, the maximum possible strain (ε_m) of the gob material and the initial tangent modulus (E_0) can be calculated as 0.36 and 15.8 MPa, respectively. The stress-strain curve for Equation (2) had been plotted and illustrated in Fig. 4a.

To determine the gob's material parameters, a simple zone was defined with dimensions of 1×1×1 m (Fig. 4a) (Wang et al., 2015b). Loading was simulated by applying a velocity to the top of a model with confined sides. The strain-stiffening curve from Equation (2) for the given variables was fitted using an iterative change in the bulk and shear moduli and the angle of friction of the gob filling material (Fig. 4a). The final parameters obtained and used in the gob DY model are listed in Table 1.

TABLE 1

Mechanical properties of the gob material

Material	**Constitutive model**	**Properties**							
Gob filling material	Double-yield	Cap pressure (MPa)	ε_m	E_0 (MPa)	Dens (kg/m^3)	Bu (GPa)	Sh (GPa)	Con (MPa)	Fri (degree)
		$P = E_0\varepsilon/(1 - \varepsilon/\varepsilon_m)$	0.36	15.8	1150	2	1	0.01	2

TABLE 2

Application investigation of gateway cross section

Cross section	**Trapezoid cross section**	**Special cross section**	**Inclined arch cross section**	**Other cross section**
Number	38	4	22	11
Proportion	50.7%	5.3%	29.3%	14.7%
Dip angle	<40°	>40°	>40°	Others

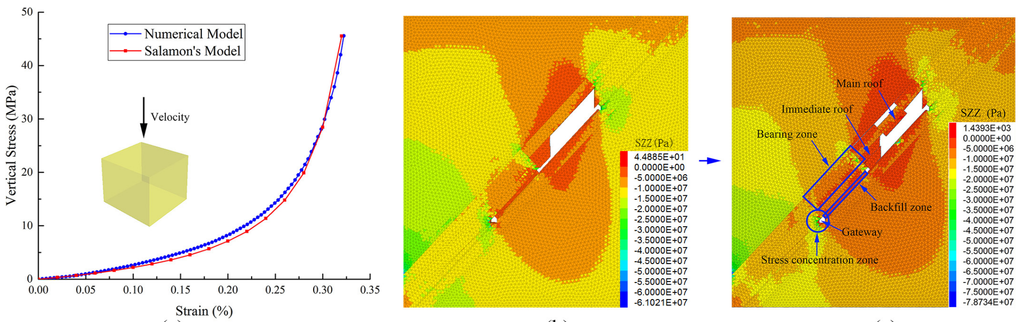
TABLE 3

The detailed support parameters

Initial driving support system	**Material**	**Specification (mm)**	**Inter-row spacing (mm)**	**Pretension force (kN)**	**Anchorage length (m)**
Bolt in roof	High-strength left-screw-thread steel	φ22 × (≥2400)	800 × 800 or 700 × 700	90	1.65
Bolt in high side wall		φ22 × (≥2400)		60	1.65
Bolt in low side wall		φ22 ×(≥2000)		60	1.43
Cable in roof	Steel wire	φ22 × 6300	(1500-2500) × (1500-2500)	200	2

3.2.3. Stress redistribution results

The vertical stress distribution characteristics are shown in Fig. 4b and c. It can be seen that significant asymmetric vertical stress is exhibited in the roof and floor. In addition, the mining-induced stress of the overlying rock strata will be released and transferred after mining occurs, and its stress state will become simpler. Then, the filling caved rock will change the stress state to a three-dimensional stress state (stress restoration) (Li et al., 2010) and it will exert more pressure. So, a large bearing zone forms above the backfill zone (Fig. 4c), and it will distribute more of the overburden pressure. However, the stress concentration zone is distributed around the gateway, as shown in Fig. 3c, and due to the role of the bearing zone, the degree of stress concentration degree is relative small (concentrated stress =10-15 MPa, and gateway stress at the buried depth = 12 MPa).



(a) (b) (c)

Fig. 4. Stress redistribution process. (a) Stress-strain curve of the numerical model; and; (b) stress-strain curve for Salamon's model for gob material; and (c) stress distribution

After the stress redistribution analysis, it was concluded that a low stress environment was formed around the gateway in a steep coal seam. This is advantageous for GER, and it is also useful in avoiding floor heave and coal bumps.

4. The key technology for GER

4.1. Support zone for steep coal seams

The maintenance of a gateway by GER is mainly conducted in two stages: (1) entry support strengthening during the mining period; and (2) construction of a gob-side support zone, as shown in Fig. 5.

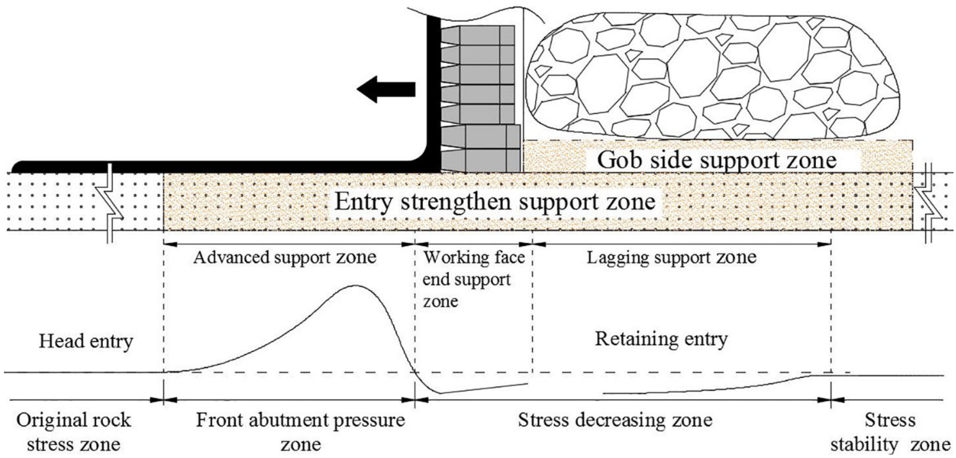


Fig. 5. The gob-side entry retaining zone

The entry support strengthening zone includes the advanced support zone, the working face end support zone, and the lagging support zone. The former is mainly located in the front abutment pressure zone, the latter two are mainly located in the decreasing stress zone and are seriously influenced by roof weighting and rotation, especially in the case of the working face end support zone. However, the influence is not that much greater than that of a flat and gently inclined coal seam, as analyzed above.

The gob-side support zone is located behind the end of the working face. Its filling material, which is significantly related to the dip of the coal seam, is usually high-water material (HWM), concrete, paste, block, and waste pack (Tan et al., 2015) when the dip angle is relative small. However, for a steep angle coal seam, so far, only natural filling with caved waste rock can be used to advantage.

4.2. Natural filling technology for caved waste rock

The natural filling of caved waste rock is superior compared to other support methods. However, when rolling or sliding to the gob-side (as shown in Fig. 6), the waste rock will have a high velocity and strong impact force. Here, taking into account the impact effect of the caved rock, the impact force was calculated to facilitate the difficult analysis of the current support and the later theoretical analysis of the supporting devices.

The impact force is calculated by Equation 5 according to the law of conservation of energy.

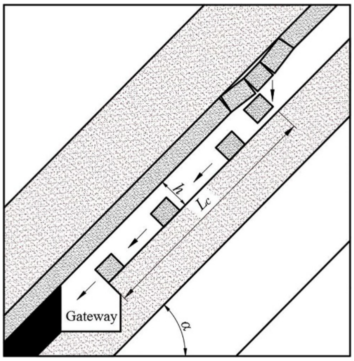


Fig. 6. Model for calculating the impact force

$$\begin{cases} mg\left(\dfrac{h}{\cos\alpha}+(L_c-h\tan\alpha)\sin\alpha\right)=mgf(L_c-h\tan\alpha)\cos\alpha+mv^2/2 \\ F=mv/t \end{cases} \tag{5}$$

where F is the impact force (in N); m is the maximum quantity of caved waste rock (in kg); t is the impact process time (in s); g is the acceleration of gravity (9.8 m/s^2); f is the bottom face friction coefficient (0.5); and L_c is approximately treated as the rolling distance of the caved rock.

Here, to calculate the impact force, we utilized the following steep coal seam parameters (Deng and Wang, 2014): L = 86 m, $\bar{h}$ = 2 m, α = 47.2°, h = 2.36 m, m = (2×4×1.5) m^3 × 2500 kg/m^3 = 30,000 kg, and t = 2 s (Zhou et al., 2012). According to the results of Equations (1) and (3), the impact force (F) of the largest caved waste rock against the support system is estimated to be 280 kN. The minimum impact width of the waste rock is 1.5 m, so the impact force over one meter is about 186.7 kN. The calculation results indicate that only single hydraulic props temporarily support the GER is difficult to prevent the impact of waste rocks. Measures must be taken to reduce the impact force by increasing the impact time (Zhou et al., 2012).

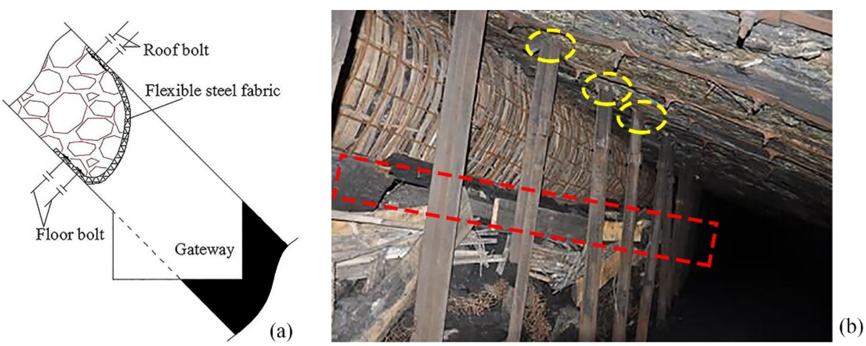


Fig. 7. Flexible support technique. (a) Flexible steel fabric block; and (b) field effect

Thus, a flexible support technique (Zhang et al., 2015) was used in the field, as shown in Fig. 7.

In practice, as shown in Fig. 7a, first bolts were set in the roof and floor, and then, the two ends of the flexible steel fabric were fixed to these bolts. However, there was large amount of weak interlayer or broken material due to the geological conditions of the floor, which can easily lead to the floor sliding along the bedding plane. This would make the floor bolt useless for fixing the flexible steel fabric in place, as indicated by the red zone in Fig. 7b. In addition, the single props are usually inserted into the roof, as indicated by the yellow zone in Fig. 7b (Zhang et al., 2015). A new rock blocking device and grouting reinforcement method were invented to solve this problem, as shown in Fig. 8.

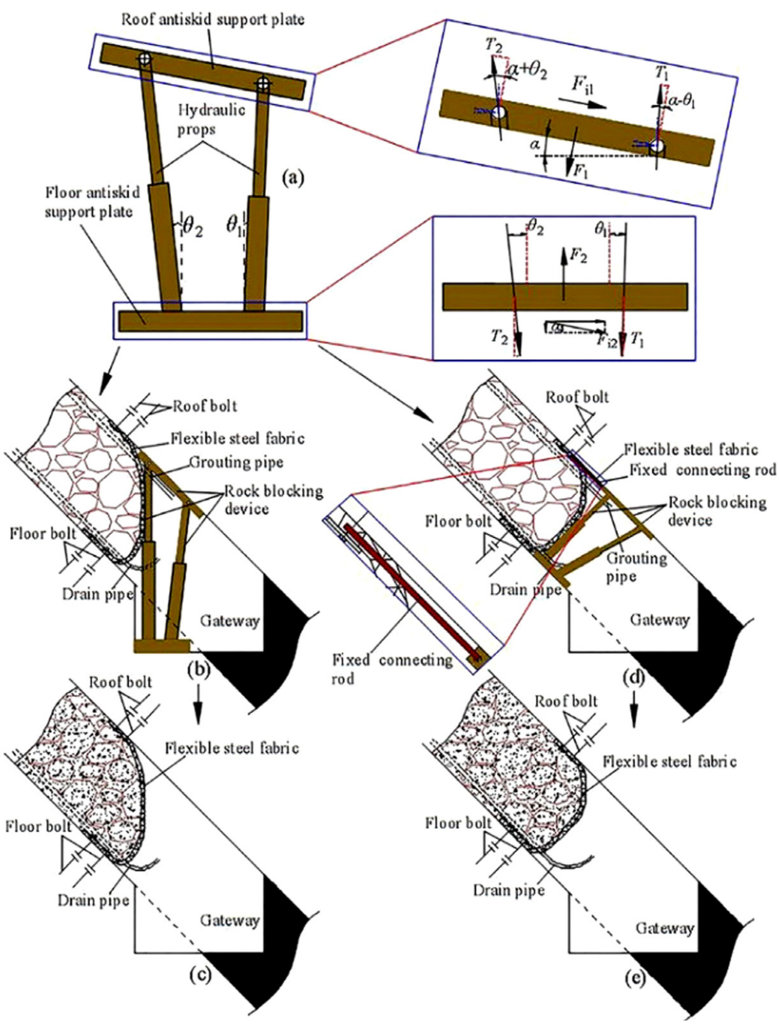


Fig. 8. Rock blocking device and grouting reinforcement method. (a) Rock blocking device; (b) gateway blocking device; (c) grouting effect; (d) gob-side blocking device; and (e) grouting effect

As seen in Fig. 8, the equipment can be divided into the gateway and gob-side rock blocking devices. The force of the roof's antiskid support plate (RASP) and the floor's antiskid support plate (FASP) are shown in Fig. 8a. Their equilibrium limit relationships are shown in Equations (6) and (7), respectively.

$$F_1 w + T_1 \sin(\alpha - \theta_1) + T_2 \sin(\alpha + \theta_2) = F_{i1} \tag{6}$$

$$F_2 w + T_1 \sin\theta_1 = F_{i2} \cos\alpha + T_2 \sin\theta_2 \tag{7}$$

where F_1 is the roof pressure, and F_2 is the floor pressure, as expressed by Equations (8) and (9), respectively.

$$F_1 = T_2 \cos(\alpha + \theta_2) + T_1 \cos(\alpha + \theta_1) \tag{8}$$

$$F_2 = T_2 \cos\theta_2 + T_1 \cos\theta_1 + F_{i2} \sin\alpha \tag{9}$$

where T_1 and T_2 are the pillar support force; w is the friction coefficient between the RASP and the roof or between the FASP and the floor; α is the dip of the coal seam; θ_1 and θ_2 are the pillar tilt angles; F_{i1} and F_{i2} are the impact forces caused by the caved waste rock.

It can be seen that the equilibrium relationship of the rock blocking device is complex and is related to several factors. When the rock blocking device is utilized in the gateway (as seen in Fig. 8b) and $\alpha > 0$, Equations (6) and (7) show that increasing T_2 will increase the stability of the RASP, but will decrease the stability of the FASP. However, when $\alpha > \theta_1$, increasing T_1 can improve the instability of the FASP and further increase the stability of the RASP. Thus, the two tilted hydraulic props hinged at the RASP and FASP can form a triangular stability support structure. Furthermore, Equation (10) can be obtained from Equations (6) to (9).

$$\left[2w\left(T_2 \cos\frac{\alpha + 2\theta_2}{2} \cos\frac{\alpha}{2} + T_1 \cos\frac{\alpha + 2\theta_1}{2} \cos\frac{\alpha}{2}\right) + 2\left(T_1 \sin\frac{\alpha}{2} \cos\frac{\alpha - 2\theta_1}{2} + T_2 \sin\frac{\alpha}{2} \cos\frac{\alpha + 2\theta_1}{2}\right) \right] \Big/ \left[F_{i1} + F_{i2}(\cos\alpha - \sin\alpha)\right] = n_1 \tag{10}$$

In addition, when $\alpha = 0$, this device can be utilized at the gob-side (as seen in Fig. 8d). Thus, Equation (10) can be transformed into Equation (11).

$$2w(T_2 \cos\theta_2 + T_1 \cos\theta_1) \big/ \left(F_{i1} + F_{i2}\right) = n_2 \tag{11}$$

where n_1 and n_2 are the safety factors. Compared to Equations (10) and (11) when $\alpha = 47.2°$, $w = 0.3$ (Jin-an and Jun-ling, 2016), $\theta_1 = \theta_2 = 20°$, $T_1 = T_2 = 200$ kN, and $F_{i1} + F_{i2} = F$ (as seen in Eq. (5)), we found that

$$n_1 > n_2 > 1 \tag{12}$$

Thus, placing the device in the gateway can achieve greater stability than placing it at the gob-side. In reality, the RASP and FASP are for nonskid purposes and the friction coefficient (w) can be more than 0.3. Based on current technological capabilities, the value of T_1 and T_2 can be greater than 200 kN; and combined with the stop function of the flexible steel fabric, the safety factor should be greater.

Furthermore, when used in steep coal seams, this method presents other features:

(1) The rock blocking device can be recycled after the grouting reinforcement, and when used at the gob-side, the initial stability can be well controlled by fixing the connecting rod to the roof bolt, as seen in Fig. 8d.

(2) The blocking device can satisfactorily support and maintain the stability and integrity of the retained gateway roof, especially for a soft roof, because of its integral structure and support capacity.

(3) The slurry can not only reinforce the broken waste rock and increase its bearing capacity to hinder deformation of the overlying rock strata (Li et al., 2008), but it can also infiltrate into the fractured floor and improve the strength of the rock mass, allowing the blocking device to be fixed to the floor.

4.3. Strengthening support device

The stability and integrity of the roof's rock mass in strengthening support zone has a significant effect on the success of GER. So far, the vast majority of strengthening support zones have been supported by two or three rows of single hydraulic props with the articulated roof beam serving as the temporary support. However, because the props cannot form an integral structure, they will lack stability due to abutment pressure and roof rotation and weighting. In addition, a single hydraulic prop is usually inserted into the floor or roof because of the existence of uneven pressure due to the low strength and stiffness of the roof's rock mass and the higher strength and stiffness of the strengthening support body, which will negatively influence the quality of the support (Yang et al., 2015). Thus, the strengthening support zone in a steep coal seam requires a hydraulic support to provide increased strength to the roof's rock mass, as shown in Fig. 9.

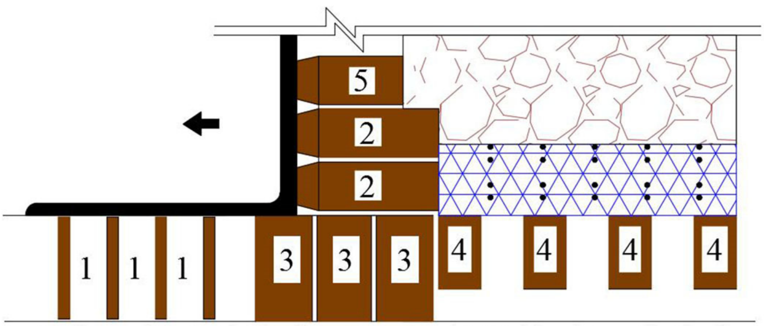


Fig. 9. Diagram of the entry strengthening support device

The advanced hydraulic support (1) has an integral structure, as shown in Fig. 10, so it can better resist mining-induced stress and large roof loading. Hence, it can improve the roof's integrity to benefit the GER.

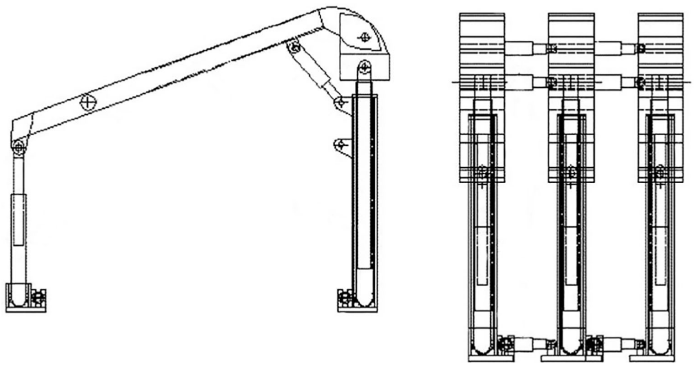


Fig. 10. The advanced hydraulic support

The working face end support zone experienced serious roof deformation, but it also requires a large space with no pillar support for the retaining gateway. This makes it very difficult to maintain the roof at the end of the working face. To facilitate the implementation of the GER and later maintenance of the retained gateway, hydraulic supports (2) and (3) at the end of the working face were developed to fulfill this role. The details of their development are included in the patents (Numbers CN204327145U and CN202628152U, China, respectively).

The lagging support zone was affected by the roof cave-in. The lagging hydraulic support (4) has a simple and compact structural design, and it is easy to operate. It can provide significant supportive force and improve safety conditions, the details of which are included in the patent (Number CN203717003U, China).

5. Gateway driving and its support technology

The development of GER has been a systematic project, which not only involves support in the late retaining stage, but also includes the design of a cross section and driving support for a gateway in the initial driving stage (Hua, 2006). These practices should be used together so as to ensure the stability and integrity of the roof in the initial stage and the success of the GER of the gateway in a steep coal seam. However, all of the retained gateways were implemented based on the current existing support conditions of gateways in China, which include expanding ribs and supplementary bolts (cable). This leads to the need for a large amount of material, is labor intensive, and expensive. Thus, the size and shape of the gateway's cross section should be considered when choosing a support method.

5.1. Gateway driving

Comprehensive mechanized driving is mainly accomplished by mechanical cutting and vibration. The degree to which this method disturbs the surrounding rock is far less than that of blast driving, so it causes less damage to the coal and rock masses. Thus, the comprehensive mechanized driving method is recommended for gateway driving. If the blasting method must be used, the borehole arrangement and blasting parameters should be chosen to minimize the number of fractures and to control their propagation depth so as to maintain the integrity of the surrounding rock.

Fully-mechanized longwall mining was successfully utilized in the Lvshui Dong mine in China with coal seam dips of 60°. As China gradually implements the development of fully-mechanized longwall mining and pillarless mining, more requirements have been put in for the size and shape of the gateway's cross section. In this study, we conclude that the the size (bottom width and roof height) of the cross section of a new gateway would not influence the production or the ability to meet any related requirements. In addition, field studies conducted by engineers on 75 statistical gateways (as seen in Table 2) in Sichuan Province, China show that a trapezoidal cross section (Fig. 11a) is preferred when $\alpha \leq 40°$, and a special cross section (Fig. 11b) and an inclined arch cross section (Fig. 11c) are preferred when $\alpha > 40°$.

In reality, the height of high side wall of a trapezoidal cross section gateway will increase as the dip angle increases, resulting in a large free face and simple stress state, which are prone to experience tensile and shear failure. However, for a gateway with a special cross section or an inclined arch cross section, the coal and rock masses will have a small free face and will experience multiaxial stress, which is conducive to increased stability. Thus, based on the current support technique, we chose a trapezoidal cross section when the dip of the coal seam is $35° < \alpha \leq 45°$, a special cross section when the surrounding rock is stable, and an inclined arch cross section when the surrounding rock is unstable and $45° < \alpha \leq 55°$, as shown in Fig. 11.

5.2. Gateway support technology

Based on the above analysis, the supporting factors favoring the use of GER in steep coal seams can be summarized as follows.

(1) It ensures reliable support during the initial driving stage, which reduces the deformation and degree of fracturing of the surrounding rock.

(2) It to minimize the need for supplementary bolts (cable) to strengthen the support zone of GER, making efficient use of the existing bolts and cables.

Thus, the gateway support system is provided in the initial driving stage, as shown in Fig. 11. The detailed support parameters are listed in Table 3.

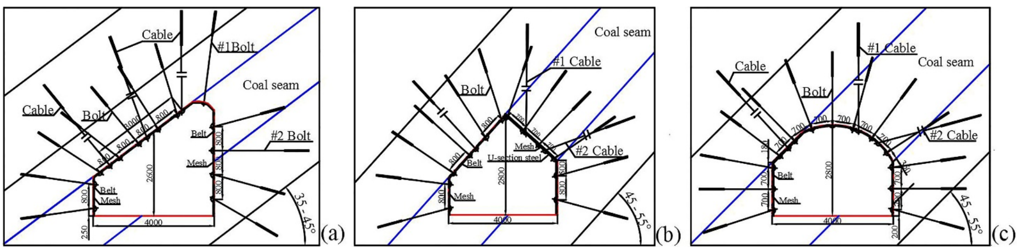


Fig. 11. Initial driving support system

Rock bolts have become a popular technique for reinforcing rock masses all over the world. The important bearing capacity mechanisms of rock bolting are suspending, nailing, beam building, arch building effects, and new rock mass theory (the behavior of a complex of rock masses and bolts can be assumed as a new rock mass with an increased RMR value) (Mohammadi et al., 2017). However, according to some statistics on the retained gateway, the bolt material, anchorage length, and pretension force of the initial driving support system usually result in poor stability and greater deformation during retaining of the gateway. In addition, one or two cables or bolts within a cross section were usually supplemented beyond the strengthening support zone to increase the roof's stability.

Thus, we improved the support parameters, used bolts and cables combined with steel mesh and steel belts, designed three cables in a gateway cross section, and appropriately increased the length of the bolts and the pretension force of the bolts and cables. In addition, the broken width of the roof and the upper side wall are usually greater than that of the lower side wall, so the roof and upper side wall require longer bolts than the lower side wall.

The support system can form a combined arch structure in the surrounding coal and rock (Mohammadi et al., 2015; Wang et al., 2015a), and the support parameters can enhance the bearing capacity of the combined arch structure. In addition, it uses an extensive bolting method, and the anchorage length is greater than 65% of the bolt length. This can increase the pre-stress sphere. The effect of the pre-stress anchor was more pronounced in the soft rock than in hard rock (Zheng et al., 2012). Cable reinforcement must be added to the support system to produce an overhanging effect in the stable roof rocks. In summary, the support system can improve the stress state of the surrounding coal and rock masses, and increase the thickness of the reinforcement structure, especially the arch foot, formed by the anchoring force, which will increase its self-bearing ability (Yang, 2010), and avoid large displacement and/or rapid failure.

Field studies have shown that the developed bolt materials and manufacturing processes, the high pretension force, and the intensive bolts and cables can effectively control the dilatant deformation and maintain stability (Wang et al., 2015b).

In addition, the #1 and #2 bolts (Fig. 11a) when appropriately extended and the #1 and #2 cables (Fig. 11b and c) can be reserved and reused to fix the flexible steel fabric after the work face is advanced, so that the roof bolt and floor bolt (as shown in Fig. 7) would not be arranged for retaining the gateway.

6. Conclusions

This paper analyzed the support technology for GER in steep coal seams based on a systematic project. In detail, roof cave-in structures, filling characteristic and dynamic effect of the gateway, and the stress redistribution within the overlying rock strata indicate that the use of GER in steep coal seams provides significant advantages compared to its use in low dip angle coal seams. In addition, for natural filling, the rock blocking device and grouting reinforcement method developed in this study hold up well against the impact force of the waste rock and reinforce the waste rock and soft floor. Furthermore, a set of strengthening support devices were developed for use in the strengthening support zone. In addition, to more effectively retain the gateway in the late retaining stage, the size (bottom width and roof height) of the cross section of a new gateway does not influence the production or the ability to meet any related requirements. The appropriate type of cross section, i.e., Trapezoidal cross section when the dip of the coal seam is $35° < \alpha \leq 45°$, special cross section when the surrounding rock is stable, and inclined arch cross section when the surrounding rock is unstable and $45° < \alpha \leq 55°$, which was determined according to the dip of the coal and the type of surrounding rock. Finally, based on two driving support principles, a support system consisting of bolts and cables combined with steel mesh and steel belts was developed. The optimized bolt material, increased length of the bolts and their anchorage length, and the increased pretension force of the bolts and cables were the main features of the support system that improved the self-bearing ability of the surrounding coal and rock masses.

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