Study on the Overlying Strata Movements and Stability Control of the Retained Goaf-side Gateroad

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by

Zhiyi ZHANG*, Hideki SHIMADA **, Takashi SASAOKA *** and Wen KAI†

Abstract

Recently, Y type gateroad layout, which is based on a retained gob-side gateroad (RGG), is being widely used for the purposes of optimizing ventilation system, increasing coal recovery rate, etc. The RGG, however, suffers several engineering disturbances including roadway excavation and two adjacent longwall panels mining during its whole serving life, which makes the stability control of this kind of roadway is difficult. In order to have a deep insight of this issue, a novel simulation software referred to as Realistic Failure Process Analysis (RFPA) is employed in present study to analyze the overlying strata movements by which the redistributed stress is induced. And then, three RGG packfillings locations are compared and the most appropriate one is selected. After that, a combined roadway supporting method involving "three-high" rock bolts and novel grouting cable bolts are put forward to maintain the RGG stability. Finally, engineering practice is carried out in the Pan Yidong deep underground coalmine, and the results of field measurements indicate that the RGG supported by the proposed roadway stabilizing method meets all anticipatory mining goals.

Keywords: Overlying strata movements, Roadway supporting, Realistic failure process analysis (RFPA), Retained goaf-side gateroad (RGG), Grouting cable bolt

1. Introduction

Concerning the conventional U type longwall mining system (**Fig. 1(a)**), high risk of gas explosion due to intensive gas accumulation and significant coal resource waste results from large

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gateroad-protecting coal-pillars have restricted its further application in gassy and deep underground coalmines [1-4]. It is generally recognized that, roof strata will break and cave after beneath longwall panel extraction, and consequently, numerous cracks will form. The gas released from nearby strata will flow through these cracks into underlying empty space, especially into the goaf where   filled with caved rock. And then, the accumulated gas in goaf will further flow towards the upper corner of coal face, because the air pressure there is lowest compared with other places of longwall face in the U type ventilation system. As a result, the upper corner becomes a dangerous place where the gas explosion takes place most frequently. It is also well known that, the coal pillars are always retained in underground coalmines to bear the heavy weight of overlying rock mass for the purposes of protecting gateroad space and preventing surface subsidence. As mining depth increases, however, this kind of roadway-protecting coal-pillar inevitably becomes extremely large to resist the high stress in deep environment, which leads to serious coal resource loss and adverse stress concentration in neighbouring surroundings. To resolve these problems, a modified gateroad layout referred to as the Y shaped one (**Fig. 1(b)**) is proposed. In the new system, a previous longwall gateroad, which is abandoned in the conventional U type settings after coal face passes by, is maintained available by building an artificial packfillings (isolating goaf area and supporting roadway roof) along the goaf side when coal face advances. This retained goaf-side gateroad (RGG) can be used as the outlet of dirty air during the first coal panel mining. Meanwhile, there are two gateroads in this ventilation system for fresh air flowing in. Consequently, the air flow field in goaf is changed and the accumulated gas in goaf area will flow backwards the coal face and the gas overrun in the upper corner will be eliminated [5]. Additionally, the RGG can also be reused as the inlet of fresh air when the adjacent coal panel mining. Large roadway-protecting coal-pillars between two coal panels are removed, and the time and cost for driving new gateroad for the adjacent coal panel mining are also saved [6].
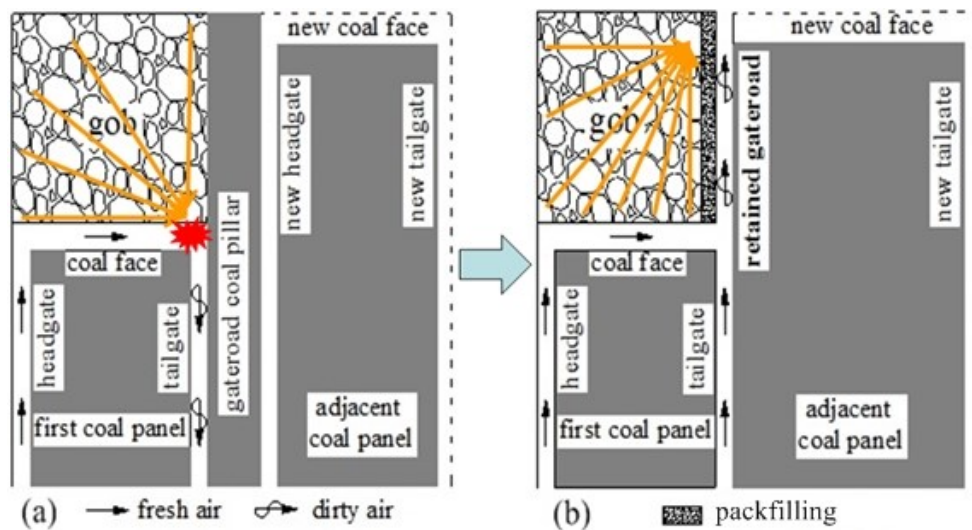
Fig. 1 Conventional U type gateroads layout (a) and modified Y type gateroads layout (b)

While, the stability control of the RGG is difficult because this roadway has to suffer several engineering disturbances, including roadway excavation and longwall panels mining. The structure of

this roadway forms during the above mentioned mining processes, therefore, an analysis to the overlying strata movements is essential if we want to throw light on the characteristics of the disturbing stress evolution, which influences the RGG stability significantly. Several relative researches have been carried out in recent decades. Whittaker (1977) firstly proposed the cantilever theory over a longwall face [7]. Smart (1982) put forward the detached block theory based on static rock mechanics [8]. And then, Qian (1982) established classical arch articulated beam mechanical model over a longwall face and claimed that the overlying strata always formed O-X style breakage after coal panel was extracted, which explains most of the ground pressure behaviors in longwall mining system [9]. Afterwards, Qian's theory was introduced into the analyses of roof structure over the RGG and several valuable models were founded, including curved triangular block model (1987), rock plate superposition model (1993), and given deformation model (2000) [10-12]. These valuable models facilitate the theoretical calculating of the required bearing strength of the RGG surroundings and give scientific guidance to the engineering practice in the field. However, some essential parameters are always difficult to be quantified because the fact that measurement of caving activity in the field is dangerous and costly to some extent. For this, some researchers turned to numerical study on the overlying movements of longwall stope. For instance, Xie (2004) used the UDEC software to analyze the movements of the surroundings of the RGG and influence of packfillings dimensions on the stability of the RGG [13]. Kan (2009) employed the FLAC3D software to study the distribution of stress and deformation in the RGG surrounding rock and designed corresponding roadway supporting technology [14]. Qian (2015) studied the influence of the width of the packfillings wall on the RGG stability in a 900 m deep mine, using UDEC software, and proposed that the packfillings with width of 3 m is the most appropriate choice in that kind of condition [15]. Nevertheless, these numerical software do not fully take into account the discontinuity, heterogeneity, inelasticity and anisotropy in the natural rock mass, and simulation results always have very different from the real situation in the field, especially when these software are used to analyze the initiation and propagation of cracks in the overlying strata when coal panel is being extracted.

In present study, a novel simulation software named as Realistic Failure Process Analysis (RFPA) is employed to study several important issues concerning the stability maintenance of the RGG, including the dynamic moving process and structure characters over advancing longwall face and RGG, and the influence of the RGG packfillings locations on the RGG stability. Based on the results of above analysis, combined supporting technology is designed to maintain the long-term stability of the RGG.

2.   Numerical Simulation

2.1 RFPA Simulation Software

The Rock Failure Process Analysis (RFPA) software was proposed by Tang in 1995 [16]. Different from the finite element software FLAC and discrete element software UDEC, this numerical method put some important natural rock mass properties into consideration to achieve realistic analysis of the

rock failure process. Particularly, several prominent features of this software are listed as follows [17].

(1) The heterogeneity of rock mass at a mesoscopic level is considered by assuming that the material properties follow the Weibull distribution. It is believed that the heterogeneity is the most important feature of rock mass properties. The heterogeneity of rock material and stochastic distribution of defects can be reflected by the statistical constitutive damage model. For convenience of solution, in the RFPA method, the Weibull distribution is used to consider the heterogeneity of rocks.

(2) Elastic damage mechanics is used for describing the constitutive law of the micro-level element. As above discussed, the Weibull distributed defects and impurities in natural rock mass will become the inducing sources of stress concentration, which generates new micro cracks. Eventually, these micro cracks coalesce and result in macroscopic failure.

(3) Finite element method (FEM) is employed as the basic stress analysis tool. The maximum tensile strain criterion and the Mohr-Coulomb criterion are utilized as the damage threshold.

(4) Acoustic emission (AE) event rate is employed as the criterion for rock engineering failure. In quasi-brittle materials, such as rocks, AE is predominantly related to the release of elastic energy. Therefore, as an approximation, it is reasonable to assume that the AE counts are proportional to the number of damaged elements and that all the strain energy released by damaged elements is in the form of AEs.

2.2 Overview of Geology and Numerical model

In this paper, Pan Yidong coalmine, located near Huainan in Anhui province, China, is selected as the modeling target, resulting from the following considerations. Firstly, this coalmine has the typical geological conditions in the east of China, where the RGG is widely applied. Its primarily minable coal seam is buried at a depth of 800 m, with relative gas content of 26.33 m³/t and a thickness of 3.0 m. According to the results of in situ stress measurements with the stress relief method, the maximum and minimum principal stresses are nearly horizontal, and are 36.7 MPa and 18.1 MPa, respectively, and the middle principle stress is nearly vertical, and is up to 19.8 MPa. It should be noted that the maximum principle stress is almost parallel to the orientation of the roadway axis, which indicates that only the middle and minimum principle stress that perpendicular to the roadway axial direction, make mainly contributions to the RGG instability. Empirically, more than 4 gateroads (2 gateroads responsible for fresh air flowing in and others for dirty air returning) are needed to account for the high gas emission, and large coal pillars wider than 15 m between adjacent gateroads are required to resist the high in situ stress, when using the conventional U type mining system. In order to eliminate the risk of gas explosion due to high gas accumulation and heavy coal resources loss results from large coal pillars, Y type mining system based on the RGG is firstly employed in this coalmine. Research results of this study can provide valuable scientific reference for other coalmines of similar geological conditions. Secondly, this coalmine is a newly built one and the supporting criterion and parameters are lack. Roadway stability controlling cannot be satisfied by copying the supporting techniques from other coalmines. Particularly, the immediate roof stratum over the minable coal seam in the Pan Yidong is

typically composed of multilayer weak mudstone with a whole height of approximately 9.0 m. Appropriate roadway stabilizing method should be designed according to the special geotechnical conditions of the Pan Yidong coalmine.

In consideration of the above geological conditions, computational precision and costing time, the dimensions of the plane strain numerical model are determined as 200 m width and 75 m height (**Fig. 2**). The length and height of the longwall panels are 80 m and 3 m, respectively. The cross section of the RGG is rectangular with dimensions of 5 m width and 3 m height, and the packfillings is 3 m in height and 3 m in width. The displacement boundaries of the model are set as roller boundaries along the sides and pinned boundary along the bottom. A stress boundary of 19.13 MPa is applied on the top boundary in the vertical direction. The horizontal stress of 18.07 MPa is applied on horizontal sides. The mechanical parameters used in this model are listed in **Table 1**.
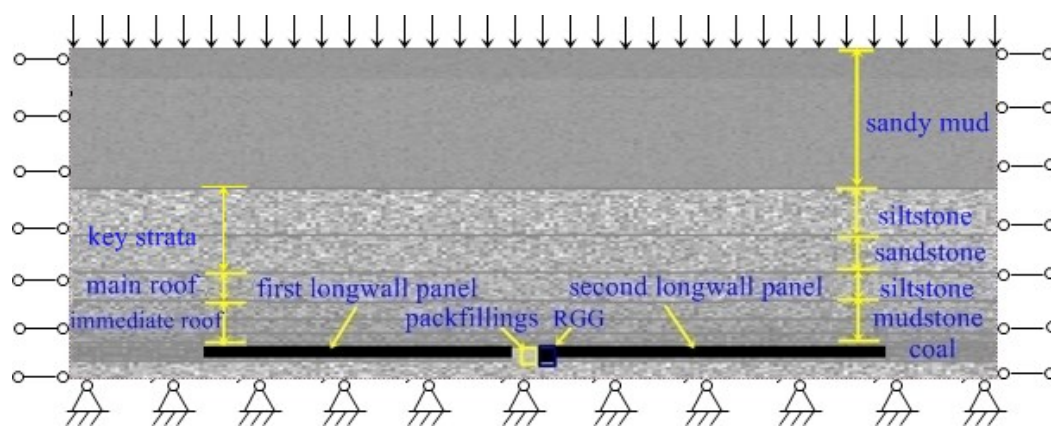

Fig. 2 Initial model mesh of the longwall face and RGG in the Pan Yidong coalmine by RFPA

**Table 1** Mechanical parameters of the rock mass used in simulation

Strata	Density (kg/m³)	Elastic modulus (GPa)	Tensile Strength (MPa)	Poisson ratio	Friction angle (°)
Sandy mud	2500	3.5	3.7	0.27	28
Siltstone	2600	4.7	5.0	0.26	32
Mudstone	2400	2.7	2.9	0.29	27
Coal	1450	1.5	1.4	0.30	23
Sandstone	2700	4.2	4.1	0.25	30
Packfillings	2600	5.1	5.4	0.23	35

The simulation sequences are as follows: (a) generating the initial equilibration state in whole model under initial stress and displacement boundaries; (b) driving a gateroad between two longwall panels and achieving new balance; and (c) extracting the first longwall panel, meanwhile, study the laws of the movements of overlying strata over extracted goaf; (d) when the disturbance results from the first longwall face mining becomes stable, the second longwall panel is mined. During this period, the characteristics of the roof structure over the RGG are studied. In this simulation, the extracted goaf is not filled because the natural caving method is used in the Pan Yidong coalmine.

2.3 Simulation Results and Analyses

2.3.1 Movements of the overlying strata of the first longwall face

When the extracting of the longwall panel starts, abutment pressure ahead of the moving coal face will concentrate and the stress concentrating zone also moves forward together with the advancing coal face. According to the classical theory of overlying strata movements by Qian [9], the main roof over a longwall panel will break and cave periodically after the first caving activity. The abutment pressure when the main roof cracks is highest and its severe disturbance will be exposed not only on the coal face, but also on the RGG that located at the side of longwall panel. From the viewpoints of this point, the moving process of the overlying strata and corresponding abutment pressure in floor strata are monitored, and results are shown in **Fig. 3**.
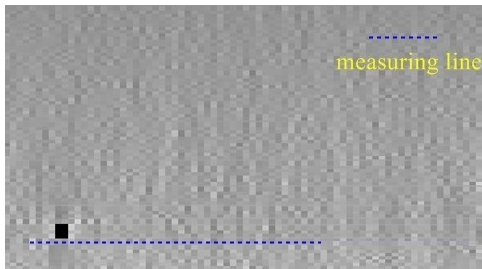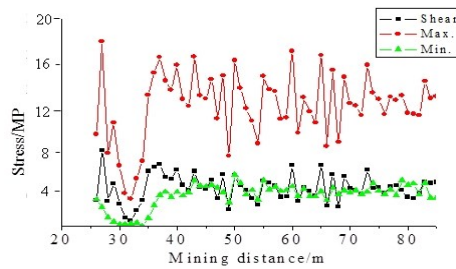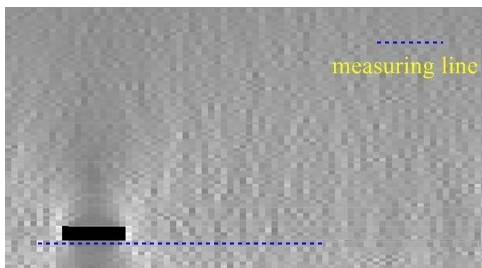

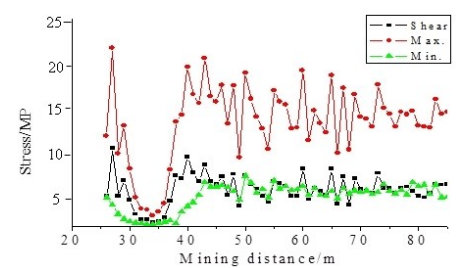**(a1)** Cracks distribution (2 m)
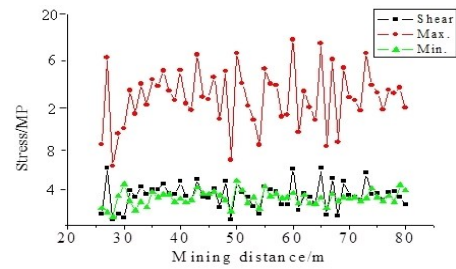
(b1) Stresses evolution (2 m)
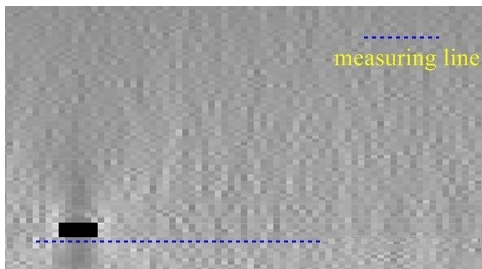
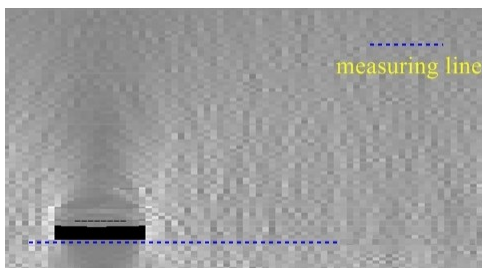
(a2) Cracks distribution (6 m)

(b2) Stresses evolution (6 m)
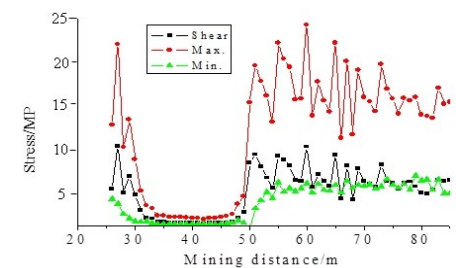

(a3) Cracks distribution (10 m)

(b3) Stresses evolution (10 m)

(a4) Cracks distribution (14 m)

(b4) Stresses evolution (14 m)

(a5) Cracks distribution (20 m)

(b5) Stresses evolution (20 m)

(a6) Cracks distribution (28 m)

(b6) Stresses evolution (28 m)

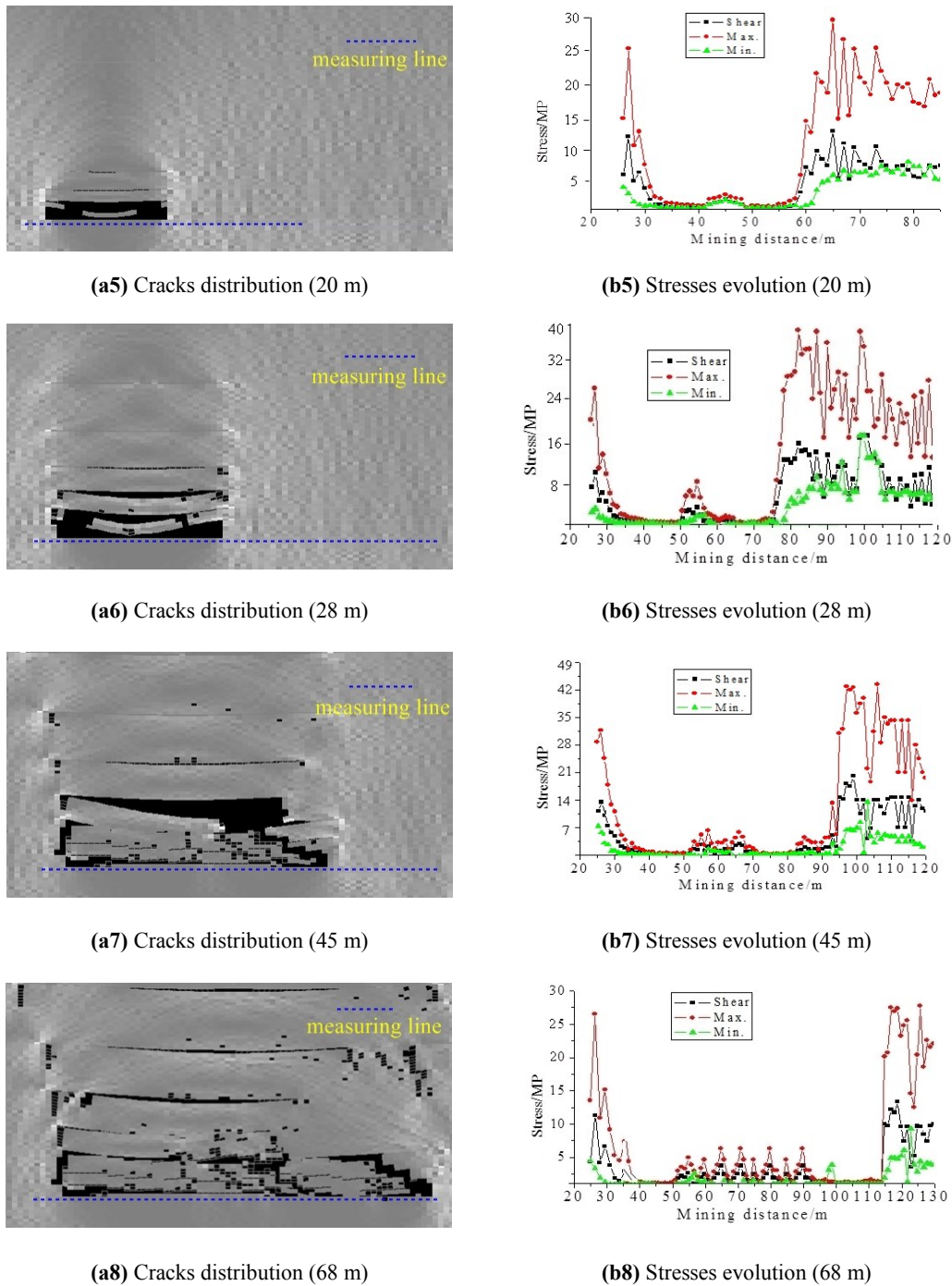**(a7)** Cracks distribution (45 m)

(b7) Stresses evolution (45 m)

(a8) Cracks distribution (68 m)

(b8) Stresses evolution (68 m)

**Fig. 3** Cracks distribution (a) and Stresses evolution in floor (b) during coal face advances

It can be seen from the **Fig. 3** that the cracks in the overlying strata develop gradually and the stresses in floor strata vary significantly during coal face moves. Particularly, when the coal face advances 14 m from the opening cut position, the first layer of the multilayer mudstone roof experiences separation because of bending generated by gravity (**Fig. 3(a4)**); meanwhile, the maximum stress appears 10 m ahead of the coal face with a value of 26.6 MPa (**Fig. 3(b4)**), and the stress concentration factor is 1.4 (26.6 MPa / 19.1 MPa). As the coal face advances continually, the exposed roof area increases gradually and the bending moment also rises. When the coal face moves 20 m away

from the start place, the first layer of the immediate roof breaks and caves, and at the same time, the second layer starts to bend (**Fig. 3(a5)**); the maximum stress before the coal face increases to approximately 30.0 MPa (**Fig. 3(b5)**), and the stress concentration factor is 1.6. When the coal face advancing distance increases to 28 m, all immediate roof strata collapse and main roof strata begin to curve (**Fig. 3(a6)**); at this time, the maximum stress ahead of coal face grows up to 39.9 MPa and the stress concentration factor is 2.1(**Fig. 3(b6)**). Finally, the main roof strata break and cave when the coal face advances to 45 m away from the opening cut (**Fig. 3(a7)**); the maximum stress in the surrounding rock ahead of the coal face increases to the largest value of 43.7 MPa (**Fig. 3(b7)**) and the stress concentration factor is 2.3. The breakage and corresponding induced stress disturb the stability of coal face and gateroad so significantly that the mining distance and abutment stress when the main roof firstly breaks has been widely used as two important references when determining the supporting technology and parameters for coal face and gateroad maintenance. When the coal face continually moves for another 23 m, the main roof caves (**Fig. 3(a8)**), and in this way, the main roof will breaks and caves periodically, as the coal face advances; it should be noted that, the maximum stress decreases to 30.0 MPa (**Fig. 3(a8)**). It is because that force connection of the roof strata is changed and its exposed area is reduced obviously after experiencing the first caving activity.

2.3.2 Structure characteristics of the RGG roof

As one of the results of overlying strata movements induced by longwall panel mining, a special roof structure forms over the RGG. Having a deep insight of the characteristics of this roof structure can facilitates the stability maintenance of the RGG. Because of this, the cracks distribution after two longwall panels mining are observed and the results are showed in **Fig. 4** and **Fig. 5**, respectively.
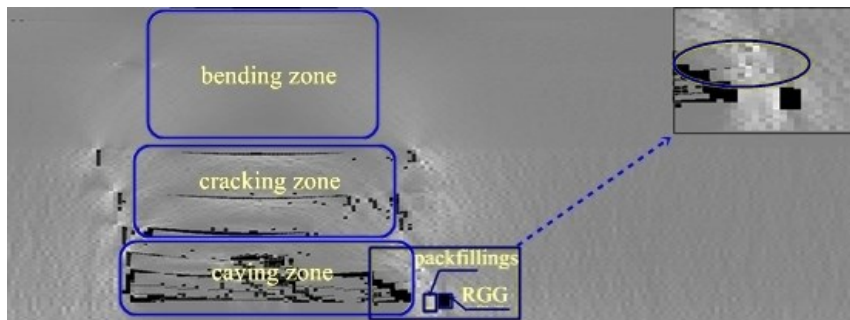
Fig. 4 Cracks distribution and roof structure of the RGG after the first coal panel mining

As presented in **Fig. 4**, after the movements of the overlying strata due to the first coal face mining become stable, three zones form. Firstly, caving zone ranges from the surface of floor strata to the boundary of main roof strata. In this zone, caved rock distributes irregularly and there is plenty of minute space among these caved rocks, which provides the possibility for gas accumulation in this zone. Secondly, cracking zone ranges from the surface of main roof to the boundary of key strata, in which horizontal separation and vertical cracks form. Therefore, stress is reduced and permeability is enlarged and gas can relief and flow towards the caving zone through some interconnected cracks. The

third is the bending zone above the key strata. In this zone, only bending phenomenon occurs in each rock seams and there are little cracks, and gas and water cannot flow effectively.

It can also be seen in the **Fig. 4** that cantilevers form along the side of goaf (over the RGG). These rock cantilevers not only sink vertically, but also rotate gradually towards the adjacent extracted goaf. It indicates that the packfillings not only undergoes the vertical compressive force generated by the vertical sink movement of the roof cantilever, but also has to resist the horizontal force triggered by the rotation movement. This conclusion is a very significant consideration when designing the flexibility of the RGG packfillings. Additionally, the roof cantilevers will sink and rotate until touch the caved rocks in goaf. It clarifies the caved rocks can help the RGG surroundings to bear the overburdens, and the more fully goaf space is filled, the less sink and rotation that roof cantilever experiences and the easier to the RGG stability maintenance.
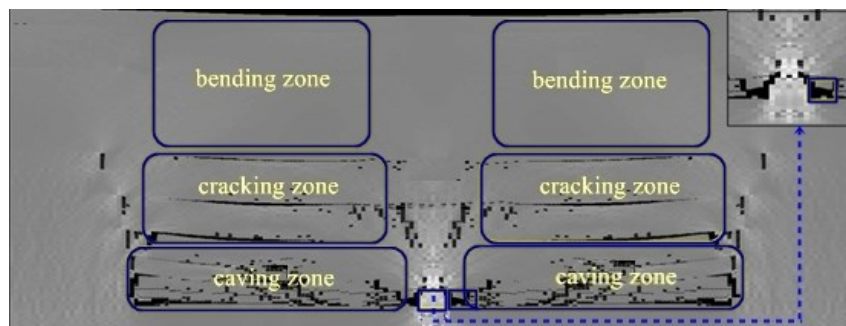

Fig. 5 Cracks distribution and roof structure of the RGG after the second coal panel mining

Fig. 5 shows the cracks distribution in the overlying strata when the second longwall panel is extracted. We can see that the three zones (caving zone, cracking zone and bending zone) generated by two coal panels extraction connect to each other, forming a wider range in which stress is decreased and gas is relief. It benefits gas drainage, eliminates the risk of gas and coal outburst. In addition, space of the RGG is destroyed after the second coal panel is extracted, which is reasonable and expected because the fact that this roadway has finished its whole mission and will not be used any longer. However, the packfillings is stable to some extent, which will inevitably lead to stress concentration in local area, which has adverse influence on the potential engineering located near by the RGG. Thereby, an appropriate strength of the packfillings that not only can resists the disturbances during longwall panel mining but also be able to collapses itself after mining process finishes, should be proposed in the future.

2.3.3 Selection of the most appropriate location of the RGG packfillings

According to the locations of the RGG's packfillings in the previous longwall gateroad, there are usually three kinds of place for packfillings building, including (Model Ⅰ) located outside the previous gateroad, (Model Ⅱ) located inside the previous gateroad partly, and (Model Ⅲ) located inside the previous gateroad entirely, as shown in **Fig. 6**.

(a) Model Ⅰ   (b) Model Ⅱ   (c) Model Ⅲ
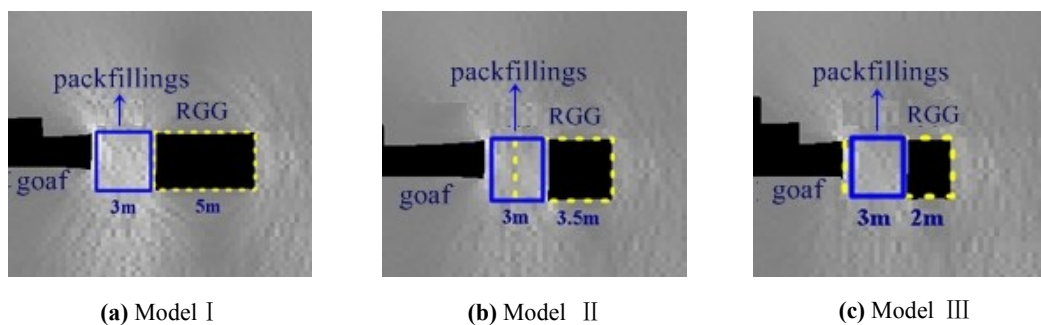
**Fig. 6** The relative position diagram of gateroad and packfillings

In order to compare the influence of these three locations on the stability of the RGG, relative considerations are taken into account during above simulation, and the results including shear stress distribution, acoustic emission and deformation vector are shown in **Fig. 7, 8** and **9**.
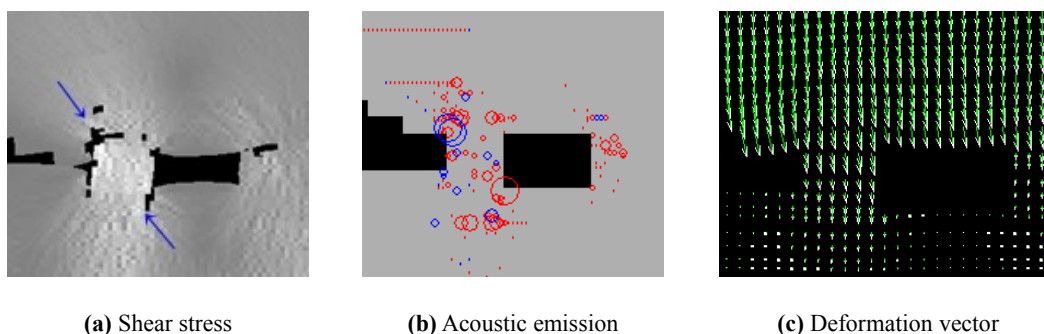
(a) Shear stress   (b) Acoustic emission   (c) Deformation vector

Fig. 7 Shear stress (a), acoustic emission (b) and deformation vector (c) in model Ⅰ

(a) Shear stress   (b) Acoustic emission   (c) Deformation vector

Fig. 8 Shear stress (a), acoustic emission (b) and deformation vector (c) in model Ⅱ
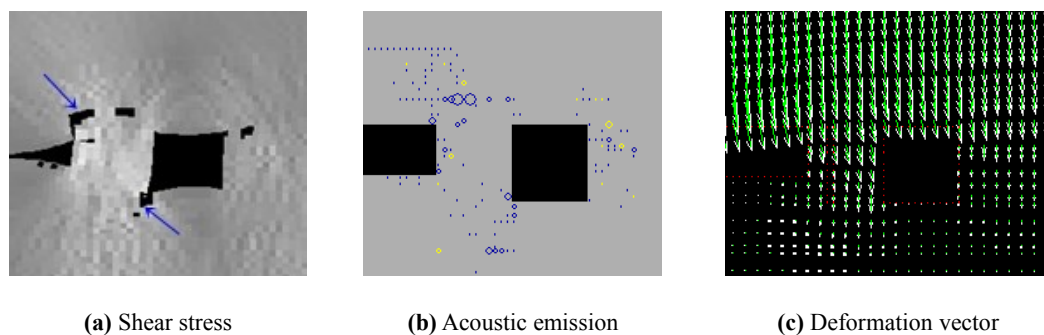
(a) Shear stress   (b) Acoustic emission   (c) Deformation vector
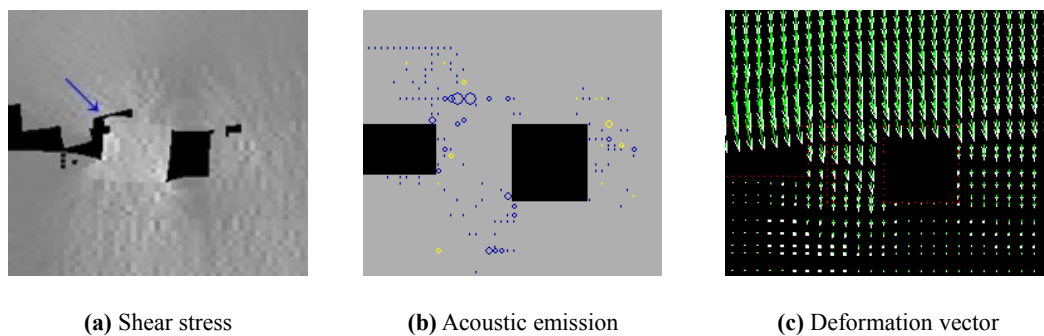
Fig. 9 Shear stress (a), acoustic emission (b) and deformation vector (c) in model Ⅲ

It can be seen from the **Fig. 7** that, the deformation of the RGG in model Ⅰ is very large.

Detailedly, in model Ⅰ, obvious roof sag and floor heave occur because of large span of the RGG (5 m). The rock mass located at top-left corner and lower-right corner of packfillings breaks significantly and large elastic energy is released due to high shear stress concentration that induced by the rotation movement of the over rock cantilever. In addition, the roof strata over this packfillings are always poorly supported before the packfillings building, which is another significant contribution to this kind breakage. When the packfillings is built inside the previous gateroad by 1.5 m, as shown in **Fig. 8**, roof sag and floor heave are reduced effectively, because that the span of the RGG is reduced to 3.5 m from 5 m. As expected, the rock mass at the two corners of the packfillings remain stable to some extent, although some breakage still happens results from shear stress concentration and elastic energy release. As the packfillings is built entirely in the previous gateroad (**Fig. 9**), the span is decreased continually to 2 m. The RGG experiences little deformations except for slight floor heave, and the rock mass at the tow corners of the packfillings are not broken almost. Above all, it can be concluded that, as the location of the packfillings moves into the previous gateroad, the space for ventilation and transportation is reduced gradually, and the stability of the RGG is easier to control. Considering that a relative large roadway section area is need to account for heavy gas emission and future roadway deformation, the model Ⅱ that the packfillings located partially in the previous gateroad is selected as the most appropriate one in the Pan Yidong coalmine.

3.   Supporting of the RGG in the Pan Yidong Coalmine

According to the above roof strata movements and structure analyses, it can be concluded that the RGG suffers several disturbances from the beginning of its excavation. Staged supporting technology should be considered according to different situations in different mining stages.

3.1 Basic supporting technology during gateroad excavating

After gateroad excavation, the stress in the surrounding rock distributes first time. Detailedly, the lateral confining stress in the shallow range of surrounding rock that perpendicular to surroundings surface is released and is smaller than the compressing stress that parallel to surroundings surface. Consequently, the bearing strength of roadway surrounding rock is reduced significantly, and deformation towards roadway space takes place, forming a range of broken and plastic zone around the roadway. A supporting technology that is capable of providing high confining stress timely after roadway excavation becomes crucial. For this, a bolting system with high prestress, high strength and high system stiffness ("three high") is employed [18]. High strength of the steel bolt rod provides the basement for high prestress during rock bolts installing and for high resistance during following serving life, and the high system stiffness broadens the controlling zone of the high prestress. Compared with conventional concrete lining supporting, steel arch supporting and conventional rock bolt supporting, this modified roadway stabilizing technology can provide high confining stress in roadway radial direction and therefore can prevent the potential deformation and cracks and bending

separation from occurring. The relationship between surrounding rock deformation and supporting strength of different supporting technology is illustrated in **Fig. 10** [18].
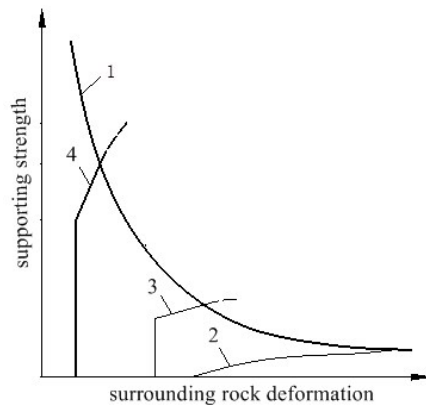
Fig. 10 Relationship between surroundings deformation and supporting strength techniques (1- classical deformation curve of roadway surroundings; 2- concrete lining and steel arch technology; conventional bolting technology; 4- "three high" bolting technology)

3.2 Reinforcing technology during longwall panels extracting

As analyzed in the above section 2.3, the RGG experiences abutment stress disturbance ahead of coal face and numerous cracks and fissures in the roadway surroundings during longwall panel mining, which makes mainly contribution to the instability of the RGG. In order to reinforce the weak surrounding rock under several dynamic stress disturbances, the grouting cable bolts that not only restrict rock mass deformation in a larger area but also inject grouts in cracks is designed, as shown in **Fig. 11**. The construction process of the grouting cable bolt in the RGG is illustrated in **Fig. 12**.
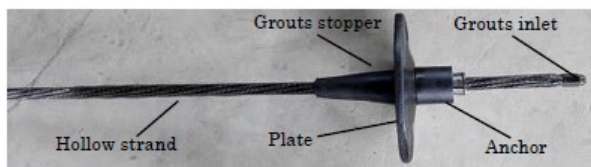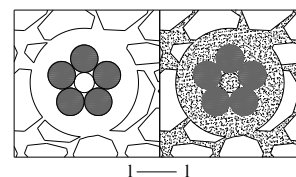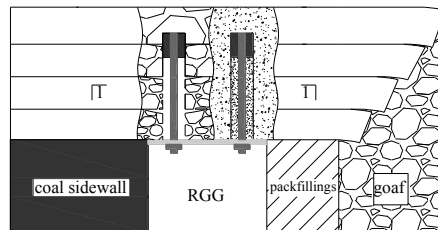

Fig. 11 Structure of the grouting cable bolt

Fig. 12 Construction process

3.2.1 Construction process of the grouting cable bolt

As shown in **Fig. 12**, the construction process consists of the following steps:

(1) Drilling the cable hole with ordinary mining drilling machine. The diameter of hole is 22 mm, and the length of holes ranges from 4 m to 10 m according to the geological condition of roadway surrounding rock.

(2) Assembling the grouting cable rod. Anchor cable strand with resin capsule, and then impose high prestress with special tensile jack.

(3) Setting the cement lining by shotcrete. In order to prevent the grouts with high pressure from flowing out when injection implementation in next stage, there is a need to seal the cracks and fissures on the surface of surrounding rock by shotcrete. The thickness of this lining is mainly governed by the pressure of grouting and the development and distribution of cracks and fissures.

(4) Injecting the grouts into surrounding rock through hollow pipe of the hollow grouting cable. In this stage, the space between the cable rod and holes wall, cracks and fissures in masses around the hole are sealed by grouts injection. Finally, the grouting pipelines are taken down and the grouting hole is sealed after the grouts solidification.
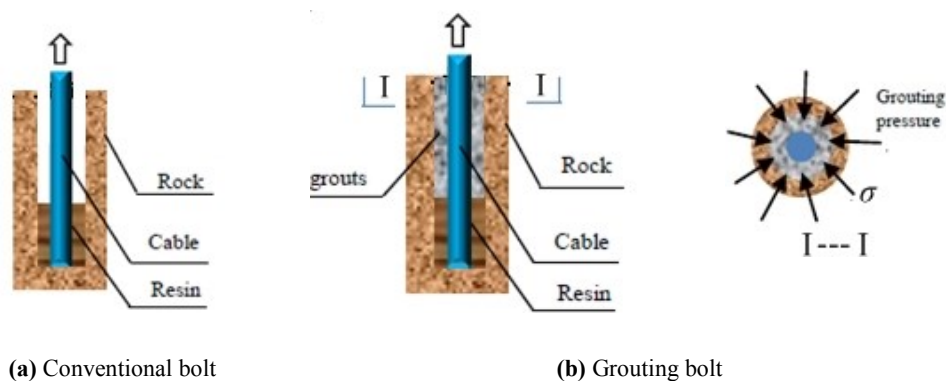
3.2.2 Improvements of the grouting cable bolt

Conventional cable bolt system always performs various failures after experiencing severe stress disturbances, especially in weak surrounding rock with cracks. Following are some popular failures measured in the field:

(1) When resin is mixed by rotating of cable rod, some fallen cracked rock will be mixed with resin, leading to the decrease of the strength of resin. As a result, resin will be destroyed under high shear stress or tension stress generated by relative movement between cable rod and surrounding rock.

(2) Resin cannot be bonded on the surface of surrounding rock effectively, duo to air, water and weak rock powder located in cracks. As a result, the interface between resin and surrounding rock will undergo breakage.

(3) The cracks of surrounding rock of cable hole will become more developed after drilling process and installing process. Consequently, the strength of shallow part of surrounding rock will be reduced severely and mechanical connection between cable rod and surrounding rock will disappear as well.

(a) Conventional bolt   (b) Grouting bolt

Fig. 13 Working mechanism of conventional bolt and grouting bolt

Grouting cable bolt has following improvements respects to above possible failures. According to the working mechanism of cable bolt (Fig. 13), shear resisting capacity of a cylinder shaped interface of cable structure can be calculated according to the following equation with the assumption that the

distribution of shear stress along cable length is uniform:

$$T = 2 \times \pi \times R \times L \times \tau \tag{1}$$

Where: T is the shear resisting capacity/N; R is the bonding radius/m; L is the bonding length/m; τ is the shear strength/MPa which can be determined according to equation (2).

$$\tau = \sigma \tan \varphi + c \tag{2}$$

Where: σ is the normal stress on the interface, equaling to the grouting pressure; φ is the friction angle of interface; c is the cohesion of interface.

According to equations (1) and (2), shear resisting capacity of a cable system can be improved by increasing bonding radius, bonding length, normal compressive stress, friction angle and cohesion of interface. In grouting cable bolt system, space between cable rod and surrounding rock is filled by cement with good performance, leading to the fully bonding along whole cable length. Additionally, high grouting pressure can compress the surface of interface, providing high normal confining stress to the interface. Thereby, the shear strength is increased as well. Eventually, the shear resisting capacity of cable is increased considerably.

Last but not the least, the grouts flow in the cracks and fissures in the surrounding rock, and replace the air, water and rock powder located in them. Mechanical parameters including inner frictional angel and inner cohesion are increased, and integrity of the cracked surrounding rock is improved, the bearing capacity of the grouted rock mass is enlarged as well. As a result, risk of roof caving, sidewall collapse and large deformation can be prevented.
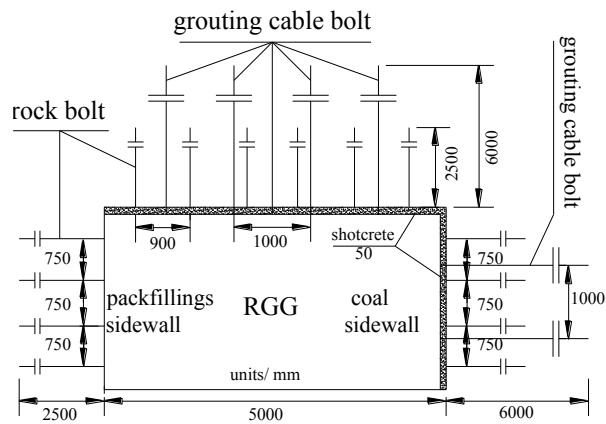
4.   Engineering Practice and Field Measurement

4.1 Engineering Practice

According above analyses, "three high" rock bolt was employed after roadway excavation, and then, grouting cable bolt was installed before longwall face passes by, to maintain the long-term stability of the RGG in the first longwall face of the Pan Yidong underground coalmine. The mining system layout is shown in **Fig. 14** and the detailed supporting parameters are illustrated in **Fig. 15**.
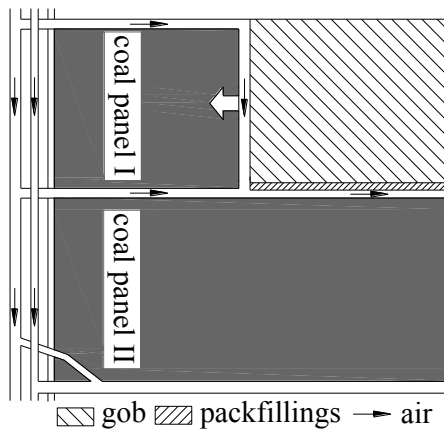
Fig. 14 Mining system layout   Fig. 15 Supporting system of the RGG

The grouts for the grouting cable bolt injection has a ratio of cement (425#) to water of 0.5, and a

chemical additive (ACZ—I) of 8% by weight. Grouting pressure is 2-3 MPa, grouting time is 3-5 minutes. The material for the concrete packfillings building is composed of cement 23%, fly ash 8%, gravel 45%, sand 23% and 1% chemical additives by weight. The testing uniaxial compressive strength of this building material is 15.6 MPa (1 day), 25.6 MPa (3 day), 30.5 MPa (7 day) and 37.0 MPa (28 day), respectively.

4.2 Field Measurement

One significant goal of the RGG is that it serves for adjacent longwall panel excavation. Maintaining a sufficient roadway section is a precondition of this goal achievement. To verify the effectiveness of this combined supporting technology on the stability of RGG, deformations of the RGG was measured, and the result is shown in **Fig. 16** and the final profile of the RGG after first longwall panel mining finishes is shown in **Fig.17**.
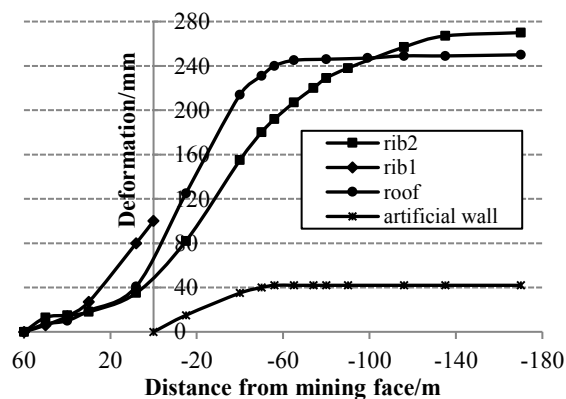
Fig. 16 Deformation curves of the target RGG

Fig. 17 Profile of the target RGG

As illustrated in **Fig. 16**, deformations of the RGG increase gradually as the relative distance between the measuring station and the coal face changes. After experiencing the stress disturbances from roadway and the first coal panel excavation, deformations of the roof, coal sidewall, rib 1 and the artificial packfillings were less than 250 mm, 270 mm, 100 mm and 40 mm, respectively. And the Final section area of the RGG (**Fig. 17**) was maintained up to 4000 mm×2800 mm (width × height). Parameters of this RGG satisfy all the requirements of safe mining.

5. Conclusions

(1) After longwall panel extraction, overlying strata will break and cave periodically. Abutment stress ahead of the coal face concentrates. Referring to the geological condition of the Pan Yidong coalmine, the largest abutment stress is generated by the main roof first breakage and the concentration factor is up to 2.3 with value of 43.7 MPa. Three zones including caving zone, cracking zone and bending zone form, which facilitate the released gas flowing and accumulating in goaf area. Roof cantilever over the RGG occurs and whose sink and rotation impose significant influence on the RGG

packfillings by vertical compression force and horizontal shear force. Partially located in the previous gateroad is selected as the most appropriate model for the packfillings building considering the difficult of maintenance and section requirement during next longwall panel mining.

(2) Combined supporting system is proposed to maintain the long-term stability of the RGG. The "three-high" rock bolt system is employed to provide high lateral confining stress during roadway excavation. Grouting cable bolt system is proposed to restrict the potential deformation and to improve the rock mechanics through large supporting capacity and grouts injection.

(3) With the application of this proposed combined supporting technology in the deep RGG, roadway convergence is maintained within an accepted limit and section area (4000 mm wide and 2800 mm high) is large enough for the second longwall panel mining.

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