Control mechanisms and design for a ‘coal-backfill-gangue’ support system for coal mine gob-side entry retaining

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Abstract: Gob-side entry retaining (GER) is an important technique for realising continuous development in coal mines in China. GER safety depends on the support of the coal wall, the gob-side backfill, the gangue in the goaf, and some auxiliary supports. However, field investigations have shown that the coal wall and gob-side supports fail easily during the second mining action, especially in the conditions of deep mining and hard main roof. To ensure the stability of a GER, this paper first establishes a deformation and control model for the 'coal-backfill-gangue' support system to determine the acceptable rotation angle of the lateral main roof. This model also determines the quantitative relationship between the roof pressure on the GER and the support forces provided by the support system. Then, based on the acceptable rotation angle of the lateral main roof, the quantitative design method for support system of the GER under different conditions is proposed. Finally, a field case study in the no. 16101 haulage roadway of the Binhu Coal Mine, China, is presented. After calculations, a 'flexible and rigid' composite material was used as the gob-side backfill and the specifications for the bolts in the coal wall were changed. Field observations showed that the improved supports can maintain stability of surrounding rocks of the haulage roadway. This research provides an effective design method for the 'coal-backfill-gangue' support system to be used in GER. [Received: May 20, 2017; Accepted: October 24, 2017]

Keywords: gob-side entry retaining; deformation; control mechanism; supporting design.

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1 Introduction

In underground coal mines, gob-side entry retaining (GER) is a type of roadway without coal pillars where the former entry roadway is retained as the return airway for the next mining panel. This is done by constructing an artificial wall along the gob-side lagging behind the working coal face. The application of this retaining technique can not only improve coal recovery rates and achieve huge economic benefits, it can also mitigate environmental pollution by recovering waste rock. Thus, it has been regarded as an important technique for improving development in coal mines and has been widely applied in China (Tan et al., 2015b; Ning et al., 2014; Zhang et al., 2014a, 2014b; Liu et al., 2016).

Artificial walls for GER are increasingly used in Chinese mines and these walls can be constructed using a number of different techniques (Gong et al., 2017; Huang et al., 2016). For instance, gob-side support with enter-in packing on its original location behind a fully-mechanised coalface, gob-side support in situ stratified filling, and gob-side support with whole casting have all been used (Tan et al., 2015b; Bai et al., 2015). There are five ways to construct a gob-side pack: concrete block, waste rock, gangue concrete, cemented filling, and high water-content rapidly-solidifying material. Cemented filling material is the most popular construction method in China. However, field investigations have shown that these gob-side packs are weak and fail to support the roadway roof when the adjacent panel is mined. These GER techniques are not effective in many coal mines. Thus, studies to determine better support methods are necessary for GER techniques to be more successful (Ning et al., 2014; He et al., 2012; Lu et al., 2015; Tan et al., 2015b).

After reviewing previous investigations, it was found that the safety of the gob-side entry depends on the coal wall, the gob-side backfill, and the gangue in the goaf. These three pars form a support system to bear the weight of the roof strata and maintain the stability of the GER (Zhang et al., 2012; Zhou et al., 2012). Any single component or pair of components of the support system, the coal wall, the backfill, or the gangue, cannot support the roof effectively by themselves. The failure of any one of these components will result in the failure of the GER (Su et al., 2015; Fan et al., 2014; Xue et al., 2013). Owing to there being significant differences in the bearing performances of the coal, backfill and gangue, one of them may become unstable from overloading during the roof movement while the others are far short of their ultimate carrying capacity (Zhang et al., 2005, 2014b; Zhang and Zhao, 2001). Gong et al. (2017) researched the control technology of GER with gangue backfilling method. Wang et al. (2016) obtained the deformation mechanisms of lateral roof influenced by mining-induced fault population activation. However, the deformation relationships among the coal, backfill and gangue have not been researched sufficiently. And their strength and deformability were not designed as a whole support system to make full use of their bearing capacities and ensure the stability of GER.

In this paper, in order to make full use of the bearing capacities of the coal, backfill and gangue, we first establish a deformation and control model for the 'coal-backfill-gangue' support system. Then, a quantitative design for the support system is proposed. Finally, a field case study of the no. 16101 haulage roadway in the Binhu Coal Mine, Shandong Province, China, is presented.

2 Deformation and control model for a 'coal-backfill-gangue' support system

2.1 Mechanical model

Observations in mines have shown that as mining progresses, part of the immediate lateral roof collapses to become gangue in the goaf after the coal face has been mined. Another part of the roof above the roadway forms rock beam A and is supported by the coal wall, cemented backfill, and other primary supports like bolts (Figure 1) (Tan et al., 2015b, 2017; Wang et al., 2015). The lateral main roof in the roadway breaks and forms rock beam B which then sinks rotationally along the fracture line until the gangue is compressed (Chen et al., 2016; Wang et al., 2016). The overlying strata bend and subside directly onto the main roof. During and after roof movement, the coal wall, backfill, and gangue bear the weights of the roof and the overlying strata and are the main supports maintaining the stability of the GER (Tan et al., 2015a, 2015b; Gong et al., 2017), as shown in Figure 1.

Figure 1 shows that the bolted coal, cemented backfill, and gangue of the GER will be compressed and deformed when the lateral roof moves. For this model, it is assumed that rock beams A and B are pinned using cables and bolts and subside together rotating along the fracture line in the lateral main roof. In Figure 1, the horizontal distance between the fracture line and the coal wall edge is L_0, the widths of the roadway and backfill are a and b, respectively, the length of rock beam B in the main roof is L_1, and the main roof rotation angle is θ. The compressive deformation of rock beams A and B are ignored during roof movement but the deformation at the edge of the coal, Δh_M, the deformation at the centre of the backfill, Δh_F, and the subsidence at the end of rock beam B, Δh, must meet the following trigonometric relationship:

$$\tan\theta = \frac{\Delta h_M}{L_0} = \frac{\Delta h_F}{L_0 + a + b/2} = \frac{\Delta h}{L_1 \cos\theta} \tag{1}$$

where $\Delta h = \Delta h_G + h - (K_m - 1)m_z$

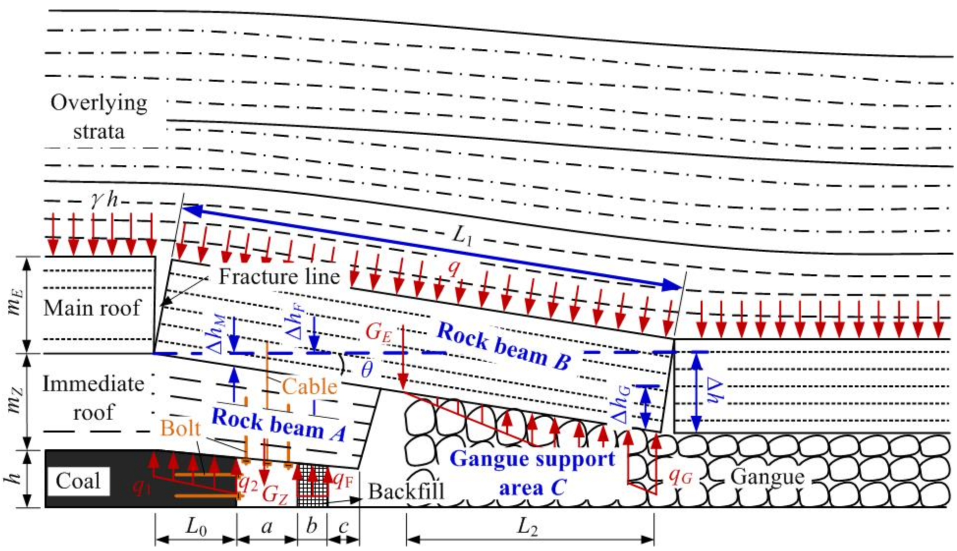
Meanwhile, the coal, backfill, and gangue must meet specific bearing requirements (Figure 1). They should be able to carry the weight of rock beam A, G_Z, the weight of rock beam B, G_E, and the force from the overlying strata, q. Presuming that the force supported by the coal is a trapezoidal distributed load ranging from q_1 to q_2, the force supported by the backfill is a uniformly distributed load q_F, and the force supported by the gangue is a triangular distributed load ranging from zero to q_G, then the total supporting forces after roof movement should satisfy equation (2).

$$\frac{q_1+q_2}{2}L_0+q_Fb+\frac{q_G}{2}L_2\geq G_Z+G_E+qL_1\cos\theta \tag{2}$$

$$L_2=\min\left\{L_1\cos\theta-L_0-a-b-c,\ L_1\cos\theta-\frac{(h-K_mm_Z)}{\tan\theta}\right\} \tag{3}$$

Equations (1) and (2) are the equations for the deformation and control of the 'coal-backfill-gangue' support system. And the coal, backfill, and gangue can deform together and share the load when the two equations are satisfied. On the other hand, if the equations are not satisfied, the coal or backfill would be destroyed, resulting in un-stability of the GER, or the roadway deforms too large, results in that the roadway cannot meet the requirements of transportation and/or ventilation.

Figure 1 Structural model for and forces on the rocks surrounding a GER under a hard roof (see online version for colours)

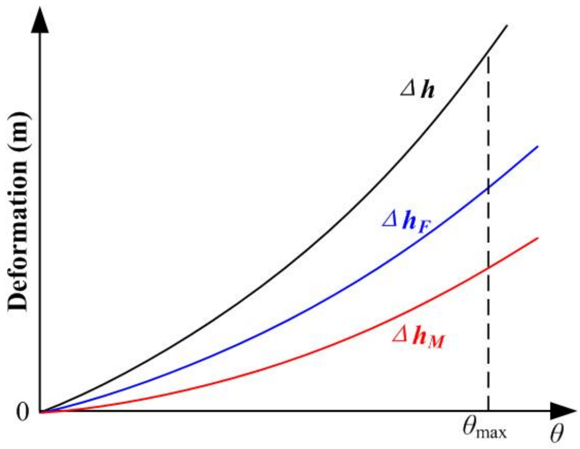


Source: Tan et al. (2015b) and Gong et al. (2017)

2.2 Acceptable rotation angle for the lateral main roof

From equation (1), we can determine how the deformation of the coal and the backfill, Δh_M and Δh_F, and the subsidence at the end of rock beam B, Δh, change with the rotation angle, θ. Figure 2 shows that the deformation of the coal, backfill, and gangue increase with the rotation angle. However, if the deformation of the coal and backfill are larger than is acceptable, the supporting structure will be destroyed, resulting in failure of the GER. Because the roadway is required for transportation and ventilation, the gob-side entry cannot be allowed to become too small. Thus, it is necessary to restrict the maximum rotation angle of rock beam B.

Figure 2 Changing deformation in the coal, backfill, and gangue (see online version for colours)



2.2.1 *Acceptable subsidence for the end of rock beam B*

Both field practices and laboratory tests show that the bearing capacity of the gangue increases as it deforms, but the deformation cannot increase forever (Ma et al., 2011; Schumacher and Kim, 2014; Ning et al., 2013; Deng and Wang, 2014). Considering the compaction coefficient of the gangue in the goaf, K_A, acceptable deformation for the gangue, Δh_{Gmax}, can be approximated by:

$$\Delta h_{G\max} = (K_m - K_A) m_Z \tag{4}$$

From equation (4), the acceptable subsidence at the end of rock beam B, Δh_{max}, should be:

$$\Delta h_{\max} = h - (K_A - 1) m_Z \tag{5}$$

2.2.2 *Acceptable deformation for the coal and backfill*

To ensure stability of the GER, the stress borne by the coal and backfill cannot exceed their peak strength, so the acceptable deformation of the coal, Δh_M, and backfill, Δh_F, can be expressed as:

$$\Delta h_{M\max} = h_1 \varepsilon_{CM} \tag{6}$$

$$\Delta h_{F\max} = h_2 \varepsilon_{CF} \tag{7}$$

2.2.3 *Acceptable maximum rotation angle*

Considering that the gob-side entry has to meet the requirements of an active coal face for transportation, ventilation, and other functions, the size of the roadway should not be reduced more than 25%. With this limitation, the acceptable maximum rotation angle of rock beam B, θ_{max}, can be expressed as:

$$\theta_{W\max} = \arctan\frac{0.25h_3}{L_0 + a + b/2} \tag{8}$$

Substituting equations (5)–(7) into equation (1) shows that the maximum rotation angle of the lateral main roof is determined by the maximum acceptable subsidence at the end of rock beam B and this determines the amount of deformation of the gangue, coal, and backfill ($\theta_{G\max}$, $\theta_{M\max}$, and $\theta_{F\max}$). Combined with $\theta_{W\max}$, the acceptable rotation angle of rock beam B, θ_{max}, can finally be defined as:

$$\theta_{\max} = \min\left\{\theta_{G\max}, \theta_{M\max}, \theta_{F\max}, \theta_{W\max}\right\}. \tag{9}$$

2.3 GER roof support design

Because the bearing capacity of the gangue in the goaf is greater than that of the coal and backfill, the gangue has the most influence on the angle to which rock beam *B* rotates. The deformation of the gangue can be partially controlled by cutting the roof next to the roadway. Thus, during the design of the GER supports, $\theta_{G\max}$ should be used as a reference value. When planning a mine, if the geological and mining parameters of the coal face are known, including the mining height, roadway widths, roof thicknesses, etc., the support system can be designed according to the following steps:

Step 1 From equation (1) and equations (4)–(8), the maximum rotation angle of rock beam B and the acceptable subsidence at the end of rock beam B along with the acceptable deformation of the gangue, coal, and backfill and the size requirements of roadway section can be obtained.

Step 2 If $\theta_{max} = \theta_{G\max} \leq \min\{\theta_{M\max}, \theta_{F\max}, \theta_{W\max}\}$ [Figure 3(a)], move to Step 5.

Step 3 If $\theta_{max} = \theta_{M\max} \leq \min\{\theta_{G\max}, \theta_{F\max}, \theta_{W\max}\}$, the subsidence of the lateral main roof is excessive. If this is the case, the lateral roof can be cut to increase the height of the gangue in the goaf so as to decrease the subsidence of the lateral main roof, Δh_{max}, to $L_1\sin\theta_{M\max}$. This will cause the maximum rotation angle, $\theta_{G\max}$, to decrease to $\theta_{M\max}$ [Figure 3(b)]. Alternatively, one can increase $\theta_{M\max}$ and $\theta_{F\max}$ to $\theta_{G\max}$ by taking certain steps to decrease the angle of subsidence of the roof, increase the amount of support in the coal (by using bolts or cables) or use ‘flexible and rigid’ composite materials as the backfill to increase the bearing capacities of the coal and backfill [Figure 3(c)]. These three steps that can be taken to decrease lateral main roof subsidence are discussed in more detail below.

1 Roadway roof cutting

The subsidence of the lateral main roof, $\Delta_{h\max}$, can be reduced to $L_1\sin\theta_{M\max}$ by cutting the lateral roof. According to equations (1), (4) and (5), the cut thickness (mc) should be:

$$m_C = \frac{h - L_1\sin\theta_{M\max}}{K_A - 1} - m_Z \tag{10}$$

Now the maximum rotation angle of the lateral main roof determined by its acceptable subsidence, $\theta_{G\max}$, is decreased to $\theta_{M\max}$, and the rotation angles can meet the relationship $\theta_{G\max} = \theta_{\max} = \theta_{M\max} = \leq \min \{ \theta_{F\max}, \theta_{W\max}\}$.

2 Gob-side backfill design

Using 'flexible and rigid' composite materials as the backfill can greatly improves its capacity to resist deformation (Tan et al., 2015b; Ning et al., 2014; Cheng et al., 2012). From equations (1) and (7), the minimum amount of compression provided by the flexible material ($\Delta' h_F$) should be:

$$\Delta h'_F = \left(L_0 + a + \frac{b}{2} \right) \tan \theta_{G\max} - (h_2 - h_4)\, \varepsilon_{CF} \tag{11}$$

3 Specifications for coal supports

The capacity of the coal to resist deformation can be significantly increased by changing the specifications for the coal supports, including the length of bolts, bolt intervals, the length and intervals of anchor cables, and bolting and grouting procedures. (Zhao et al., 2012). From equations (1) and (6), the minimum acceptable deformation of the bolted coal should be:

$$\Delta h' = L_0 \tan \theta_{G\max} \tag{12}$$

Then the strain of the bolted coal at peak stress should be:

$$\varepsilon'_C \geq \frac{L_0 \tan \theta_{G\max}}{h} \tag{13}$$

By testing the bearing performance of the bolted coal with different supports, the optimum support configuration for strain ε'_c can be determined.

Step 4 If $\theta_{\max} = \theta_{F\max} \leq \min \{ \theta_{G\max}, \theta_{M\max}, \theta_{W\max}\}$, it is necessary to conduct lateral roof cutting or to improve the capacities of the backfill and coal to resist deformation using the measures mentioned in Step 3. The cutting thickness for the lateral roof can be calculated from equation (14), and the support capacities for the backfill and coal can be designed on the basis of equations (11) and (13).

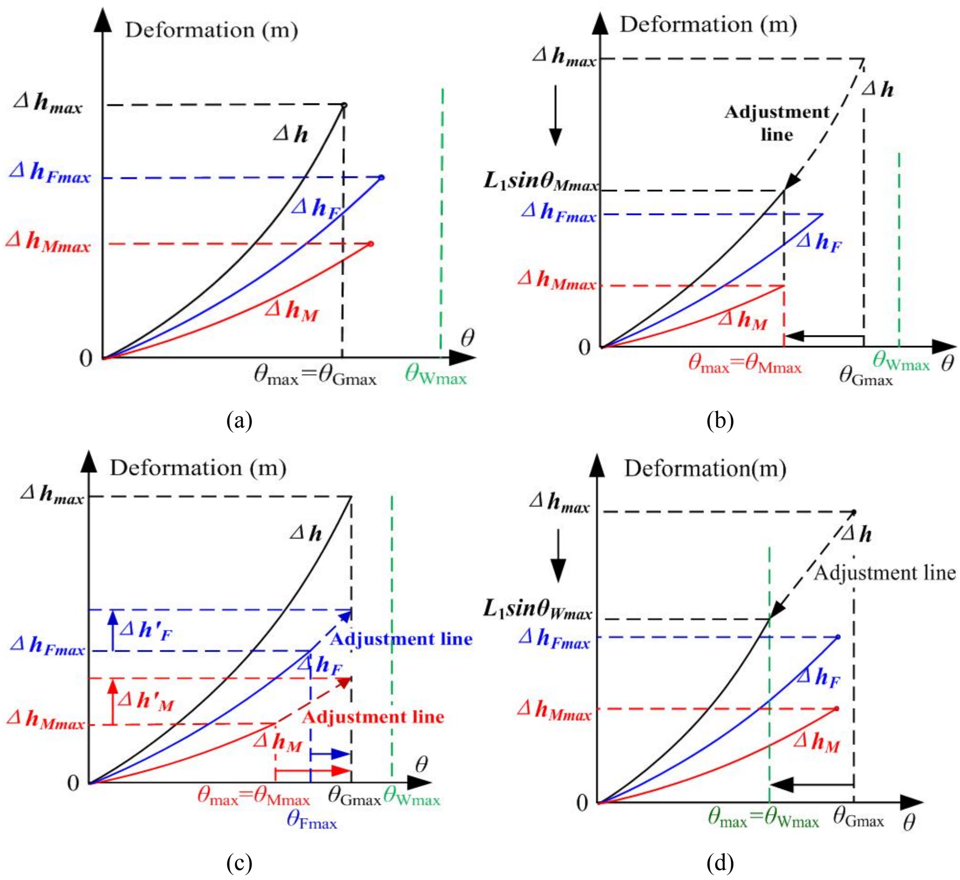
$$m_C = \frac{h - L_1 \sin \theta_{F\max}}{K_A - 1} - m_Z \tag{14}$$

Step 5 If $\theta_{\max} = \theta_{W\max} \leq \min \{\theta_{G\max}, \theta_{M\max}, \theta_{F\max}\}$, it will be necessary to cut the roof in the goaf to reduce the subsidence of the lateral main roof, $\Delta h_{\max}$, to $L_1 \sin\theta_{M\max}$. Again, the value of $\theta_{G\max}$ will be reduced to $\theta_{W\max}$ [Figure 3(d)]. The cutting thickness of the lateral roof should be:

$$m_C = \frac{h - L_1 \sin \theta_{W\max}}{K_A - 1} - m_Z \tag{15}$$

Step 6 After the rotation angle of rock beam B, $\theta_{\max}$, is determined, the maximum deformation of the coal, backfill, and gangue can be computed by introducing $\theta_{\max}$ into equation (1). Whether the support forces of the coal, backfill, and gangue are enough to maintain the safety of the GER or not can be determined by substituting their bearing strengths into equation (2). The support design is valid when equation (2) is satisfied. If equation (2) is not satisfied, steps should be taken to improve the bearing capacities of the coal, backfill, and gangue as necessary to satisfy equation (2).

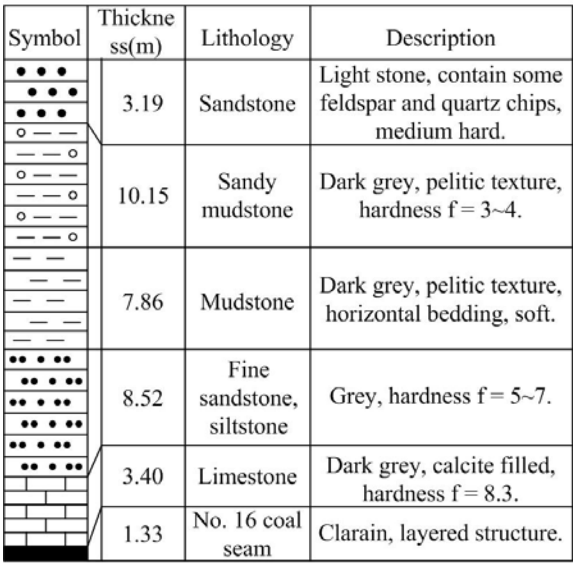
Figure 3 Designs for surrounding rock support in the GER, (a) $\theta_{G\max} \leq \min\{\theta_{M\max}, \theta_{F\max}, \theta_{W\max}\}$ (b) $\theta_{M\max} \leq \min\{\theta_{G\max}, \theta_{F\max}, \theta_{W\max}\}$ (adjust the deformation of the gangue in the goaf) (c) $\theta_{M\max} \leq \min\{\theta_{G\max}, \theta_{F\max}, \theta_{W\max}\}$ (adjust the deformation of coal and backfill) (d) $\theta_{W\max} \leq \min\{\theta_{G\max}, \theta_{M\max}, \theta_{F\max}\}$ (see online version for colours)



2.4 Case overview

This field study was conducted on the no. 16 coal seam in the Binhu Coal Mine operated by the Shandong Energy Zaozhuang Mining Group Co., Ltd., China. The average depth of burial of the no. 16 seam is about 531 m and the seam is 1.3–1.4 m thick with an average thickness of 1.33 m. The seam dips between 2° and 6°. The immediate roof is a 3.4-m-thick dark grey limestone with a density of 2,835 kg/m^3. The lateral hanging length of the immediate roof in the goaf is zero. The initial bulking coefficient of the gangue in the goaf, K_m, is 1.36, and the compaction coefficient, K_A, is 1.22 (obtained from the bearing performance test of the gangue in the goaf described in the next subsection, 'field and laboratory tests'). The main roof is an 8.52-m-thick bed of interbedded fine sandstone and siltstone, density 2,784 kg/m^3. Field observations show that the breaking length of rock beam B, L_1, is 16.8 m and the horizontal distance between the fracture line and the coal wall edge, L_0, is 3 m (Ning et al., 2013). There are 7.86-m-thick mudstones and 10.15-m-thick sandy mudstones with an average density of 2,631 kg/m^3 above the main roof (Figure 4). The no. 16101 panel is the first panel to be mined in the no. 16 coal seam; its average strike length is 651 m and the average incline length is 181 m. Figure 5 shows the layout of the panel. The coal mining was fully mechanised, and the roadway was excavated along the roof. When the panel was mined, the no. 16101 haulage roadway was preserved to serve as the ventilation roadway for no. 16103 panel. The retained roadway was 3.4 m wide and 2.0 m high (Figure 6). The roadway support parameters were: bolt diameter 18 mm; bolt length 2 m; bolting interval 900 mm (horizontally and vertically).

Figure 4 No. 16 coal seam roof strata, Binhu Coal Mine



Symbol	Thickness(m)	Lithology	Description
	3.19	Sandstone	Light stone, contain some feldspar and quartz chips, medium hard.
	10.15	Sandy mudstone	Dark grey, pelitic texture, hardness f = 3~4.
	7.86	Mudstone	Dark grey, pelitic texture, horizontal bedding, soft.
	8.52	Fine sandstone, siltstone	Grey, hardness f = 5~7.
	3.40	Limestone	Dark grey, calcite filled, hardness f = 8.3.
	1.33	No. 16 coal seam	Clarain, layered structure.

At an early stage, 2 m wide cemented material was used as the backfill beside the no. 16101 haulage roadway for support. The filling material proportions are listed in Table 1. The cement mixing proportion has great influence on the strength of the filling material. With the increasing of cement mixing proportion, its strength increased accordingly (Ning et al., 2015). After mining of the no. 16101 panel had been completed, the roadway roof subsided significantly and many large cracks appeared in both the coal and backfill. It was clear that the roadway support needed to be reinforced to ensure the stability of the surrounding rocks.

Table 1 Mixing proportions for initial filling material used beside the no. 16101 roadway

Amount of each kind of material in the backfill (kg/t)							*Granularity (mm)*
Water	*Cement*	*Coal ash*	*Gangue particles*	*River sand*	*Early strength agent*	*Water reducing agent*	
134	164	44	85	227	2.8	3.2	5 to 10

Figure 5 Map showing the layout of the no. 16101 mining panel, Binhu Coal Mine (see online version for colours)

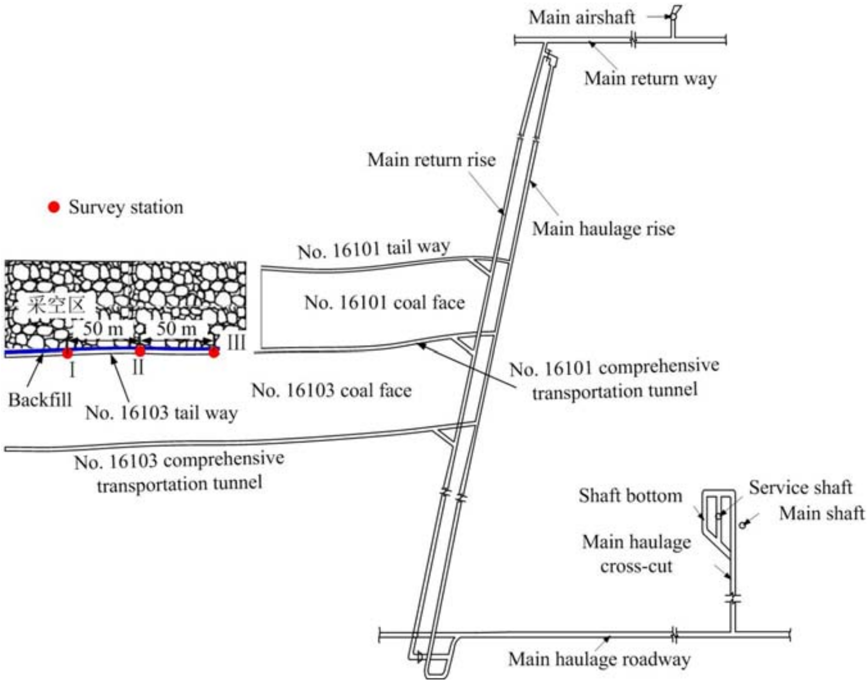
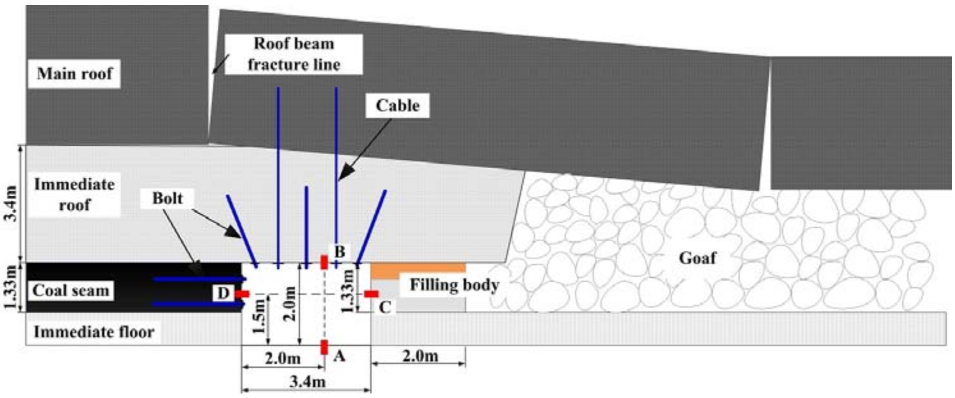


Figure 6 Section showing the no. 16101 haulage roadway. The letters A–D indicate the location of survey monitoring points (see online version for colours)



3 Results and discussion

3.1 Field and laboratory tests

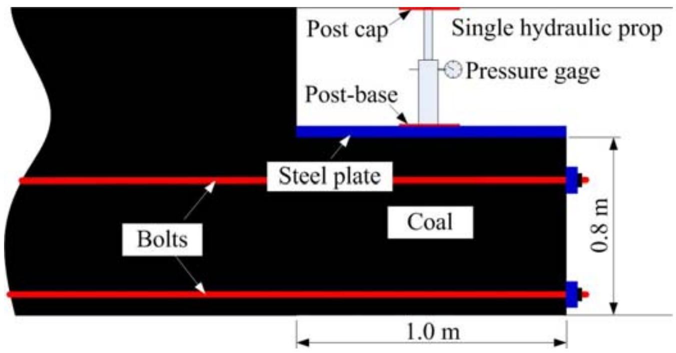
3.1.1 Bolted coal compression test

To obtain the bearing performance of the bolted coal wall, loading tests were performed in the no. 16101 haulage roadway (Figure 7). After the 16101 portion of the coal seam was mined out, a 0.8-m-thick wall of bolted coal was left. An area of bolted coal measuring 1.0 m × 1.0 m × 0.8 m was used for the test sample. The area chosen was one whose surface was the flattest one exposed. Next, a 1.0 m × 1.0 m × 0.05 m steel plate, post-bases, a single hydraulic prop, and post caps were placed against the test area. The single hydraulic prop could apply load to the coal. Finally, the hydraulic pressure in the prop p and the corresponding deflection of the coal Δx were recorded. The stress and strain of the bolted coal wall can be calculated according to equation (16), and the stress-strain curve is shown in Figure 8.

$$\sigma = p\frac{\pi D^2}{4\times 0.8}, \quad \varepsilon = \frac{\Delta x}{0.8} \tag{16}$$

Figure 8 shows that the peak stress of the bolted coal was around 14.25 MPa, the corresponding strain was about 0.0225.

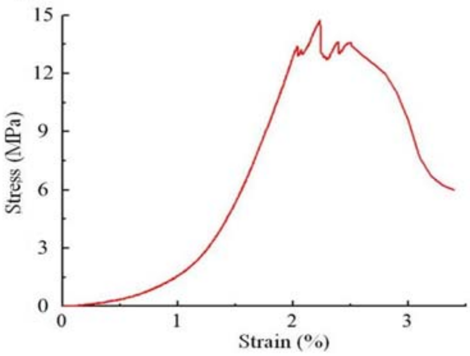
Figure 7 In-situ loading system for the bolted coal (see online version for colours)



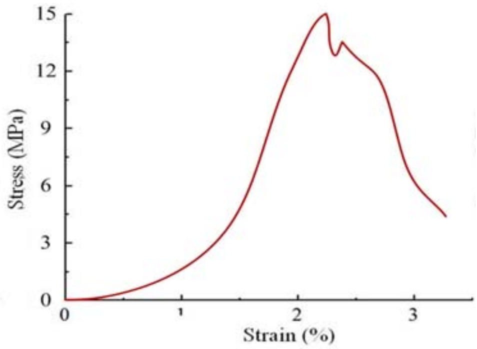
3.1.2 Backfill uniaxial compression tests

Test specimens, cubes measuring 100 mm on each side, were made in the laboratory (Table 1, Figure 9). The test specimens were put into a curing chamber at 25°C and a humidity of 90% for 28 days. After that, the specimens were subjected to uniaxial loading (Figure 10). The peak stress q_F and the corresponding strain ε_{CF} of the backfill were around 20.8 MPa and 0.048, respectively.

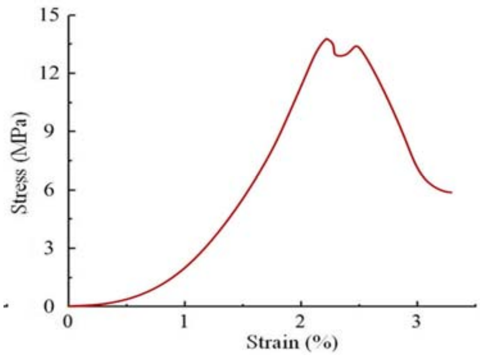
Figure 8 Stress-strain curves: bolted coal in the no. 16 coal seam, Binhu Coal Mine, (a) position A (b) position B (c) position C (see online version for colours)



(a)



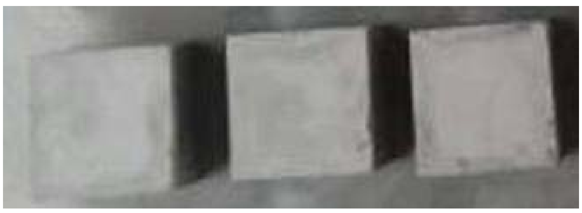
(b)



(c)

Figure 9 Backfill test specimen paste filling material, (a) before form removal (b) after form removal

(a)



(b)

3.1.3 Bearing performance test of the gangue in the goaf

To acquire a bearing capacity for the gangue in the goaf, a mould – a cube 600 mm on each side – was made using Q350 stainless steel plates. The gangue specimens were obtained from goaf near the no. 16101 roadway through crossheadings. The mass percentages of gangue with different sizes were counted according to the *Specification* of rock test (SL237-19699). Particle sizes for the gangue are shown in Table 2, and the testing device into which the samples were loaded is shown in Figure 11. The loading tests used a WE-1000C hydraulic testing machine with a loading rate of 0.011 mm/s to acquire the relationship between the bearing capacity of the gangue and the compression ratio (Figure 12). Figure 12 shows that the bearing capacity of the gangue increased exponentially with the compression ratio. The maximum compression ratio of the gangue was about 0.1029, and the corresponding maximum deformation $\Delta h_{G\max}$ was around 0.476 m. This means that the compaction coefficient of the gangue was 1.22.

Figure 10 Stress-strain curve from the backfill paste filling material uniaxial compression tests (see online version for colours)

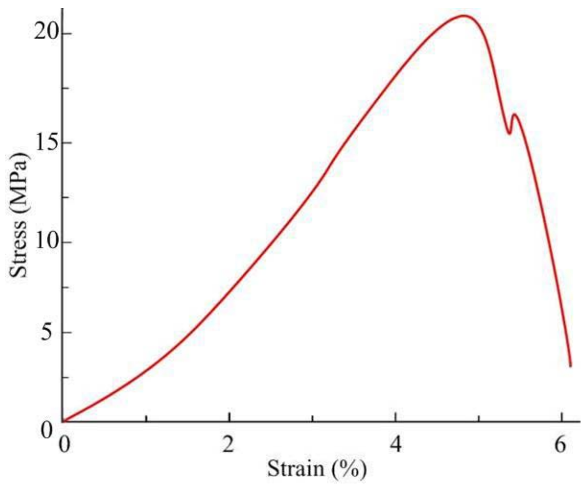


Figure 11 Load testing device and mould for gangue (see online version for colours)

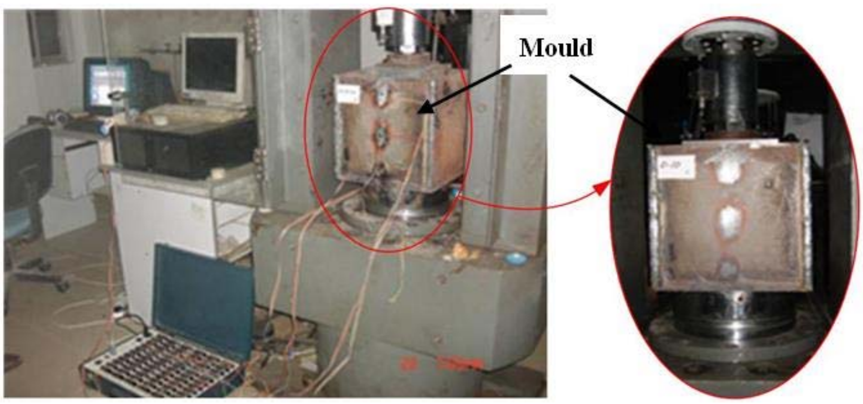
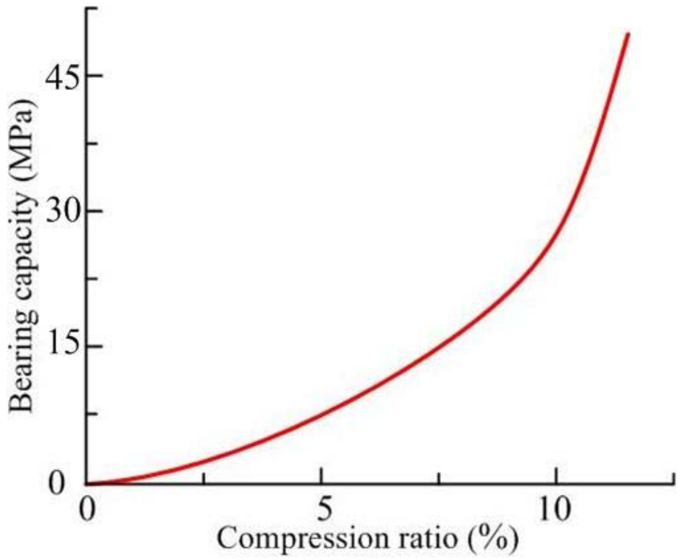


Table 2 Particle size the gangue used for bearing capacity tests

Particle size/mm	*> 60*	*40–60*	*20–40*	*10–20*	*5–10*	*2–5*	*< 2*	*Totality*
Quality/g	3,005	3,768	5,775	4,481	2,680	2,368	1,817	23,894
Percentage/%	12.58	15.77	24.17	18.75	11.22	9.91	7.60	100

Figure 12 Baring capacity and compression ratio for gangue in the goaf of the no. 16 coal seam, Binhu Coal Mine (see online version for colours)



3.2 Support system optimisation

3.2.1 Determination of the acceptable rotation angle

Substituting the appropriate parameters obtained from the tests just described into equations (4)–(7), we can determine that the deformation of the coal, backfill, and gangue are 0.030 m, 0.0638 m and 0.476 m, respectively. Then substituting these deformations into Eq. (1), the maximum rotation angles, $\theta_{M\max}$, $\theta_{F\max}$ and $\theta_{G\max}$, are be 0.57274°, 0.494° and 1.9853°, respectively. According to equation (8), the maximum rotation angle required by the roadway section $\theta_{W\max}$ is 4.2624°.

Because $\theta_{F\max} < \min \{\theta_{G\max}, \theta_{M\max}, \theta_{W\max}\}$, the requirements for the deformation and co-bearing of the support system have not been met. Referring to Step 4 in Part 3 of the 'Design' section, above, it would be necessary to either cut the roof or improve the capacities of the coal and backfill to resist deformation to make $\theta_{\max} = \theta_{G\max}$.

3.2.2 Artificial control of the support system

The priority for the support system is to improve the deformation capacities of both the coal and backfill to ensure that $\theta_{\max} = \theta_{G\max}$. The capacity of the backfill to resist deformation can be improved by using flexible and rigid composite materials (Tan et al., 2015b), and that of the coal can be enhanced using bolts and cables. According to the equations (11)–(13), the required deformation provided by the flexible materials $\Delta h_F'$ should be not less than 0.193 + 0.048h^4 m, and the minimum strain at peak stress for the bolted coal should be 0.0782.

3.2.3 Support design for the surrounding rocks

a Backfill support

'Youle I' filling material developed by Uroica Mining Safety Engineering Co., Ltd. (Tai'an, Shandong, China) was used as the flexible materials beside the roadway. The stiffness of the material is much lower than that of rigid materials and its maximum compress coefficient is 0.5. The required thickness of the flexible filling materials h_4 can be computed from equation (17):

$$\Delta h_F' = 0.5 h_4 \quad (17)$$

The thicknesses of the flexible and rigid materials beside the roadway should be 0.427 m and 0.903 m, respectively.

b Coal support

To obtain the optimal support parameters for the coal, loading tests were performed on the coal in the no. 16101 haulage roadway with different support schemes. Six different support schemes were tested, and they are listed in Table 3. The accompanying stress-strain curves for the different support schemes are shown in Figure 13.

As shown in Figure 13, the capacity of the coal to resist deformation can meet the requirements for roof movement by using support Schemes 3 and 5. Owing to Scheme 3 being more economical than Scheme 5, Scheme 3 was used as the support scheme for the coal, with a bolt diameter of 20 mm, a bolt length of 2.2 m, and a bolting interval of 0.6 m × 0.7 m in a trefoil pattern.

c Bearing capacity verification

The coal stress can be approximated by the virgin stress of the coal. A ZLGH type borehole stress metre produced by Luosaier Sensor Technology Co., Ltd., Shandong Province, China was used to determine that the virgin stress for the coal is approximately 12.85 MPa. From Figure 13(c), it can be seen that when the strain on the bolted coal is 0.0782, the corresponding stress is 18.3 MPa. Therefore, the support pressure at the edge of the coal could also be approximately 18.3 MPa. The experiments showed that the stress on the backfill was 20.8 MPa and the maximum stress on the gangue in the goaf was 30.5 MPa. Previous geologic investigations had shown that the stress acting on the main roof was around 6.19 MPa and the bearing width of the gangue in the goaf was 4.7 m. By substituting these values into the left-hand side of the equation (2), it can be shown that when the rotation angle of rock beam B peaks, the bearing capacities provided by the coal, backfill, and gangue will be 1.45 × 105 kN. By substituting the values into the right-hand side of the equation (2), it can be shown that the roof weight that needs to be supported is

1.09 × 105 kN. Thus designing the optimal support for the no. 16101 haulage roadway in the Binhu Coal Mine had been accomplished.

Figure 13 Bearing performance of the coal under different support schemes, (a)–(f) schemes 1–6 (see online version for colours)

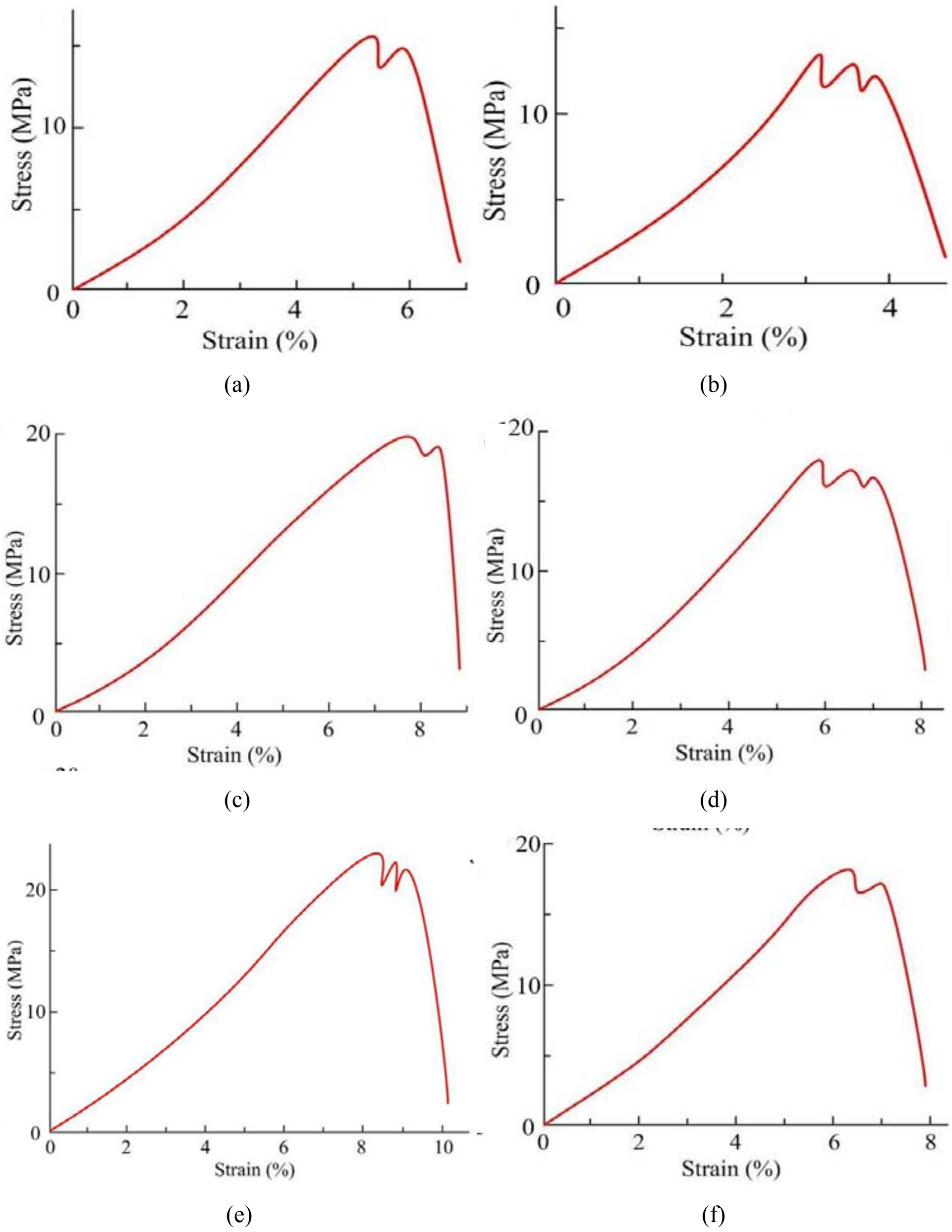
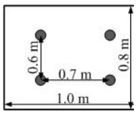
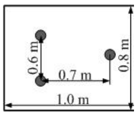
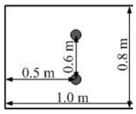


Table 3 Different support schemes considered for the no. 16101 haulage roadway

Schemes	*Scheme 1*	*Scheme 2*	*Scheme 3*	*Scheme 4*	*Scheme 5*	*Scheme 6*
Bolt parameters	Diameter:18 mm, length: 2.2 m	Diameter:18 mm, length: 2.2 m	Diameter: 20 mm, length: 2.2 m	Diameter: 20 mm, length: 2 m	Diameter: 22 mm, length: 2.2 m	Diameter: 22 mm, length: 2 m
The layouts of sections			Similar with scheme 2		Similar with scheme 2	Similar with scheme 4

3.3 *Field observations and analysis of the new support scheme*

The support scheme for the no. 16101 haulage roadway was modified according to the above design, and a field surveying program was carried out to assess the design (Figure 14). Three survey stations, stations 1, 2 and 3, were set up to monitor the effects of the new supports on the retained roadway. The space between adjacent stations was 50 m (Figure 5). At these stations, a crossing method was used to monitor deformation in the surrounding rock. Four base points designated as points A, B, C, and D were set in the floor, roof, backfill and coal (Figure 6).

Figure 14 Processes for the filed filling construction, (a) plan graph (b) section plane along line A-A (see online version for colours)

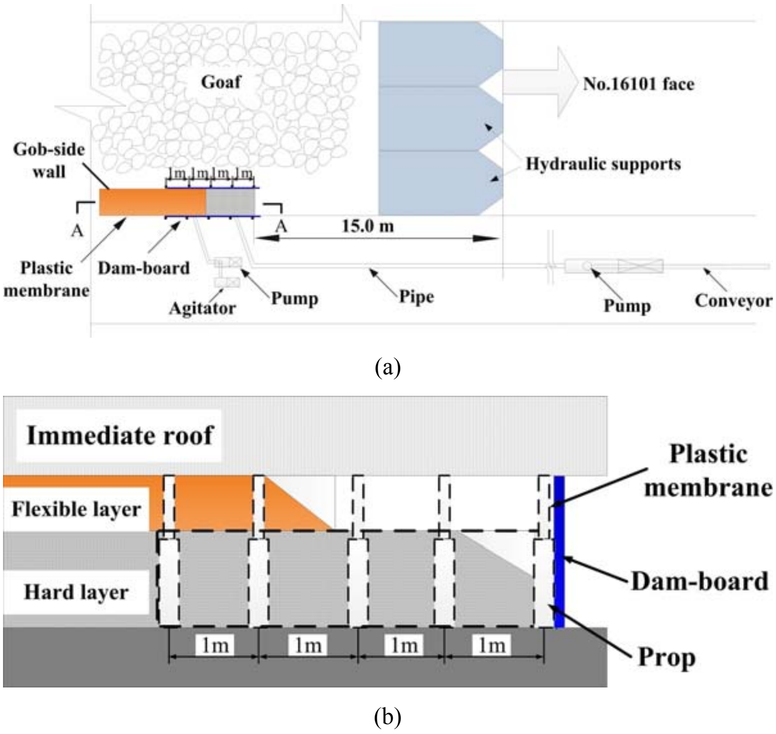
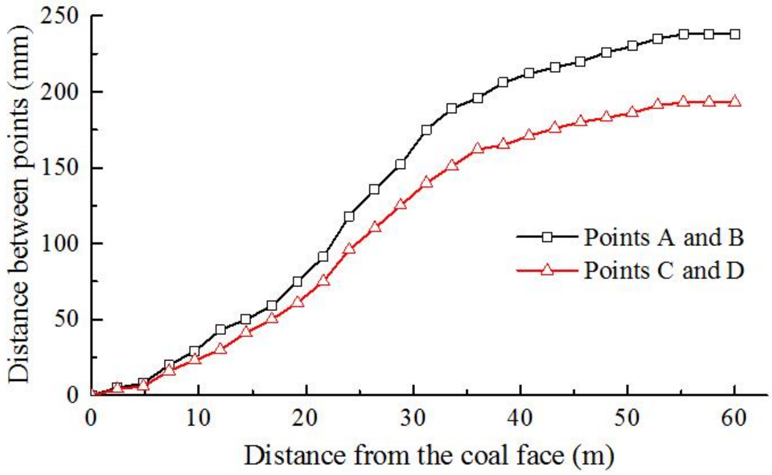


Figure 15 Average measurement results from each survey station (see online version for colours)



As the coal face advancing, the active mining face moves progressively farther from the base points. The distance between base points A and B and that between points C and D were measured every time the face advanced 2.4 m (Figure 15). As shown in Figure 15, as the distance between the survey stations and the coal face increased, the convergence between the roof and floor and that between the two sides increased at first and then remained unchanged. The convergence between the roof and floor was a little larger than that between the two sides. When the distance between the survey stations and the coal face was less than 40 m, the convergence speed between the roof and floor ranged from 5 to 8.5 mm/m and that between the two sides ranged from 3 to 7 mm/m. When the distance between the survey stations and the coal face was greater than 40 m, the roof-to-floor convergence and the side-to-side convergence were practically constant as the coal face advanced.

Overall, the maximum roof-to-floor convergence was 238 mm whereas the maximum side-to-side convergence was 193 mm. Thus, the deformation of the surrounding rocks of no. 16101 haulage roadway was effectively controlled by using the new support scheme. Moreover, no large cracks appeared in the coal or backfill, thus achieving the economic and mine safety benefits of retaining the roadway.

4 Conclusions

The purpose of this study was to ensure the stability of GER in coal mines by researching the reasonable supporting structure and parameters of GER. By comparing with the previous studies, this work contains at least two original aspects:

1 A deformation and control model for the 'coal-backfill-gangue' support system was established, which can determine the acceptable rotation angle for the lateral main roof and obtain the quantitative relationship between the roof pressure on the GER and the support forces provided by the coal, backfill, and gangue.

2 The quantitative design method for the support system of the GER under different conditions was proposed based on the acceptable rotation angle of the lateral main roof, which can ensure sufficient bearing capacities of the gangue.

The model analysis and case investigations showed that:

1 The coal, the gob-side backfill, and the gangue in the goaf should meet a certain deformation relationship, and their deformation capacities should be adjusted to make full use of their bearing capacities so as to ensure the stability of the GER.

2 A deformation and control model for the 'coal-backfill-gangue' support system is proposed that determines the acceptable deformation for the coal wall and backfill and the acceptable rotation angle for the lateral main roof. The model also quantifies the relationship between the roof pressure on the GER and the bearing forces of the coal, backfill, and gangue as well as specifying the artificial control methods for support of the GER under different conditions.

3 To determine the conditions in the no. 16101 haulage roadway in the Binhu Coal Mine, compression tests were conducted to obtain the bearing capacities of the bolted coal, gob-side backfill, and gangue. These tests showed that artificially increasing the bearing capacity of the coal wall and the backfill was required. To do that, 0.427 m of 'flexible and rigid' composite material was used in the backfill and the diameter of the coal bolts was changed to 20 mm and their length changed to 2.2 m. The bolt interval was changed to 0.6 m × 0.7 m. Field observations show that the improved supports can maintain the stability of the rocks surrounding the roadway well.

It should be noted that only two kinds of artificial control measures, adding bolts to the coal and using 'flexible and rigid' composite materials for the backfill, were applied during this research. In the future, more support design work and field tests for different geological conditions should be conducted to verify the proposed model and methods and to modify them to suit different ground conditions as necessary.

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List of symbols

h	Mining height
h_1	Thickness of the coal seam
h_2	Height of the backfill
h_3	Initial height of the roadway
h_4	Thickness of the flexible materials
m_z	Thickness of the immediate roof
L_0	Horizontal distance between the fracture line and the coal wall edge
L_1	Length of rock beam B in the main roof
L_2	Bearing length of the gangue
a	Roadway width
b	Backfill width
c	Hanging length of the immediate roof
K_m	Initial bulking coefficient for the gangue in the goaf
K_A	Compaction coefficient of the gangue in the goaf
ε_{CM}	Strain of the coal at peak stress
ε_{CF}	Strain of the backfill at peak stress
Δh_G	Deformation of gangue at the end of the gangue support area C
Δh_M	Deformation of coal wall
Δh_G	Deformation of backfill
Δh_F	Deformation at the centre of the backfill
Δh	Subsidence at the end of rock beam B
D	Inner diameter of the single hydraulic prop
θ	Rotation angle of the main roof

References

Bai, J.B., Shen, W.L., Guo, G.L., Wang, X.Y. and Yu, Y. (2015) 'Roof deformation, failure characteristics, and preventive techniques of gob-side entry driving heading adjacent to the advancing working face', *Rock Mech. Rock Eng.*, Vol. 48, No. 6, pp.2447–2458.

Chen, S.J., Yin, D.W., Liu, X.Y., Wang, H.L. and Pu, Z.Q. (2016) 'Collaborative mining using different equipment for a coal seam varying in thickness in a long wall working face', *Int. J. Oil Gas Coal Tech.*, Vol. 13, No. 1, pp.73–86.

Cheng, Y.H., Jiang, F.X., Lin, J.K., Chen, Q.H., Zhang, D. and Feng, F.S. (2012) 'Experimental study on gob-side entry retaining by roadside flexible packing under hard roof', *J. Min. Safety Eng.*, Vol. 29, pp.757–761.

Deng Y.H. and Wang S.Q. (2014) 'Feasibility analysis of gob-side entry retaining on a working face in a steep coal seam', *Int. J. Min. Sci. Tech.*, Vol. 24, No. 4, pp.499–503.

Fan, K.G., Liang, H.G., Ma, C.S. and Zang, C.W. (2014) 'Non-harmonious deformation controlling of gob-side entry in thin coal seam under dynamic pressure', *J. Rock Mech. Geotech. Eng.*, Vol. 6, No. 3, pp.269–274.

Gong, P., Ma, Z.G., Zhang, R.R., Ni, X.Y., Liu, F. and Huang, Z.M. (2017) 'Surrounding rock deformation mechanism and control technology for gob-side entry retaining with fully mechanized gangue backfilling mining: a case study', *Shock Vi.*, Vol. 2017, No. 4, pp.1–15.

He, H., Dou, L.M., Fan, J., Du, T.T. and Sun, X.L. (2012) 'Deep-hole directional fracturing of thick hard roof for rockburst prevention', *Tunn. Undergr. Sp. Tech.*, Vol. 32, No. 6, pp.34–43.

Huang, Z.M., Ma, Z.G., Zhang, L., Gong, P., Zhang, Y.K. and Liu F. (2016) 'A numerical study of macro-mesoscopic mechanical properties of gangue backfill under biaxial compression', *Int. J. Min. Sci. Technol.*, Vol. 26, No. 2, pp.309–317.

Liu, X.S., Ning, J.G., Tan, Y.L. and Gu, Q.H. (2016) 'Damage constitutive model based on energy dissipation for intact rock subjected to cyclic loading', *Int. J. Rock Mech. Min. Sci.*, Vol. 85, pp.27–32.

Lu, C.P., Liu, G.J., Liu, Y., Zhang, N., Xue, J.H. and Zhang, L. (2015) 'Microseismic multi-parameter characteristics of rockburst hazard induced by hard roof fall and high stress concentration', *Int. J. Rock Mech. Min. Sci.*, Vol. 76, pp.18–32.

Ma, Z.G., Gong, P., Fan, J.Q., Geng, M.M. and Zhang, G.W. (2011) 'Coupling mechanism of roof and supporting wall in gob-side entry retaining in fully-mechanized mining with gangue backfilling', *Min. Sci. Technol.*, China, Vol. 21, No. 6, pp.829–833.

Ning, J.G., Liu, X.S., Shi, X.S. and Wang, J. (2015) 'Study on cement – coal refuse backfill structure model of mining goaf in underground mine', *Coal Sci. Tech.*, Vol. 43, No. 12, pp.23–27.

Ning, J.G., Ma, P.F., Liu, X.S., Zhao, J. and Liu, W. (2013) 'Supporting mechanism of 'yielding-supporting' beside roadway maintained along the goaf under hard rocks', *J. Min. Safety Eng.*, Vol. 33, No. 3, pp.369–374.

Ning, J.G., Wang, J., Liu, X.S., Qian, K. and Sun, B. (2014) 'Soft–strong supporting mechanism of gob-side entry retaining in deep coal seams threatened by rockburst', *Int. J. Min. Sci. Tech.*, Vol. 24, No. 6, pp.805–810.

Schumacher, F.P. and Kim, E. (2014) 'Evaluation of directional drilling implication of double layered pipe umbrella system for the coalmine roof support with composite material and beam element methods using FLAC3D', *J. Min. Sci.*, Vol. 50, No. 2, pp.335–338.

Su, H., Bai, J.B., Yan, S., Chen, Y. and Zhang, Z.Z. (2015) 'Study on gob-side entry retaining in fully-mechanized longwall with top-coal caving and its application', *Int. J. Min. Sci. Tech.*, Vol. 25, No. 3, pp.503–510.

Tan, Y.L., Liu, X.S., Ning, J.G. and Lu, Y.W. (2017) 'In situ investigations on failure development of overlying strata induced by mining multiple coalseams', *Geotech. Test. J.*, Vol. 40, No. 2, pp.244–257.

Tan, Y.L., Liu, X.S., Ning, J.G. and Tian, C.L. (2015a) 'Front abutment pressure concentration forecast by monitoring cable-forces in the roof', *Int. J. Rock Mech. Min. Sci.*, Vol. 77, pp.202–207.

Tan, Y.L., Yu, F.H., Ning, J.G. and Zhao, T.B. (2015b) 'Design and construction of entry retaining wall along a gob side under hard roof stratum', *Int. J. Rock Mech. Min. Sci.*, Vol. 77, pp.115–121.

Wang, F., Xu, J.L., Xie, J.L., Li, Z. and Gao, J.K. (2016) 'Mechanisms influencing the lateral roof roadway deformation by mining-induced fault population activation: a case study', *Int. J. Oil Gas Coal Tech.*, Vol. 11, No. 4, pp.411–428.

Wang, M., Bai, J.B., Li, W.F., Wang, X.Y. and Cao, S.G. (2015) 'Failure mechanism and control of deep gob-side entry', *Arab. J. Geosci.*, Vol. 8, No. 11, pp.9117–9131.

Xue, D.P., Wang, J.P., Tu, H.P., Wang, F.T. and Zhao, J. (2013) 'Deformation failure mechanism and application of the backfill along the goaf-side retained roadway', *Int. J. Min. Sci. Tech.*, Vol. 23, No. 3, pp.329–335.

Zhang, D.S., Ma, L.Q., Feng, G.M. and Miao, X.X. (2005) 'The technique of gob-side entry retaining with entry-in packing on its original position in fully-mechanized coalface with top-coal caving', *Chin. J. Rock Mech. Eng.*, Vol. 24, No. 7, pp.1164–1168.

Zhang, N., Han, C.L., Kan, J.G. and Deng, X.G. (2014a) 'Theory and practice of surrounding rock control for pillarless gob-side entry retaining', *J. China Coal Soc.*, Vol. 39, No. 8, pp.1635–1641.

Zhang, Y.Q., Tang, J.X., Xiao, D.Q., Sun, L.L. and Zhang, W.Z. (2014b) 'Spontaneous caving and gob-side entry retaining of thin seam with large inclined angle', *Int. J. Min. Sci. Tech.*, Vol. 24, No. 4, pp.441–445.

Zhang, N., Yuan, L., Han, C.L., Xue, J.H. and Kan, J.G. (2012) 'Stability and deformation of surrounding rock in pillarless gob-side entry retaining', *Safety Sci.*, Vol. 50, No. 4, pp.593–599.

Zhang, S.H. and Zhao, Y.L. (2001) 'Simulation on surrounding rock activities of gob-side entry retaining in fully-mechanized top-coal caving faces', *Ground Pressure Strata Control*, Vol. 18, No. 3, pp.51–55.

Zhao, T.B., Tan, Y.L., Liu, S.S. and Xiao, Y.X. (2012) 'Analysis of rheological properties and control mechanism of anchored rock', *Rock Soil Mech.*, Vol. 33, No. 6, pp.1730–1734.

Zhou, B.J., Xu, J.H., Zhao, M.S. and Zeng, Q.L. (2012) 'Stability study on naturally filling body in gob-side entry retaining', *Int. J. Min. Sci. Tech.*, Vol. 22, No. 3, pp.423–427.