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## AN ASYMMETRICAL BOLT SUPPORTING DESIGN OF RETREATING COAL ROADWAY BASED ON PHYSICAL SIMULATION IN LARGE DIP COAL SEAM\*\*\*

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### 1. Introduction

Inclined coal seams result in asymmetrical entries, and make it more difficult to design suitable parameters and control rock stability when bolt support system is implemented, which limits safely and high-efficiently mining LDCSs whose obliquity changes from 25° to 45°. Simulation material physical experiment has become an important method to investigate into movement, deformation and breakage of rocks and redistribution patterns of mining-induced stress. Also, the method has been used to study mechanical characteristics of rocks surrounding roadways with bolt supporting system, and it plays an active role in mining engineering [1–11]. But researches on retreating entries during longwall mining in LDCSs are few due to experimental equipment limitation. In this paper, according to geological and technological conditions in Huainan mine area, a self-developed rotatable experimental frame for similar material simulation was used to build physical model with an obliquity of 30° and to analyze the mechanical characteristics in roadway of rocks surrounding with bolt supporting system. Consequently, an asymmetrical bolt supporting was designed and implemented for retreating entry with longwall mining.

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## 2. Geological conditions

The A3 seam in the roadway No. 63103 is a large-dip coal seam in one coal mine of Huai Nan Coal Field. Its obliquity is 28 to 31°, averaging 30°; and its thickness is 2.4 to 4.5 m, averaging 3.2 m. The elevation of the roadway No. 63103 is about – 678 m. The main roof is coarse-grained sandstone, whose thickness is 3.5 to 6.6 m, averaging 5.0 m. The immediate roof is composed of siltstone, whose thickness is 1.9 to 5.3 m, averaging 3.1 m. The immediate floor is muddy sandstone with a thickness of 3.2–6.7 m, averaging 4.3 m. The width, lower-rib and upper-rib heights of the roadway are 4.8, 1.6 and 4.15 m, respectively.

## 3. Physical modeling

### 3.1. Material proportion

According to geological conditions, engineering practices and experimental equipments, the geometrical ratio, density ratio and stress ratio are determined as 0.02, 0.6 and 0.012, respectively. In the physical model, powder sand was used for aggregate, lime and gypsum were used for cementing materials, and mica powder was used for separated material (Tab. 1).

### 3.2. Model design

In this paper, a self-developed rotatable experimental frame was used to build the physical model with the obliquity of 30°, whose sizes are 2 m × 0.2 m × 1.5 m (length × width × height) (Fig. 1).

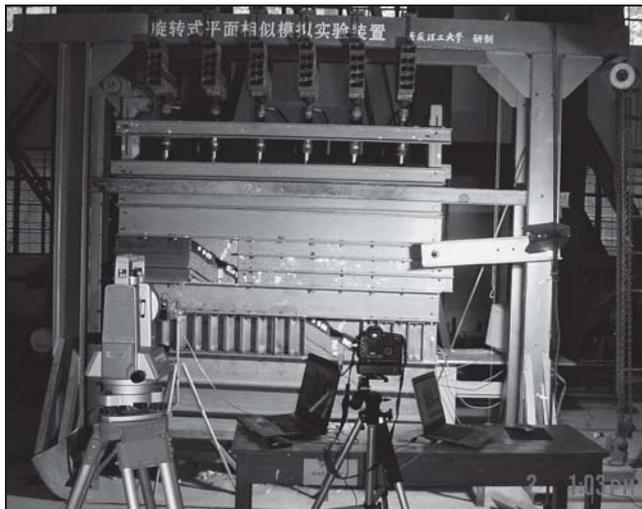


Fig. 1. Physical model and rotatable testing device

TABLE 1  
Material proportion in physical model

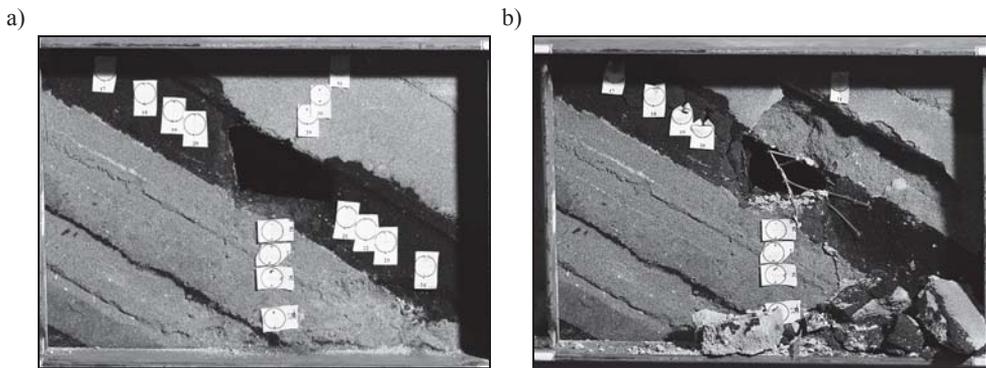
Stratum	Thickness, cm	Prototype		Model		Sand : lime : gypsum
		$\rho$ , kg/m <sup>3</sup>	$\sigma_c$ , MPa	$\rho$ , kg/m <sup>3</sup>	$\sigma_c$ , MPa	
Additional stratum		2 500	69.6	1.5	0.835	6 : 7 : 3
Powder sandstone	9	2 873	135.33	1.724	1.624	4 : 5 : 5
Middle sandstone	14.2	2 890	135.83	1.734	1.630	4 : 5 : 5
Sandy mudstone	4.4	2 530	39.17	1.518	0.470	8 : 6 : 4
Coarse sandstone	14	2 850	147.36	1.710	1.768	6 : 3 : 7
Powder sandstone	8.8	2 873	135.33	1.724	1.624	4 : 5 : 5
Middle sandstone	10	2 890	135.83	1.734	1.630	4 : 5 : 5
Powder sandstone	6.2	2 460	30.17	1.476	0.362	8 : 6 : 4
A3 coal	6.4	1 380	4.50	0.828	0.054	4 : 4 : 6
Muddy sandstone	8.6	2 460	30.17	1.476	0.362	8 : 6 : 4
A2 coal	0.86	1 380	4.50	0.828	0.054	4 : 4 : 6
Muddy sandstone	8	2 460	30.17	1.476	0.362	8 : 6 : 4
A1 coal	3.8	1 380	4.50	0.828	0.054	4 : 4 : 6
Muddy sandstone	6.4	2 460	30.17	1.764	0.362	8 : 6 : 4
Muddy sandstone	7.4	2 492	33.6	1.494	0.403	8 : 6 : 4
Coarse sandstone	10.2	2 850	147.36	1.71	1.768	6 : 3 : 7

In the model, 15A fuse, 20A fuse, plastic yarn nets of 1 mm — thick celluloid, and emulsion paint were used to simulate bolt, cable, metal net and anchoring agent, respectively. The BX120-50AA resistance strain gage, transit and the YJD-27 data collection and transformation system were adopted to conduct rock stress measurement, rock movement observation and data collection, respectively. Six jacks were used to load mining-induced stress in 3 hours with the relative stress concentration coefficient ( $K$ ) ranging from 1.12 to 2.5 for simulating the effect of different loading stages of mining-induced stress on rocks in the entry during the working face advancing. Two roadways with the elevation of  $-667$  and  $-689$  m, respectively, were extracted.

## 4. Mechanical characteristics of physical model

### 4.1. Deformation characteristics of rocks surrounding roadway

Influenced by asymmetrical structure of rock surrounding extracting roadway and mining-induced stress, at the end of loading, rib caving occurs in upper-rib, which expands roof space that lead roof dislocation along dip to break and cave. And lower-rib is to failure squeezed and compressed by roof dislocating. All above result stability of surrounding rock rapidly reduce and roadway structure composed of surrounding rock and bolt support system are instability (Fig. 2).

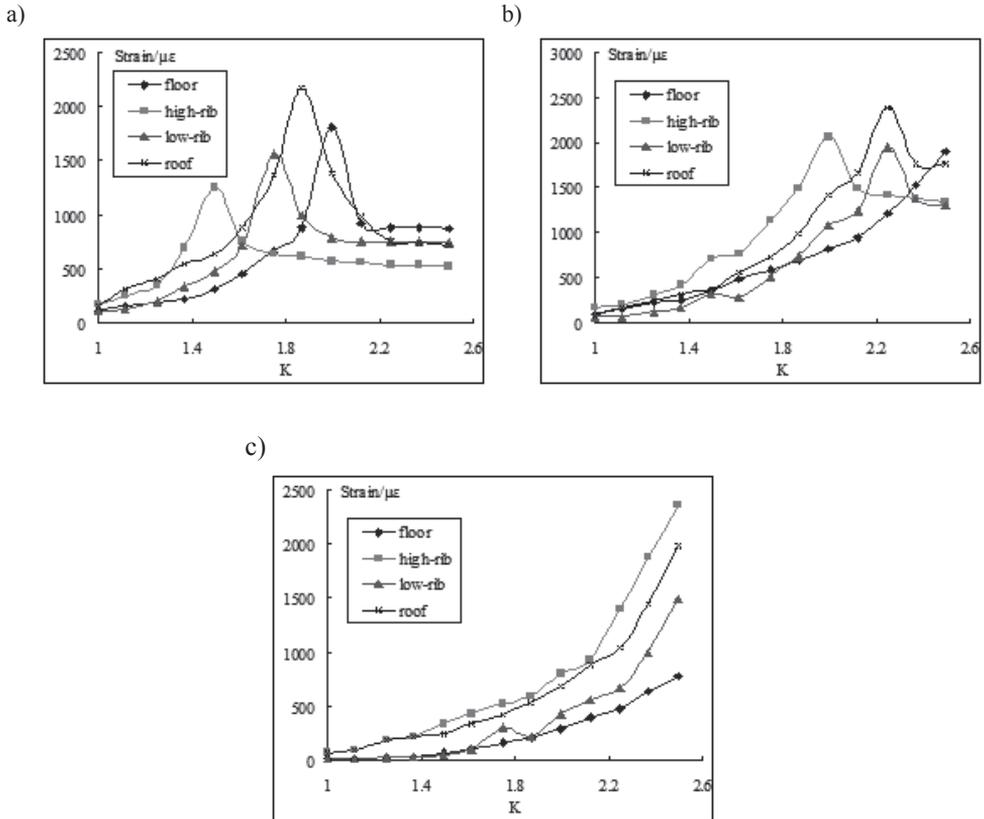


**Fig. 2.** Deformation and failure features of surrounding rock:  
a) after extracting and bolting support; b) end of loading

### 4.2. Stress characteristics of rocks surrounding roadway

To some extent, the strain value of resistance strain gage can be used to stand for mining-induced stress. Based on analyzing strain development curves (Fig. 3, legend stand for distance to roadway surface) of rock located different distances to surface during loading process, mining-

induced stress development laws of roof, upper-rib, lower-rib, and floor are unsymmetrical. Rock stress development is out-of-step and the upper-rib is quickest, roof and low-rib are followed, floor is slow to get peak value. In 1.5 m to surface, the  $K$  of getting peak value rock is 1.5, 1.62, 1.75, and 2 in upper-rib, low-rib, roof, and floor respectively (Fig. 3a). In 2.5–3 m to surface, the  $K$  of getting peak value rock is 2, 2.25, and 2.25 in upper-rib, lower-rib, and roof (Fig. 3b). And in 5–7 m to surface, mining-induce stress are still rise (Fig. 3c).



**Fig. 3.** Stress development curves of different distance to rock surface of entry:  
a) 1.5 m; b) 2.5–3 m; c) 5–7 m

### 4.3. Displacement characteristics of rocks surrounding roadway

From rock displacement curves and characteristics, displacement development laws of rock surrounding extracting roadway are obtained as follows (Fig.4, engineering unit):

- In roof, when  $K$  is to 1.87, quick deformation and subsidence occurs and roof begins to be broken till to cave at side of upper rib while  $K$  is more than 2. And in floor, when  $K$

exceeds 2, quick deformation and floor heaving occurs in floor and rib begins to be collapsed by roof dislocation acting. Finally, the total displacement of roof and floor is 368 mm.

- In upper-rib, when  $K$  is to 1.5, quick deformation and rib heaving occurs and ribs begin to be sheared failure by roof dislocation acting at top corner of upper-rib. And in lower-rib, when  $K$  is to 1.75, quick deformation and rib heaving occurs and rib begins to be collapsed by roof dislocation acting. Finally, the total displacement of upper & low ribs is 518 mm.

Consequently, upper-rib in LDCS solid coal is most easily to be sensitive by mining-induced stress action and followed is roof and lower-rib. But roof dislocation is the key factor for accelerating upper and low rib failure because of tensile stress and compressive stress.

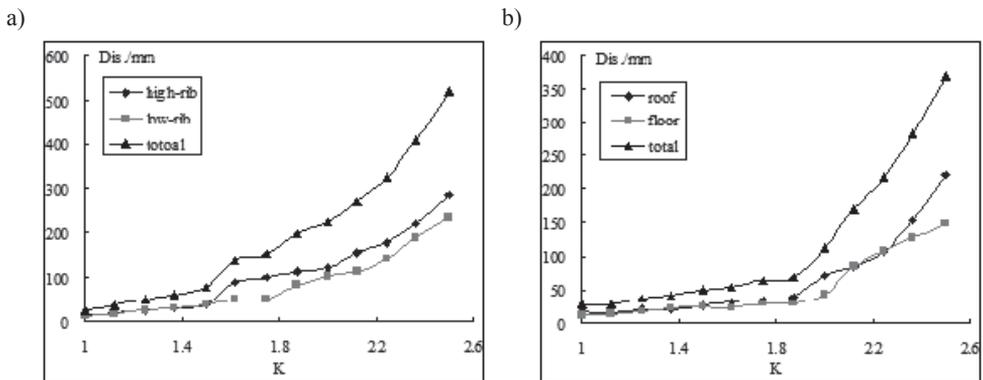


Fig. 4. Displacement curves of rock surrounding roadway: a) upper-rib and lower-rib; b) roof and floor

#### 4.4. Theory analyses

To summarize, influenced by geometrically asymmetric configuration of the roadway, the mechanical characteristics of different sections of the roadway are anisotropic and dissymmetrical in the LDCSs. Actually, On one hand, the large dip delays the development of deformation and peak strain in the roof and lower-rib, but the bearing capacity of rocks is increased. On the other hand, the large dip accelerates the development of deformation and peak strain in the upper-rib, but the bearing capacity of rocks is decreased.

While extracting entries in solid coal, the roof and upper-rib are usually weak stability sections. Tensile fracture at the top corner and dislocation often occur in the roof, which lead to the collapsing of the lower-rib. Shear failure, rib squeezing and collapsing often occur in the upper-rib. Consequently, the roof and the upper-rib should be controlled as key sections in implementing bolt supporting system. Engineering practices show that implementing bolt

supporting system with high shear strength to resist high tensile stress and limit movement along the dip is effective in controlling roof stability, and setting cable along the strike to enhance rib strength is feasible. However, stability control of the rocks in the floor without supporting is the most difficult.

## **5. Engineering application**

### **5.1. Asymmetrical bolt supporting design**

Based on geological conditions and theory analyses of bolt supporting mechanism, the main asymmetrical parameters of coal roadway of solid (no mining) and gob-side with small width pillar prevention were designed shown as Figure 5. In Figure 5, No. 1, No. 2, No. 3, No. 4, No. 5, and No. 6 stands for roadway section, prestressing force bolt, steel mesh, steel band, prestressing force cable, and prop respectively. And the inclination angle is  $28^\circ$ , width, lower-rib height, and upper-rib height of roadway is 4800 mm, 1600 mm, and 4150 mm respectively, and the pillar width is 5000 mm to 10000 mm.

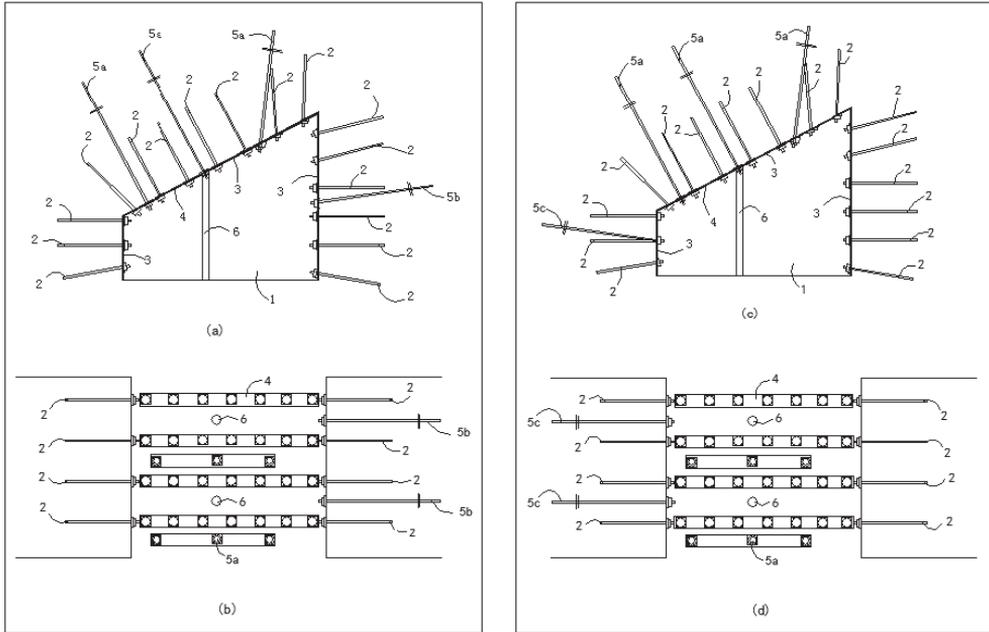
### **5.2. Supporting steps**

Firstly, After extracting roadway (No. 1), prestressing force bolts, steel meshes, and steel bands are immediately set up to control rocks surrounding roadway. In which, the main parameters of prestressing force bolt (No. 2) with screw thread material include diameter is 25 mm, length is 2500 mm, interval and row spacing are 700–800 mm, breakage force is not less than 400 KN and prestressing force is no less than 300 KN. The diameter, length, width, and intervals of steel mesh (No. 3) is 4.5 mm, 1700 mm, 900 mm, and 100 mm  $\times$  100 mm. And steel band (No. 4) is GRT-M4.

Secondly, prestressing force cables are installed in middle and sides of roof end of basic supporting implemented. In which, the main parameters of prestressing force cables (No. 5a) with 7 steel strands include diameter is 21.6 mm, length is 6300 mm, interval and row spacing are 1200–1600 mm, extensibility is not less than 4%, breakage force is not less than 470 KN and prestressing force is no less than 250 KN.

Thirdly, after 5–7 days of extracting, to solid coal roadway prestressing force cables are set up in upper-rib and in lower-rib to coal roadway with small pillar prevention. In which, the main parameters of prestressing force cables (No. 5b, 5c) used in ribs are 7 steel strands and diameter is 21.6 mm, length is 5300 mm, row spacing is 1600 mm, extensibility is not less than 4%, breakage force is not less than 470 KN and prestressing force is no less than 200 KN.

Finally, in order to act as an assistant support and signal function, props of wood or steel are usually built at side to lower-rib.



**Fig. 5.** Bolt supporting parameters in different roadway types: a) section of solid roadway; b) planform of solid roadway; c) section of gob-side with small width pillar prevention; d) planform of gob-side with small width pillar prevention

### 5.3. Engineering practice analyses

The asymmetrical bolt supporting has been used to the tailentry with small pillar prevention and the headentry with solid coal in No. 63103 longwall mining face of Xinzhuanzhi coal mine. The accumulative displacement of roof-floor and upper-lower rib is 610 mm and 515 mm respectively in tailentry and 268 mm and 312 mm in headentry. So, Engineering show the assymetrical design method is suitable and reliable based on integrated analyzing.

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