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TECHNICAL NOTE

Roadway Stagger Layout for Effective Control of Gob-Side Rock Bursts in the Longwall Mining of a Thick Coal Seam

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1 Introduction

Currently, rock bursts pose a serious threat to the safety of miners and equipment in underground coal mining operations, especially in China. The number of coal mines with rock burst hazards is increasing year by year with no signs of letting up. By 31 December 2013, there were 142 coal mines in China which had experienced rock bursts. Each year, rock bursts cause considerable economic loss and enormous casualties. For instance, a rock burst induced by a large thrust fault caused 10 deaths and trapped 75 people on 3 November 2011 during the headgate excavation of LW21221 in Qianqiu coal mine, Yima City, China (Cai et al. 2014a, b; Li et al. 2014). Rock bursts are serious not only because the hazard itself can cause damage, but because it can cause a series of secondary disasters, such as coal and gas outbursts, and gas explosions. The most serious gas explosion recorded to date killed 214 people, injured 30 people, and caused a direct economic loss of ¥49.689 million. It happened on 14 February 2005 in Sunjiawan coal mine, Fuxin City, China. Investigation revealed that the gas explosion was induced by a rock burst (State Administration of Work Safety, State Administration of Coal Mine Safety 2005).

Existing research mainly concentrates on the monitoring, prediction, and prevention of rock bursts (Adoko et al. 2013; Cai et al. 2014a, 2014b; Kornowski and Kurzeja 2012; Mu et al. 2013). Control measures against rock bursts are usually passive and include: de-stress blasting (Konicek et al. 2013), directional fracturing (He et al. 2012a), large-diameter drilling (Li et al. 2014), etc. These measures are time-consuming and only reduce rock burst potential without complete elimination of the hazard. The latest statistical data (Pan et al. 2013) show that 87 % of rock bursts occurred in roadways in China's coal mines. Compared with the 72.6 % seen from previous statistics (Dou and He 2001), this proportion has increased. Among roadway rock bursts, gob-side rock bursts (GRBs) (i.e., rock bursts occurring in gob-side roadways) account for the majority. For example, the eight rock bursts in the No. 17 coal seam of Xing'an coal mine, Hegang City, China, caused damage to the tailgate and the longwall face eight and two times, respectively. However, the headgate was not damaged at all (see Fig. 1a). The 22 rock bursts in the No. 17 coal seam of Junde coal mine, Hegang City, caused damage to the tailgate, the longwall face, and the headgate 18 times, five times, and once, respectively (see Fig. 1b). Both tailgates in the two coal mines are gob-side roadways. It is common in other coal mines that GRBs account for the majority of rock bursts because gob-side roadways bear a higher stress. If GRBs are controlled effectively, rock burst hazards will be significantly mitigated.

In this work, a case study of Yuejin coal mine (YCM) in Yima City, China, was analyzed to ascertain whether, or not, roadway staggered layouts could control GRBs. The aim of this study was to deduce whether, or not, this

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Fig. 1 Rock burst occurrences in **a** the No. 17 coal seam of Xing'an coal mine and **b** the No. 17 coal seam of Junde coal mine. For instance, the 22nd rock burst in Junde coal mine caused damage to the tailgate, the longwall face, and the headgate simultaneously

method was favorable to GRB control in the longwall mining of a thick coal seam. If so, insights into its underlying behavior mechanism were also to be deduced.

2 A Case Study

2.1 Basic Conditions of the Panel

The selected panel, LW25110 in YCM, is located in the west of Henan Province, China. YCM suffers high rock burst risk and is a characteristic coal mine with rock burst hazards, making it an appropriate site for case studies examining the mechanism, monitoring, prediction, and prevention of rock bursts.

The No. 2-1 coal seam in the selected panel has a cover depth varying from 950 to 1000 m. The thickness of the coal seam ranges from 8.4 to 13.2 m (11.5 m on average) with an average inclination of 12°. The coal seam is overlain successively by mudstone (18 m thick), the No. 1-2 coal seam (1.5 m), mudstone (4 m), and glutenite (190 m), and underlain successively by mudstone (4 m) and sandstone (26 m). The length and width of LW25110 are 865 and 191 m, respectively. In the No. 25 mining district, the fully mechanized top-coal caving method was first adopted to extract the entire thickness of the coal seam in this panel. Mining in LW25110 began on 17 July 2010 and was completed on 7 January 2013.

LW25110 adopted a staggered roadway layout method (see Appendix for further details) as a trial to control rock

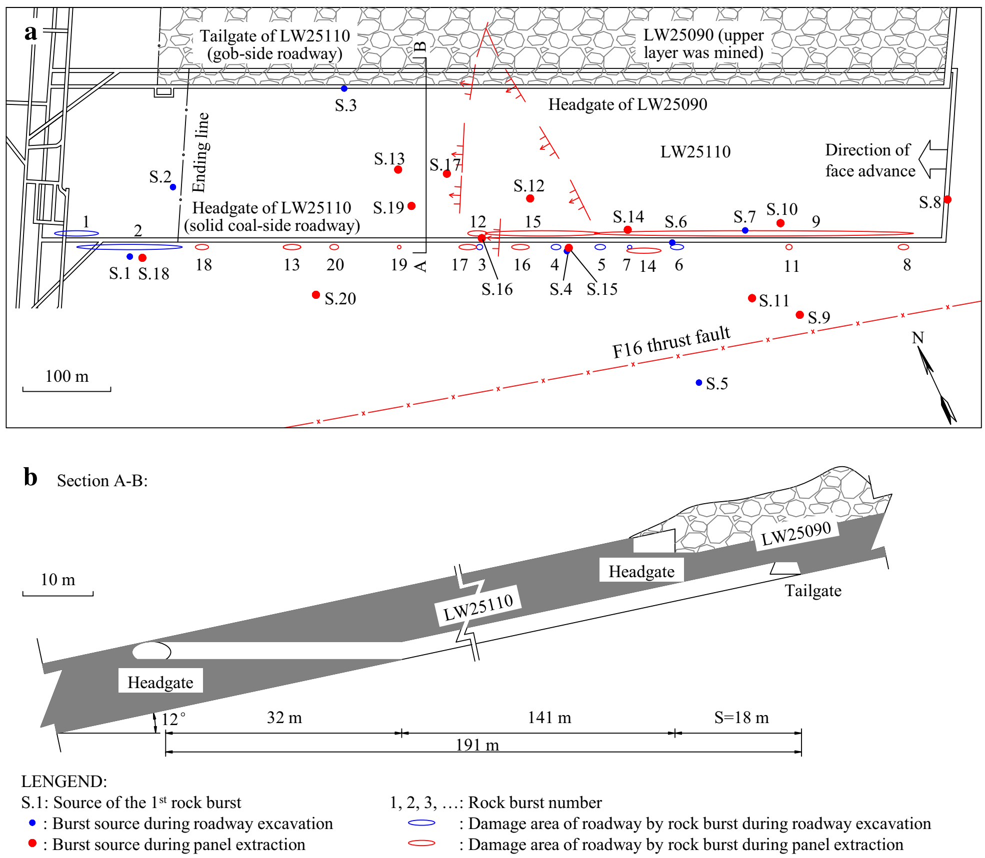

Fig. 2 Roadway layout and rock burst locations at LW25110

bursts. The tailgate (and headgate) of LW25110 lies along the bottom (and top) of the coal. LW25090, whose upper layer was extracted by slice mining, is situated adjacent to LW25110 in the north. The tailgate of LW25110 lies just below LW25090, but offset by the designed stagger. The interlaced distance S is 18 m. The F16 thrust fault lies to the south, and the distance between LW25110 and the fault ranges from 75 to 230 m. Three small faults are situated in the middle of LW25110. The eastern and western parts of the area adjacent to LW25110 contain solid coal (see Fig. 2).

2.2 Monitoring Measures

The ARAMIS M/E system (a microseismic monitoring system) developed by the Institute of Innovative Technologies (EMAG) of Poland was installed in YCM to locate tremors and rock bursts, determine energy released, and assess rock burst risk [refer to Li et al. (2014) for further details]. Figure 3a shows typical locations of and energy released by tremors as monitored by the system. It may be seen that a large number of tremors occurred near the tailgate.

The KJ216 mine pressure monitoring system was installed in LW25110 to investigate the support pressure. The system contains a total of 13 digital pressure gauges that were installed on hydraulic supports (from the third to the 123rd on every tenth support) to the longwall face. By correlating the pressure data with the extent to which the face had been advanced, it was found that the image map of the support pressures was as shown in Fig. 3b. It was seen that the pressure on those supports close to the tailgate (20–45 MPa) was higher than that on the supports close to the headgate (20–35 MPa, with 20–30 MPa accounting for the majority), which indicated that the coal body close to the tailgate bore a higher static stress.

2.3 Controlling Measures

Rock burst control measures were taken more in the headgate than in the tailgate. Large-diameter drilling and de-stress blasting measures taken in LW25110 from 17 July 2010 to 31 December 2011 were investigated, as shown in Fig. 4 and Tables 1 and 2. About 235 boreholes for de-stress blasting were drilled along the headgate in the coal body and about 3210 boreholes for large-diameter

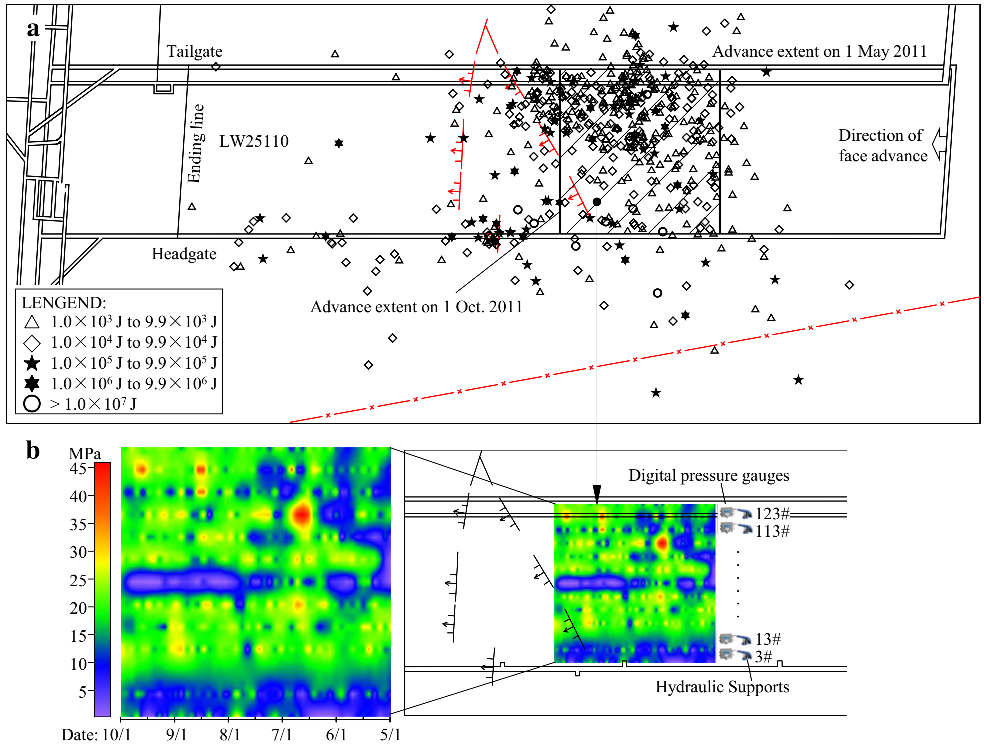

Fig. 3 Monitoring data for LW25110 from 1 May 2011 to 1 October 2011: **a** typical locations and energy of tremors; **b** support pressure at the coal face. The *shaded area* in **a** was mined during this period of time

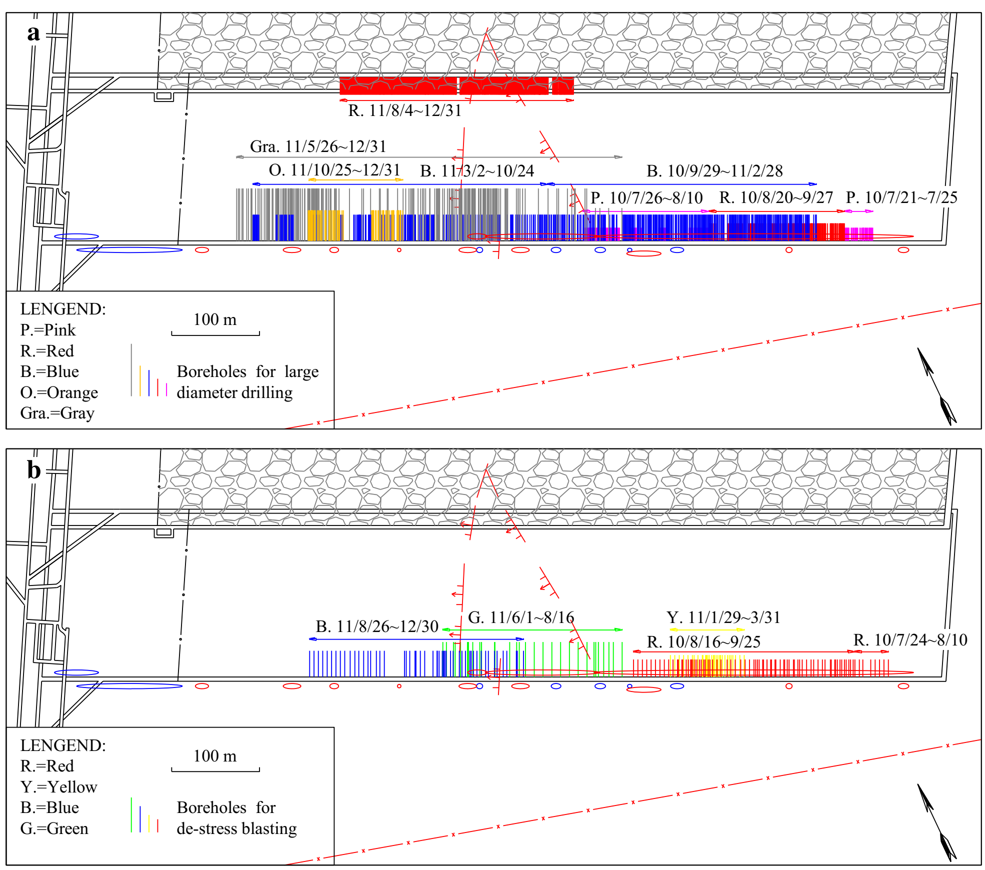

Fig. 4 Borehole layout for **a** large-diameter drilling and **b** de-stress blasting in LW25110 from 17 July 2010 to 31 December 2011

Table 1 Borehole parameters for large-diameter drilling

Boreholes for large-diameter drilling	Borehole length (m)	Borehole diameter (mm)	Date
Pink ones	15	75	21 July to 10 Aug. 2010
Red ones	20	75	20 Aug. to 27 Sep. 2010
			4 Aug. to 31 Dec. 2011 (tailgate)
Blue ones	30	100	29 Sep. 2010 to 24 Oct. 2011
Orange ones	35	113	25 Oct. to 31 Dec. 2011
Gray ones	60	113	26 May to 31 Dec. 2011

Table 2 Borehole parameters for de-stress blasting

Boreholes for de-stress blasting	Borehole length (m)	Explosives' weight (kg)	Date
Red ones	20	10.8	24 July to 25 Sep. 2010
Yellow ones	25	18	29 Jan. to 31 Mar. 2011
Green ones	40	36	1 June to 16 Aug. 2011
Blue ones	30	36	26 Aug. to 30 Dec. 2011

drilling were opened, of which 2445/765 boreholes were situated along the headgate/tailgate, accounting for 76.2/23.8 % respectively. Boreholes for large-diameter drilling and de-stress blasting along the headgate were approximately uniformly distributed and covered the entire roadway. However, boreholes for large-diameter drilling along the tailgate only covered a small proportion of the roadway. Tables 1 and 2 show that the parameters for large-diameter drilling and de-stress blasting along the headgate were improved five- and fourfold, respectively. Each time, the measures were enhanced.

2.4 Roadway Support

The headgate support is much stronger than that of the tailgate, as shown in Fig. 5. There are a total of 24 bolts and seven cables in the cross-section of the headgate, while there are only 15 bolts in the cross-section of the tailgate. The tailgate is trapezoidal and the support consists of bolts, wire mesh, No. 12 I-type steel members, and single hydraulic props. All the bolts are 1800 mm long and are drilled into the coal at 750 mm intervals. The diameter of bolts in the roof and walls is 22 and 18 mm respectively. Two single hydraulic props are situated in the middle of the roadway. The headgate is elliptical and relies on a rigid-flexible energy-absorbing support system. It is a three-level support system composed of three parts, namely, the bolt–mesh-cable support, the O-type support made of energy-absorbing materials and No. 36 U-type steel members, and the hydraulic burst-resisting bracket. The wire mesh and bolts are identical to those of the tailgate except that the length of bolts in the roof is 2250 mm. The cables are 8000 mm long and 17.8 mm in diameter and are drilled into the roof (and up into the walls) at intervals of 1400 mm (and 1500 mm). Energy-absorbing materials consist of wood and sand. The bracket (type: ZD/6400/27/42G) is developed by Shengyang Tian An Mining Machinery Stock Co., Ltd. to have a working resistance of 36.62 MPa.

2.5 Rock Burst Occurrence

Since the first rock burst on 21 July 2009 during roadway excavation, rock bursts, and their consequences, had been recorded in detail except for the tenth incident. Records mainly consist of what was damaged, the extent of that damage, and damage locations (e.g., floor heave for 1.0 m, overturned the stage loader, ruptured bolts, etc.). Sources and energy of rock bursts were recorded by the ARAMIS M/E system. A simplified record of rock bursts is shown in Fig. 2 and Table 3. It is seen that the headgate was damaged when each rock burst occurred and there were a total of 20 rock bursts causing damage to the headgate (note the damage location of the tenth rock burst was still within the headgate). The gob-side roadway (i.e., tailgate) was not damaged, which was abnormal. Damaged zones during panel extraction were much larger than those caused by roadway excavation. The most severe rock burst was the ninth one which caused a 363-m-long damaged section to appear in the headgate. Sources of rock bursts are mainly distributed in areas around the headgate, the F16 thrust fault, and other smaller faults. The energy released by rock bursts during panel extraction (10^6–10^8 J) was much greater than that during roadway excavation (10^4–10^5 J). Figure 6 shows record photographs of roadway support and

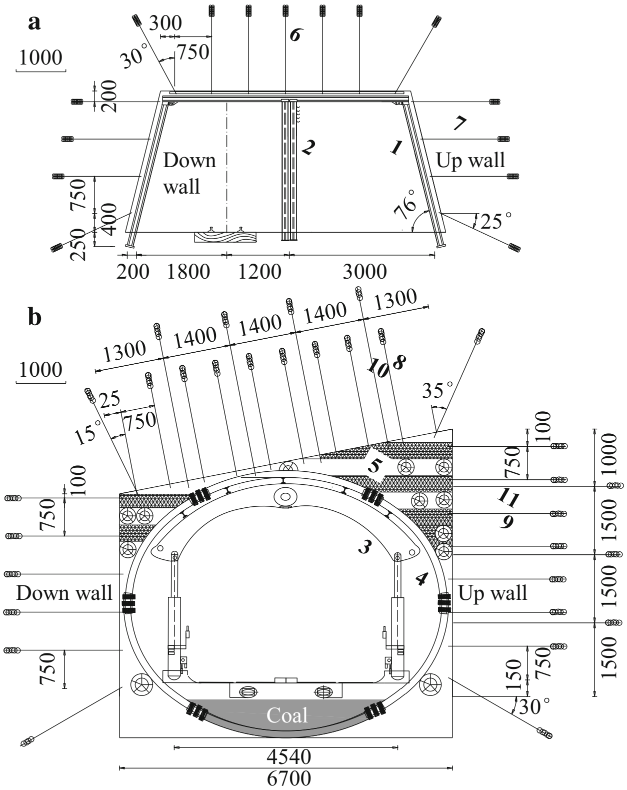

Fig. 5 Support parameters for **a** the tailgate and **b** the headgate of LW25110

rock burst hazards. It is seen that the headgate (solid coal-side roadway) of LW25110 and the gob-side roadway in Xing'an coal mine (without using the roadway stagger layout method) were severely damaged. However, the gob-side roadway (i.e., tailgate) of LW25110 remained intact when each rock burst occurred.

3 Discussion

It is well known that rock bursts in coal mines are the result of static stress (abutment stress) coupled with dynamic stress (seismic waves) in coal seams around mining areas (Dou et al. 2012, 2014; He 2013; He et al. 2014). Tremors caused by mining activities propagate in the form of seismic waves and produce dynamic stress within coal-rock masses. A rock burst may occur when the total stress (due to superposition of static stress within the coal and dynamic stress induced by tremors) within a coal-rock mass reaches a certain critical level (He et al. 2012b; Li et al. 2015).

Table 3 Statistics on previous rock bursts

No.	Date	Energy (J)	Damage range (m)	Note
1	2009/7/21	6.97×10^5	50	R.Exc.
2	2009/8/31	4.51×10^5	120	R.Exc.
3	2009/12/25	3.84×10^4	7	R.Exc.
4	2010/1/8	4.06×10^4	11	R.Exc.
5	2010/1/19	1.10×10^4	12	R.Exc.
6	2010/2/10	1.21×10^5	15	R.Exc.
7	2010/3/16	1.39×10^5	5	R.Exc.
8	2010/7/23	2.31×10^7	10	P.Ext.
9	2010/8/11	9.00×10^7	363	P.Ext.
10	2011/1/29	3.39×10^6	–	P.Ext.
11	2011/2/7	2.23×10^6	7	P.Ext.
12	2011/3/1	1.45×10^8	20	P.Ext.
13	2011/4/10	1.57×10^8	20	P.Ext.
14	2011/5/26	1.09×10^7	39	P.Ext.
15	2011/8/26	1.47×10^7	130	P.Ext.
16	2011/8/29	1.77×10^7	20	P.Ext.
17	2011/12/3	9.32×10^6	20	P.Ext.
18	2012/1/4	1.94×10^7	15	P.Ext.
19	2012/1/31	1.27×10^6	4	P.Ext.
20	2012/4/22	1.06×10^7	10	P.Ext.

R.Exc. roadway excavation, *P.Ext.* panel extraction

Figure 3a indicates that the majority of tremors occurred close to the tailgate, which implies that tremor-induced dynamic stress was supposed to be imposed on the coal-rock mass surrounding the tailgate more than that of the headgate. Figure 3b shows that the coal body close to the tailgate bears a higher static stress. In the case of normally arranged roadways (i.e., without using the roadway stagger layout method), the total stress within the coal-rock mass surrounding the tailgate was more likely to reach the critical stress level, which is the reason why GRBs account for the majority of rock bursts occurring in roadways. In addition, support of the tailgate is poorer and the tailgate only had a smaller number of controlling measures installed, as investigated in Sect. 2. All these factors mentioned above fooled us into thinking that the tailgate was under a higher rock burst risk (i.e., rock bursts were supposed to be more likely to occur in the tailgate rather than in the headgate). However, all the rock bursts occurred in the headgate, and the tailgate remained intact each time: this was abnormal. The main difference was the roadway stagger layout when comparing LW25110 with normally arranged panels, such as the roadway layout of Xing'an and Junde coal mines in Fig. 1. Therefore, it was concluded that it was the layout itself that generated this anomaly (i.e., the roadway stagger layout was favorable for GRB control).

The mechanism underlying roadway stagger layout for effective control of GRBs is stated thus: (1) this layout

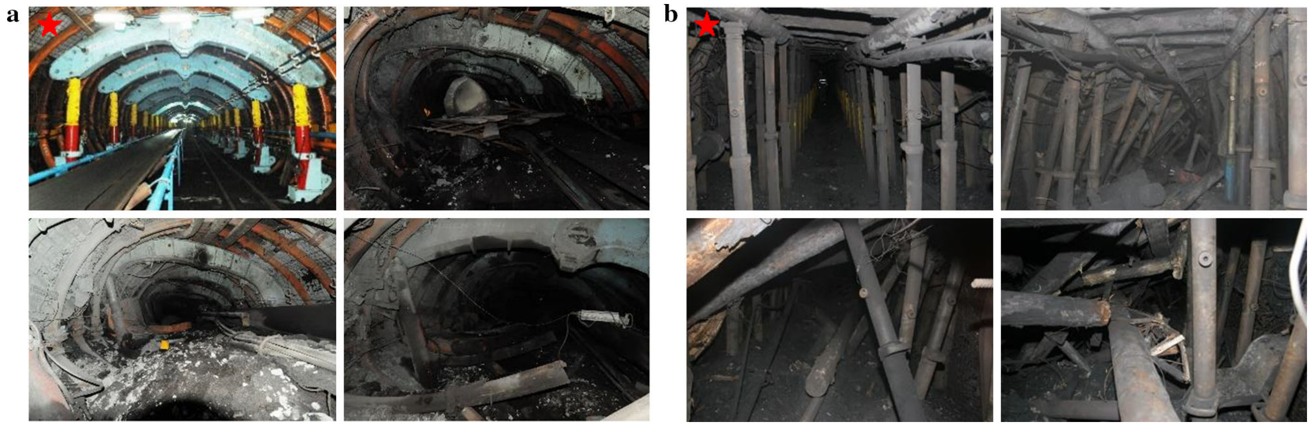

Fig. 6 In situ record photos showing roadway support and rock burst hazards of **a** the headgate of LW25110 and **b** a gob-side roadway in Xing'an coal mine where the roadway stagger layout method was not used. Photos with the *symbol star* were taken without a rock burst; others were taken after a rock burst

reduces the static stress within the coal-rock mass surrounding the gob-side roadway, and (2) attenuates the dynamic stress transmitted to the roadway. As a result, the total stress within the coal-rock mass surrounding the gob-side roadway is less likely to reach the critical stress level and thus the rock burst risk is concomitantly reduced. These two aspects are discussed in the following paragraphs. It should be noted that this layout can only reduce the total stress within the surrounding coal-rock mass in a limited area around the gob-side roadway rather than along the entire longwall panel. Hence, the gob-side roadway is safe, but the rock burst risk still exists in other areas (e.g., the coal face, and the solid coal-side roadway).

Figure 7 shows the vertical stress distribution adjacent to gob areas. There are an abutment stress zone and de-stressed zones in Fig. 7a. Field investigation (Qian and Shi 2003) shows that the abutment stress zone within the coal normally ranges from 15 to 30 m and can reach 35 to 40 m in a minority of cases, and that the abutment stress peak is usually 15–20 m away from gob (*i.e.*, 15 m $\leq S_1 \leq$ 30 m or 35 m $\leq S_1 \leq$ 40 m, and 15 m $\leq S_2 \leq$ 20 m). In cases of normally arranged roadways, the gob-side roadway is 6 to 30 m away from gob (i.e., 6 m $\leq S_3 \leq$ 30 m), which implied that the roadway lay mostly in the abutment stress zone. Figure 7b shows the numerical simulation result assuming that the cover depth was 800 m and the bulk unit weight of the overlaying strata was 25 kN/m^3. Thus, counter 2 is equal to the in situ stress. It is seen that the vertical stress within the floor and below the gob was much less than the in situ stress, which implied that the roadway is in a de-stressed zone in the case of a staggered layout. According to Fig. 7b, it was concluded that the vertical stress around the tailgate of LW25110 was less than 25 % of the in situ stress. This stress was significantly reduced when compared with normally arranged cases.

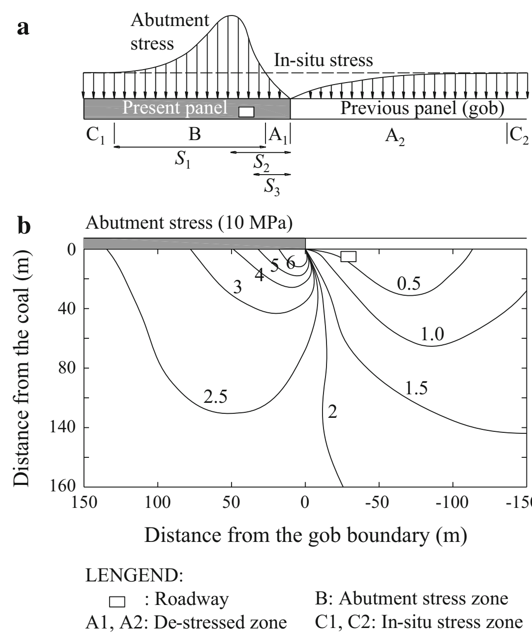

Fig. 7 Vertical stress distribution within **a** the coal seam and **b** the floor strata adjacent to gob areas. **b** Derived from Qian and Shi (2003)

Mining-induced tremors are precursor to rock bursts because they increase the stress and energy within the coal-rock mass around the roadway. Tremors transmit energy to the far-field in the form of seismic waves. Along their path, seismic waves are attenuated. Experimental studies (Gao et al. 2007) reveal that tremor energy attenuates exponentially with distance from the source in geo-materials, as shown in Eq. (1). The attenuation index η is considerably smaller in dense, stiff, rock and soil media, while much larger in cracked, loose, media. For example, in intact cement grout and, fine, sand, the attenuation index is 1.1509 and 2.1309, respectively. In cases of normally arranged roadways, the roof strata of the gob-side roadways are intact rock. The attenuation of seismic waves is limited when passing through this intact rock. In the case of LW25110, roof strata above the gob-side roadway (tailgate) contain cracked and broken rock. These rocks attenuate seismic waves significantly. As a result, the influence of tremor-induced dynamic stress on the tailgate is mitigated, making rock bursts less likely to occur therein.

$$E = E_0 l^{-\eta} \quad (1)$$

where E is energy at a given location, E_0 is energy of the tremor source, l is the distance from the source, and η is the attenuation index of the geo-material.

4 Conclusion

Application of a staggered roadway layout in LW25110 of Yuejin coal mine verifies that this method is favorable to gob-side rock burst control. The mechanism underlying this method for effective control of GRBs lies in the facts that: (1) the method reduces the static stress within the coal-rock mass surrounding the gob-side roadway, and (2) it attenuates those dynamic stresses transmitted to the roadway.

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Appendix: Roadway Stagger Layout Method

The roadway stagger layout method was first proposed by Zhao and Wu (2002) for the main purpose of raising coal recovery ratios in top-coal caving mining. Subsequently, Zhao (2004) discussed the details of this roadway layout system and the method was successfully applied in coal seams with an inclination of 0° to 25°. The layout system may be described as follows:

- The headgate and tailgate of a longwall panel are located at different horizons within a thick coal seam (i.e., along the top or bottom of the coal seam): when the inclination is small, the headgate (gob-side roadway, used for inputting fresh air) is disposed along the bottom of the coal seam, and the tailgate (solid coal-side roadway, used for outputting waste air) is disposed along the top of the coal seam, as shown in Fig. 8a; when the inclination is large, the headgate (solid coal-side roadway, used for inputting fresh air) lies along the top of the coal, and the tailgate (gob-side roadway, used for outputting waste air) lies along the bottom of the coal, as shown in Fig. 8b.

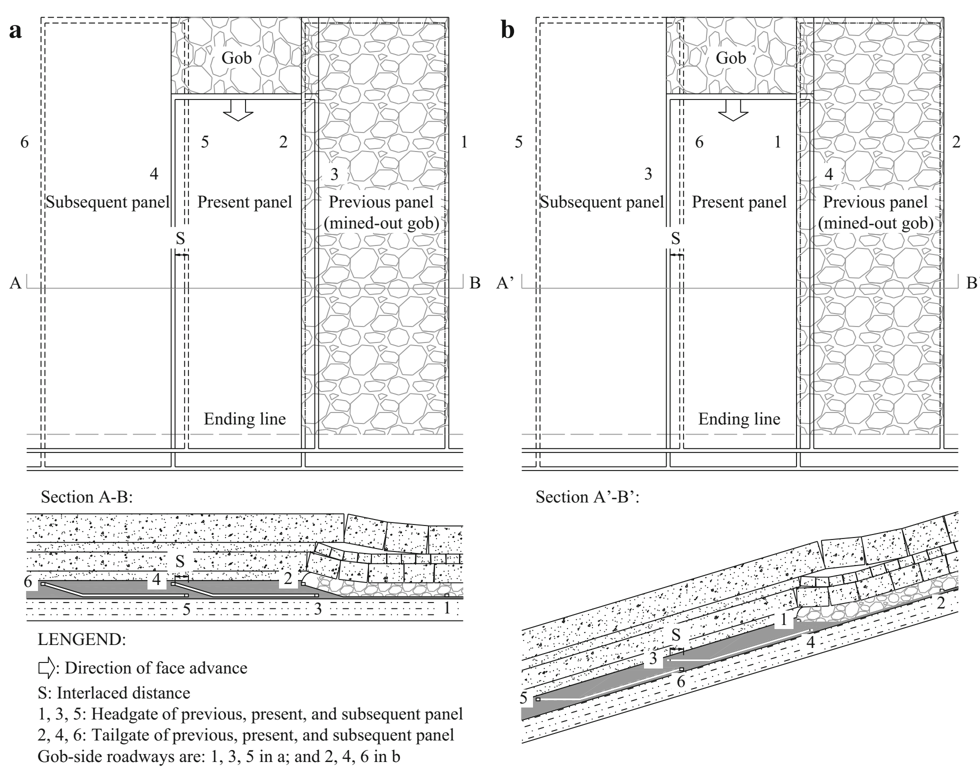

Fig. 8 The staggered roadway layout system in a thick coal seam with **a** a small inclination and **b** a large inclination

- The gob-side roadway (headgate or tailgate) is arranged so as to sit just below the previous mined-out panel.
- Adjacent panels are interlaced and there are no coal pillars between them.
- The interlaced distance *S* (i.e., the distance from the gob-side roadway of the present panel to the solid coal-side roadway of the previous panel) is normally not less than 3–5 m according to certain mining-geological conditions.

Advantages: the coal recovery ratio was higher, gob-side roadways were easier to support, and there was much less coal left in gob areas and thus spontaneous combustion risks were reduced.

Disadvantages: rock strata above the gob-side roadway were cracked and broken coal fragments and rocks spalled off, when the coal seam was not thick enough, the broken coal fragments and rocks may be directly next to the top surface of the roadway, making the top of the roadway an air-leakage source during ventilation of the panel. In addition, water in the previously mined-out panel may easily flow into the gob-side roadway, making the roadway muddy.

The disadvantages may be overcome by laying reinforcing mesh above the gob-side roadway and grouting the broken coal fragments and rocks.

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