Research Article

Technology of Coal Seam Long Borehole Blasting and Comprehensive Evaluation Method of Pressure Relief Effect in High Rockburst Proneness Longwall Panel

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On August 2, 2019, a catastrophic rockburst disaster occurred in Tangshan mine, causing death of 7 miners. After the investigation, the coal mine is facing reproduction. Taking the 0291 panel as the engineering background, this paper studies the coal seam blasting pressure relief and ground stress monitoring technology in working face retreat. During the roadway development and working face excavation, coal seam blasting was adopted to transfer the high ground stress of coal seam to the deep ground of the coal body. The blasting operation is presented in detail in this paper. In the working face retreat stage, drilling powder method, hydraulic shield resistance monitoring, roof displacement, and vibration monitoring methods are implemented. The results show that the pressure relief range of coal seam is 4–12 m in the coal mass after blasting. The shield working resistance is stable at 20–30 MPa. The range of relative displacement of the roof is about –1.0 to 2.5 mm, and the maximum vertical vibration velocity is in the range of 7–11 cm/s, up to 12 cm/s. The measured parameters are acceptable, so it is concluded that 0291 panel can be safely mined. This study provides a reference for the coal seam blasting design for rockburst coal mine and provides a technical means for the analysis of pressure release effect and dynamic pressure monitoring during working face retreating.

1. Introduction

Rockburst is a dynamic phenomenon of violent destruction of coal and rock mass in roadway or working face in the process of coal mining, which is often accompanied by coal and rock mass throwing, loud noise, and air wave. It is one of the major disasters in coal mine. With the depletion of shallow resources, coal mining extends to the deep underground. The geo-structure and stress environment of underground rock mass are becoming complicated, resulting in more intensity and high frequency disasters in deep mining compared with shallow engineering. According to statistics, by the end of 2020, there are 138 rockburst mines in China [1]. On August 2nd, 2019, a serious rockburst accident occurred at the Tangshan coal mine, causing 7 deaths.

The essential method to prevent rockburst is to reduce the ground stress of surrounding rock or migrate the high

stress to the far ground [2–4]. Coal seam blasting, water injection, and borehole drilling are pressure relief manner to reduce the risk of rockburst. Among them, coal seam blasting can effectively release the elastic deformation energy accumulated in coal and rock mass, change the structure of the surrounding rock, and reduce the stress concentration, so it can be used to prevent rockburst disaster.

In view of the mechanism of coal seam blasting on the prevention of rockburst, Zhang et al. [5, 6] developed strength weakening and impact reduction theory. The peak stress area of the roadway rib is moved to the deep coal and rock mass via blasting, and the stress concentration near the roadway surface is reduced. Qi et al. [7–9] put forward the stress control theory, which believes that seam blasting causes stress redistribution of coal and rock mass. It can effectively reduce the ground stress level and move the peak stress far away from the rib to reduce impact risk. Xue et al.

[10–12] put forward a hydromechanical coupling model and analyzed the damage characteristics of coal seams during gas fracturing and put forward the fracture criterion of type-I fracture under the action of multiple stress states in the seams fracturing process.

In terms of influencing factors and application of seam blasting, Pan et al. [13–15] proposed hole blasting technology based on impact start-up theory and established a “double peak” model of blasting pressure relief and impact prevention. The field test of blasting conducted by Sun [16–18] in Gushan coal mine shows that the prevention of rockburst by blasting is a process of modification-stress reduction-energy dissipation. It leads to surrounding rock support deterioration, structural instability, and roadway large deformation. Ouyang [19] proposed multistage blasting technology for roof, floor, and coal seam under the dual influence of extremely thick and hard roof and complex geological conditions.

This paper uses the reproduction of Tangshan coal mine after 82 rockburst disaster as engineering background. Coal seam blasting was implemented to a panel before and after working face retreating. Blasting design and operation are presented in detail. The technics to evaluate the pressure relief effect after blasting are manipulated. This paper provides a case study on the coal seam blasting design, operation, and pressure relief effect monitoring technology of high rockburst proneness coal mine, which provides a reference for similar engineering projects.

2. Study on Blasting Parameters

During the explosion, a cavity is formed at the explosion point. Compression fracture zone, fracture zone, and elastic vibration zone are successively formed from the center to outside [20]. The blasting technology can also be applied to prevent coal and gas outburst, rock softening, high ground stress chamber, and large deformation [21–24].

In terms of blasting parameters, Wang [25] used numerical simulation to analyze five key parameters affecting the pressure relief effect, namely, the distance from starting point of the pressure relief area to the roadway surface, the distance from the end of the pressure relief area to the roadway surface, the radial range of the pressure relief area, the depression angle of the pressure relief hole, and the spacing of the pressure relief area.

Liu et al. [26] selected 6 influencing factors of blasting and 3 indexes on pressure relief blasting parameters to conduct numerical calculation and orthogonal test. It is found that the optimal parameters are 3# coal, 8 kg charge, 90 mm hole diameter, 15 m hole depth, 3 m hole spacing, and decoupling coefficient is 4. Wei et al. [27] used FLAC3D to simulate the stress change and migration in the roadway before and after pressure relief under different blasting parameters. It shows that the larger the charge, the more the transfer of the stress peak area to the deep ground. However, the peak stress magnitude does not decrease significantly. The greater the thickness of coal seam, the more the decrease in stress peak, but the transfer distance of stress peak area is

about 12 m. The greater the mining depth, the smaller the transfer distance from the stress peak area to the depth.

After blasting, a pressure relief zone is formed, which transfers the high ground stress to the deep ground. It is necessary to calculate the radius of compression fracture zone and fracture zone formed by blasting. Under the condition of uncoupled charge in the infinite medium, the pressure of transmitted shock wave generated after cylindrical charge explosion is as follows [28, 29]:

$$P = \frac{1}{2(1+\gamma)} \rho_0 V_d^2 K^{-2\gamma} l_c n, \quad (1)$$

where p is the pressure of shock wave transmitted into coal, MPa; ρ_0 is the explosive density, kg/m³; V_d is the detonation velocity of explosive, m/s; K is the radial decoupling coefficient of charge, $K = r_b/r_c$, r_b and r_c represent the radius of explosion hole and charge radius, respectively, mm; and l_c is the axial coefficient of charge. When there is no air column in the axial direction, $l_c = 1$; γ is the expansion adiabatic index of explosion products, generally taken as 3; n is the stress increase coefficient during expansion, normally taken as 10.

The explosion stress in the coal body can be solved as a plane strain problem, and the 3D stress field in the coal body can be expressed as follows:

$$\sigma_r = P\bar{r}^{-\alpha}, \quad (2)$$

$$\sigma_\theta = -\lambda\sigma_r, \quad (3)$$

$$\sigma_z = \mu_d(1+\lambda)\sigma_r, \quad (4)$$

where σ_r , σ_θ , and σ_z represent radial, tangential, and axial stresses, respectively; $\bar{r}$ is the contrast distance, $\bar{r} = r_i/r_b$, r_i is the distance from the calculation point to the charging center and r_b is the radius of the blasting hole; α is the pressure attenuation coefficient, in compression crushing zone $\alpha = 2 + \mu_d(1 - \mu_d)$ and in fracture zone $\alpha = 2 - \mu_d(1 - \mu_d)$; μ_d is the material dynamic Poisson's ratio, $\mu_d = 0.8\mu$, μ is the static Poisson's ratio of the material; and λ is the lateral pressure coefficient, $\lambda = \mu_d(1 - \mu_d)$; the stress analysis of blasting pressure relief is shown in Figure 1.

The expression stress intensity (equivalent stress) in coal and rock is as follows:

$$\sigma_i = \sqrt{\frac{(\sigma_r - \sigma_\theta)^2 + (\sigma_\theta - \sigma_z)^2 + (\sigma_z - \sigma_r)^2}{2}}. \quad (5)$$

Substituting equations (3) and (4) into (5), we get

$$\sigma_i = \frac{1}{\sqrt{2}} \sigma_r \left[(1+\lambda)^2 - 2(1-\lambda)^2(1-\mu) + (1+\lambda^2)^{1/2} \right]. \quad (6)$$

Let equation (6) $B = [(1+\lambda)^2 - 2(1-\lambda)^2(1-\mu) + (1+\lambda^2)^{1/2}]$ be simplified as $\sigma_i = (B/\sqrt{2})\sigma_r$.

According to Mises's criterion, the criterion of blasting compression fracture zone and fracture zone is as follows:

FIGURE 1: Stress analysis diagram of deep hole pressure relief.

$$\sigma_i > \sigma_{cd} \text{ (compression fracture zone),} \quad (7)$$

$$\sigma_i > \sigma_{td} \text{ (fracture zone),} \quad (8)$$

where σ_i is the stress at any point in coal and rock mass, MPa; σ_{cd} is the dynamic compressive strength of coal and rock mass, MPa; and σ_{td} is the dynamic tensile strength of coal and rock mass, MPa.

Substitute equations (1), (2), and (6) into (7) and (8), respectively, and the estimation formulas of radius R_c of compression fracture zone and radius R_p of fracture zone are as follows:

$$R_c = \left[\frac{\rho_0 V_d^2 n K^2 \gamma l_c B}{2\sqrt{2} (1 + \gamma) \sigma_{cd}} \right]^{(1/\alpha)} r_b, \quad (9)$$

$$R_p = \left[\frac{\sigma_{cd}}{\sigma_{td}} \right]^{(1/\beta)} R_c r_b,$$

where R_c is the radius of compression crushing area after blasting, mm; R_p is the radius of fracture area after blasting, mm; and r_b is the radius of blasting hole, mm.

3. Engineering Practice of Long Borehole Blasting of #9 Seam in Tangshan Coal Mine

3.1. Engineering Background. The studied area is 0291 panel of Tangshan coal mine. The buried depth of 0291 panel is about 800 m. The coal seam thickness is 8.6–11.7 m, with an average thickness of 10.5 m. It is semidark briquette. The inclination of the coal seam is 5° – 39° , with an average of 22° . Absolute gas emission is $16.51 \text{ m}^3/\text{min}$, and the absolute carbon dioxide emission is $1.82 \text{ m}^3/\text{min}$. Coal dust is explosible, and the spontaneous combustion period of coal seam is 10 months.

The coal seam, roof, and floor of 0291 panel have impact tendency. Microearthquake events appear continuously in the excavation and mining operations. In the roadway development, through the monitoring and analysis of the drilling cuttings method, the area with burst risk above the medium in the 0291 panel was identified, as shown in Figure 2.

3.2. Blasting Layout in Roadway Development. The principle of “pressure relief first” was adopted in the roadway development in medium burst risk segment. One single borehole blasting scheme is adopted around the heading machine, as shown in Figure 3.

One $\Phi 42 \text{ mm}$ borehole is arranged for head-on loosening blasting along the axis of the roadway. The rib blasting is arranged within 30 m from the heading face. The rib boreholes are 0.5–1.5 m away from the floor, and their interval is less than 10 m. The length of all blasting borehole is of 12 m. If the width of coal pillar is less than 5 m, no blasting hole is drilled.

#3 emulsion explosive specific for coal mining was used in all blasting. One single column charge is 300 mm in length and 300 g in weight. The charge quantity of each borehole is 10–15 columns. Two identical electric detonators were used in each hole. The sealing length was less than 1/3 of the borehole length. The blasting method is forward charging, parallel connection in holes, series connection between holes, and simultaneous initiation of 5 holes. Pressure releases first, and then excavation was operated in a circular manner; it ensures that there is always more than 5 m blasting section in the front of the heading face.

3.3. Blasting Layout in Working Retreat. During the face retreat of 0291 panel, the burst risk area is determined through the drilling powder method. In the risk area and 10–15 m outside of two ends, coal seam blasting is adopted, as shown in Figure 4.

The blasting began from 10–15 m from the burst risk area and ended 10–15 m away from the burst risk area. Blasting hole is $\Phi 42 \text{ mm}$, 12 m in length, and interval less than 5 m, and the drilling is 0.5–1.5 m away from the floor. The explosive and blasting method are the same as those during development. Blasting started on September 21, 2019, and ended by January 9, 2021, as shown in Table 1 and Figure 5. The blasting location included open cut, main gate, and tailgate.

A total of 309 blasting holes were drilled during working face retreat; among them, 16 were in open cut, 171 were in main gate, and 122 were in tailgate.

3.4. Monitoring Technology for Pressure Release Effect. In terms of the evaluation of pressure relief effect, several technics can be used combined or lonely, such as conventional electromagnetic radiation, microseismic, seismic CT, and drilling powder [30–33]. The following literature is noticeable. Cong et al. [34] proposed the test method of blasting pressure relief effect by using seam wave velocity

TABLE 1: 0291 panel blasting working logo.

Date	No.	Position	
2019.09.21	17	Tailgate 90–125 m to entry	Both ribs
2019.09.22	17	Tailgate 116–152 m to entry	
2019.09.23	52	Tailgate 156–358 m to entry	Mine rib
2019.09.24	12	Tailgate 377–422 m to entry	Mine rib
	32	Main gate 101–163 m to entry	Both ribs
2019.09.25	38	Main gate 166–242 m to entry	Both ribs
	6	Tailgate 428–446 m to entry	Mine rib
2019.09.26		Main gate 246–270 m to entry	Pillar rib
	30	Main gate 627–680 m to entry	Both ribs
2019.09.27	40	Main gate 47–129 m to open cut	
2019.09.28	12	Main gate 25–43 m to open cut	
2020.07.17	2	Main gate 20 m to open cut	Both ribs
2020.07.20	2	Main gate 15 m to open cut	
2020.07.22	2	Main gate 10 m to open cut	
2020.08.05	1	Head gate of open cut	Pillar rib
2020.08.20	1	Tailgate 150 m to open cut	Mine rib
2020.08.28	2	Tailgate 138 m to open cut	
2020.08.29	3	Tailgate 126 m to open cut	
2020.08.30	3	Tailgate 114 m to open cut	Both ribs
2020.09.01	2	Tailgate 102 m to open cut	
2020.09.02	3	Tailgate 90 m to open cut	
2020.09.04	2	Tailgate 78 m to open cut	Pillar rib
2020.09.06	2	Main gate 12 m to open cut	
2020.09.09	2	Main gate 32 m to open cut	
2020.09.12	2	Main gate 52 m to open cut	
2020.09.13	1	Main gate 67 m to open cut	Mine rib
2020.09.15	1	Main gate 77 m to open cut	
2020.09.16	2	Main gate 91 m to open cut	
2020.09.23	2	Tailgate of open cut	Mine rib
2020.10.06	2	Open cut 11 m to tailgate	
2020.10.09	2	Open cut 23 m to tailgate	
2020.10.12	2	Open cut 35 m to tailgate	
2020.10.17	2	Open cut 47 m to tailgate	
2020.10.24	2	Open cut 59 m to tailgate	Both sides of working face
2021.01.01	2	Open cut 71 m to tailgate	
2021.01.07	2	Open cut 83 m to tailgate	
2021.01.09	2	Open cut 95 m to tailgate

FIGURE 5: Layout of borehole blasting during working face retreat.

In this study, the monitoring technology of drilling powder, hydraulic shield pressure, roof separation, and vibration monitoring were used to evaluate the effect of the pressure relief measure.

3.5. Drilling Powder. The length of the blasting borehole is 12 m, and the distance between the blasting point to the roadway rib is 9–12 m. The drilling powder was measured before and after blasting to evaluate the pressure relief. The

monitoring hole is $\Phi 42$ mm with a depth of 12 m. The results are shown in Figure 6. Compared with the coal powder generated before and after blasting, it shows that the pressure relief range is 6–12 m from the rib. The amount of pulverized coal in the explosion section decreases noticeably, which reflects the reduction of coal stress in this section. The amount of pulverized coal from 4–9 m decreased to some extent, indicating that the blasting influencing area is from 4–12 m.

3.6. Hydraulic Shield Resistance. The working face retreat began from October 2019. Before mining, the coal in front of the open cut has been blasted for pressure relief. Therefore, the monitoring data of the hydraulic shield working resistance are selected from March 2020, as shown in Figure 7. Among them, 77–98 shields are the upper support of the mining face, 28–71 shields are the middle support, and 7–21 shields are the lower support. The results show that the support resistance is in the range of 20–30 MPa, which is normal. After blasting, the working resistance of all shield supports is stable, and there is no severe pressure on the working face in the face retreating.

3.7. Roof Movement in Front of the Working Face. The roof movement was monitored during the working retreat. The monitoring duration is 3 days, while the working face retreat is 5 m.

The orifice is taken as the zero point, the position of each anchor point is calculated, and the data of the same anchor point are combined at different times to obtain the overall change of roof movement. The calculation method of anchor point position in the monitoring hole is (10). The L1 sensor data are taken as an example; the calculation result is shown in Figure 8.

$$S_i = a_i - (A - A_0), \quad (10)$$

where S_i represents the depth of each anchor point of the measuring hole, mm; a_i is the measured value of each anchor point in the measuring hole, mm; A represents the total length of the measured line of the Ionometer, mm, $A = 10$ m; and A_0 represents the hole depth of each measuring hole, mm.

The first anchor of each measuring point is selected as a fixed point. The displacement between other anchor points and the fixed anchor point can be calculated. The displacement in different dates is compared, and the movement of the roof at the measuring point can be determined. The calculation method of the displacement between each anchor point and the fixed anchor point is equation (11), and the relative displacement can be obtained using equation (12).

$$L_i = b_i - b_1, \quad (11)$$

$$L_s = L'_i - L_i, \quad (12)$$

where L_i is the distance between the i th anchor point and the first anchor point ($i = 2, 3, \dots, n$), mm; b_i is the measured

FIGURE 6: Typical drilling powder before and after blasting.

FIGURE 7: Working resistance of hydraulic shield during face retreat.

value of the i th anchor point, mm; b_1 is the measured value of the first anchor point, mm; n is the number of anchor points; and L'_i is the spacing value in different time periods, mm.

The relationship between the relative spacing of measuring points and the distance between measuring points and working face is shown in Figure 9.

Result shows that the relative displacement is in the range of -1.0 – $+2.5$ mm, which suggests that the displacement of the roof is acceptable under dynamic pressure. It is worthwhile to mention that the relative displacement has negative value, which indicated a vibration movement of the roof due to the disturbance of the mining activities, such as cutter movement or hydraulic shield advance.

3.8. Roof Vibration Monitoring. The TC-4850 vibrometer is used to monitor the roof vibration in front of the working face. The equipment was installed in the tailgate on June 26, 2020, and the monitoring period is 2 days.

The vibrometer was installed at the tailgate 7 m away from the working face, as shown in Figure 10. A total of 4

FIGURE 8: Overall displacement of the L1 sensor.

FIGURE 9: Relationship between relative displacement of the roof at L1 measuring point.

FIGURE 10: The arrangement of the vibrometers

vibrometers were installed, namely, Z1–Z4, respectively. The drilling position is 1.0 m away from the mining rib, and the installation hole is 6–7 m length in vertical direction. The probe is a cube with a dimension of 66 mm. A steel cable is anchored in the hole. At the exposed part of the cable, the probe is glued to the steel plate using gypsum. The collector is fixed on the metal mesh with rope. The trigger level of the

measurement is 0.2 cm/s, the acquisition rate is 8000 SPS (sample points/s), and the recording time is 5 s.

The waveform curve of each vibration can be obtained via interpretation of collected data, as shown in Figure 11. Since this monitoring is only aimed at the vertical movement of the roof, the data are screened accordingly. The data related to roof vertical movement are selected, and the

FIGURE 11: Vibration curve interpreted by workbench.

FIGURE 12: Z1 monitoring data diagram.

distribution of the maximum vertical velocity can be obtained. Figure 12 shows Z1 monitoring data as an example.

Result shows that along with the advances of the working face, the vibration of the roof in front of the working face increases. The maximum vertical velocity of the roof 7 m in front of the working face is about 6 cm/s, and it increases to 15 cm/s while the working face advances 3 m. However, the overall vibration intensity is still normal based on engineering experience. It suggests that the roof vibration is most likely attributed to the mining activity, such as coal cutting or hydraulic movement, rather than the dissipation of the energy release accumulated in the coal mass.

From monitoring data, it can be concluded that the panel can be mined safely after coal seam pressure release via seam blasting.

4. Conclusion

Taking the reproduction of Tangshan coal mine 0291 panel as the engineering background, this paper adopts the coal seam blasting pressure relief method during roadway development and panel excavation to reduce the burst risk. The pressure relief effect and the influence of mining disturbance on the surrounding rock around the working face are

monitored and analyzed by the jointed method in the mining stage. The following conclusions can be drawn:

- (1) Coal seam blasting can transfer high ground stress of coal seam to the deep ground of coal mass, leading pressure relief of coal body near the roadway rib. The results of drilling powder measurement show that the pressure relief range of coal seam blasting is 4–12 m. For a rockburst panel, the intensive blasting parameters should be adopted.
- (2) During working face retreat, the working resistance of the hydraulic shield is stable at 20–30 MPa. The result of roof movement monitoring shows that the relative displacement of the roof is about −1.0 to 2.5 mm. The overall displacement is small and shows a characteristic of vibration.
- (3) Roof vibration monitoring is a new technical means for roof motion monitoring. The monitoring results show that the maximum vertical vibration velocity of the roof in front of the working face is 7–11 cm/s, up to about 12 cm/s. Combining with the analysis results of other monitoring means, it is concluded that 0291 panel can be safely mined.

At present, the mining of 0291 panel has been underway, and the monitoring and analysis results have been verified. This study provides a reference for the blasting pressure relief design of deep rockburst mine and provides a technical means for the analysis of pressure release effect and dynamic pressure monitoring during mining.

Data Availability

The data used to support this study are included within the article.

Conflicts of Interest

The authors declare that there are no conflicts of interest regarding the publication of this paper.

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