



Study on the Control Effect of Surrounding Rock Stress in Coal Mine Roadways on Coal and Gas Outburst

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Abstract. In order to reveal the control mechanism of surrounding rock stress on coal and gas outburst, and make up for the insufficient quantitative research of mining stress and support stress in outburst prevention, taking E0103 transportation roadway of Guanyinshan Coal Mine in Yunnan as the engineering background, ZLGH three-direction borehole dynamometer and MGH bolt-cable dynamometer were used to monitor the mining stress evolution and support stress characteristics in situ. Residual gas pressure, residual gas content, drilling cuttings gas desorption index K_1 and drilling cuttings quantity S were measured synchronously. The results show that the mining stress increases first and then tends to be stable with roadway excavation, and the roof support stress is significantly higher than that of the rib. The optimized bolt-cable combined support forms a high support stress field, transfers the stress peak to the deep coal mass, reduces the concentrated stress on the coal wall, and makes all drilling cuttings quantity S below the critical value. The support stress blocks the outburst initiation conditions by improving the surrounding rock bearing capacity and optimizing the stress and energy field. The research provides a theoretical basis and engineering reference for roadway support optimization and outburst prevention in deep coal mines.

Keywords: Surrounding Rock Stress; Mining Stress; Support Stress; Coal and Gas Outburst.

1 Introduction

With the continuous increase in the mining depth of coal resources, deep rock masses exhibit distinct mechanical characteristics known as the "three highs and one disturbance"—namely, high in-situ stress, high gas pressure, high permeability pressure, and strong mining disturbance [1-2]. Roadway excavation disrupts the original in-situ stress equilibrium, triggering stress redistribution and persistent deformation of the surrounding rock. Concurrently, delayed support or insufficient support intensity can easily induce roof instability, rib spalling, and coal and gas outburst accidents. Such disasters

are particularly frequent at unsupported tunneling faces [3-4]. Coal and gas outburst is a dynamic disaster resulting from the combined effects of geo-stress, gas parameters, and the physical and mechanical properties of the coal mass. However, there is still a lack of quantitative indicators regarding the controlling effect of geo-stress, particularly mining stress, on outbursts. Furthermore, research on the correlation mechanism between surrounding rock support and outburst hazard remains weak [5-6].

Domestic scholars have conducted extensive research on the stress evolution of surrounding rock and the prevention and control of outbursts [7-10]. Kang and Bai have systematically proposed the theory and complete sets of technology for bolt support in coal roadways. They clarified the controlling effect of high prestress on the structural stability of surrounding rock and verified the application effect of bolt support in deep roadways through numerous engineering case studies [11-12]. Song and Fang have deeply investigated measurement methods for in-situ stress in deep mines, providing a technical foundation for field monitoring of underground stress [13-14]. Some researchers have applied methods such as expert systems, neural networks, and rheological analysis to support design and surrounding rock classification, promoting the quantitative and intelligent development of support parameter optimization [15-18]. Yang preliminarily discussed the controlling effect of high prestress bolt support on coal and gas outbursts, offering a theoretical framework for controlling outbursts through support [19]. Additionally, some scholars have focused on the mechanical response of surrounding rock and the timing of support in gateroads and half-coal-rock roadways, revealing the influence laws of mining stress on surrounding rock failure [20-21]. Globally speaking, existing studies mostly focus on either the stability control of roadway surrounding rock or the individual prevention of gas outbursts. The support stress of surrounding rock has not been systematically investigated as a core factor in outburst control, and the coupling effect between mining stress and support stress, as well as their quantitative controlling relationship with outbursts, remains ambiguous.

Taking the E0103 transportation roadway of Guanyinshan Coal Mine in Yunnan Province as the engineering background, this study monitors the evolution laws of mining stress and support stress in situ. The coal seam gas parameters and local outburst prediction indicators are measured, and the controlling effect of surrounding rock stress on coal and gas outbursts is analyzed. This research aims to provide a theoretical basis and technical reference for safe tunneling and disaster prevention and control in deep, highly outburst-prone mines.

2 Experimental Methods and Instruments

A ZLGH vibrating wire three-direction borehole dynamometer was utilized for mining stress monitoring. It is configured with one vibrating wire stress sensor in each of the X, Y, and Z directions, which directly bears the load via pressing sheets. The signals are transmitted through three 4-core cables, and a matching GSJ2A intelligent detector is used to read the stress data. The measuring range of the ZLGH dynamometer in the X, Y, and Z directions is 25 MPa, with a sensor frequency of 1300–2100 Hz, an accuracy of 1.0% FS, an operating temperature of -30°C to 75°C, a waterproof pressure of

less than 3 MPa, external dimensions of $\Phi 80 \text{ mm} \times 280 \text{ mm}$, and an installation pipe diameter of $\Phi 32 \text{ mm}$. The GSJ-2A detector is a portable, intrinsically safe instrument capable of storing 64 sensor constants and 500 groups of measurement data.

The monitoring sections for support stress were located close to the driving face to monitor the variations in bolt and anchor cable support stress throughout the entire excavation process. The monitoring commenced immediately upon the installation of the bolts or anchor cables and concluded when the data stabilized. An MGH vibrating wire bolt-cable dynamometer was employed as the monitoring instrument. The monitoring equipment is illustrated in Fig. 1.

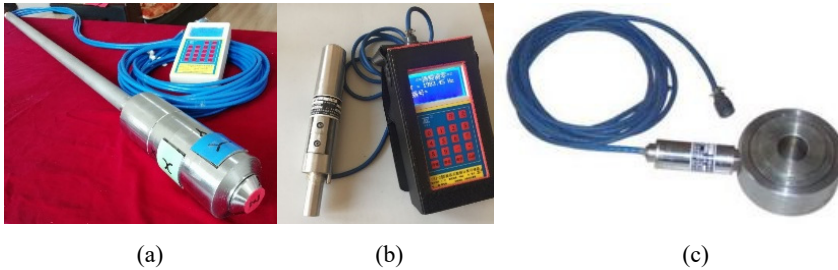


Fig. 1. Physical photograph of the monitoring equipment: (a) ZLGH vibrating wire three-direction borehole dynamometer; (b) GSJ-2A intelligent detector; (c) MGH vibrating wire bolt-cable dynamometer.

3 Experimental Scheme

The experimental site was located in the E0103 transportation roadway of Guanyinshan Coal Mine in Yunnan Province. Five groups of measuring points were arranged along the tunneling direction within the E06 drainage roadway (a floor roadway). For each group, one cross-bedding borehole (W1–W5) was drilled to measure the coal seam gas pressure and gas content, and another cross-bedding borehole (d1–d5) was constructed to monitor the mining stress of the coal seam. The spacing between the final holes of the stress monitoring boreholes and the gas pressure measurement boreholes was kept greater than 15 m.

During the roadway excavation, support stress monitoring section I (E03-Y8+11m) and section II (E03-Y9+0m) were established along the driving direction to monitor the bolt pre-tightening force and the deep displacement of the surrounding rock. Dynamometers #14, #16, #18, #20, and #22 were deployed in section I, while dynamometers #13, #15, #17, #19, and #21 were deployed in section II. The layout of the measuring points is depicted in Fig. 2 and Fig. 3.

To ensure temporal consistency, all data acquisition for mining stress (d1–d5) and gas parameters (W1–W5) was synchronized with the progress of the roadway. The sampling time windows for all sensors were aligned to identical geological advancement intervals, ensuring that the comparative analysis reflects the stress-deformation state at corresponding mining stages.

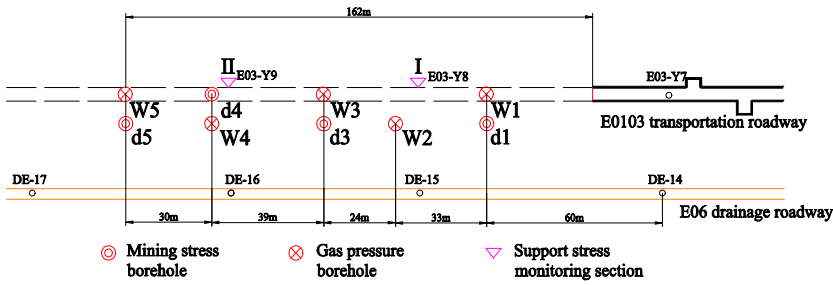


Fig. 2. Plan layout of measuring points in E0103 transportation roadway

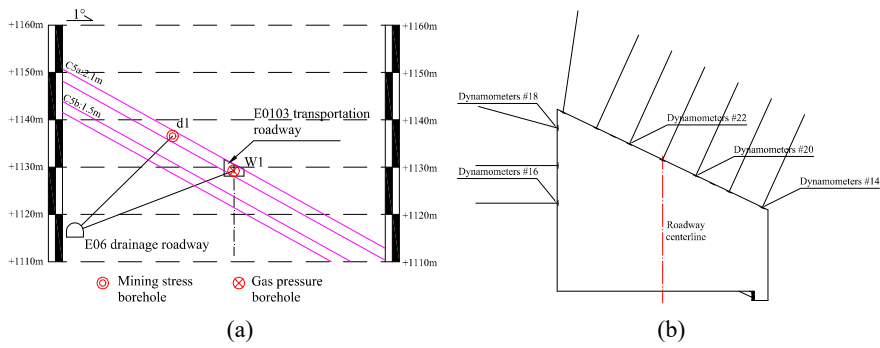
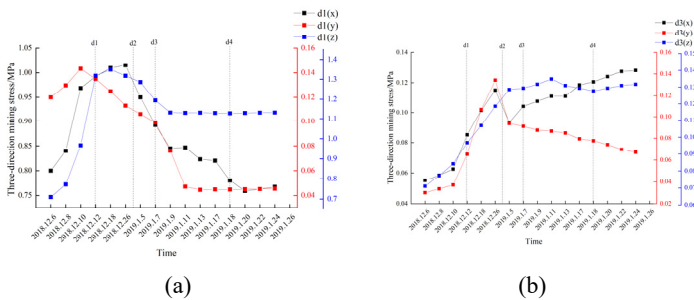


Fig. 3. Cross-sectional layout of measuring points in E0103 transportation roadway: (a) Mining stress monitoring section; (b) Support stress monitoring section.

4 Experimental Results and Analysis

4.1 Results of Mining Stress Monitoring

The coordinate system is defined relative to the roadway orientation: the *x*-axis is parallel to the excavation direction, the *y*-axis is perpendicular to the borehole axis, and the *z*-axis represents the borehole direction. The observed discrepancies in stress magnitudes across these axes (Figure 4) are primarily attributed to anisotropic geological conditions and mining-induced stress redistribution.



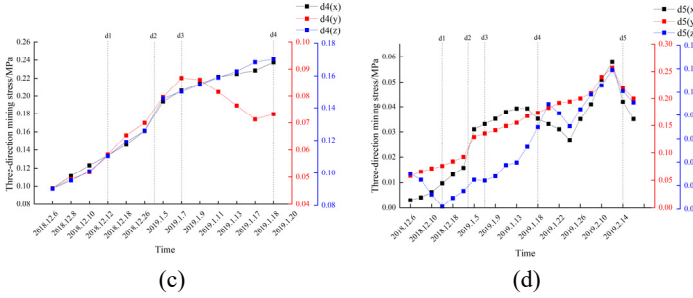


Fig. 4. Variation curves of mining stress over time at different measuring points: (a) point d1; (b) point d3; (c) point d4; (d) point d5.

During the experimental period, the stress gauge at measuring point d2 was damaged during installation, failing to yield valid data. The variation curves of the three-direction mining stress over time at measuring points d1, d3, d4, and d5 are shown in Fig. 4.

The monitoring results reveal a consistent trend where the mining stress initially rises to a peak due to the advancing tunneling face and subsequently stabilizes as the excavation disturbance weakens. The specific variations are as follows:

- Point d1: The stress gradually increased and peaked around the d1 monitoring section (d1(x): 1.02 MPa, d1(y): 0.14 MPa, d1(z): 1.35 MPa), before decreasing to a stable state.
- Point d3: The stresses peaked before the roadway advanced to the d3 section (d3(x): 0.11 MPa, d3(y): 0.14 MPa, d3(z): 0.13 MPa). Afterwards, the d3(y) and d3(z) stress decreased, while the d3(x) stress progressively increased until stabilization.
- Point d4: The stress in the d4(x) and d4(z) directions peaked at approximately 0.24 MPa and 0.17 MPa, respectively, before the tunneling face reached the d4 section. Conversely, the d4(y) stress peaked earlier at 0.09 MPa near the d3 section, decreased, and then rebounded before reaching the d4 section.
- Point d5: The stress slowly increased and reached their respective peaks (0.06 MPa, 0.25 MPa, and 0.12 MPa) prior to the d5 section, followed by a rapid decline.

Although this study focuses on in-situ monitoring, the observed stress evolution patterns—characterized by initial peak concentration followed by stabilization—show strong qualitative agreement with established numerical simulation results [22] for similar coal seam conditions, validating the reliability of the field data.

4.2 Results of Support Stress Monitoring

The variation curves of the bolt support stress over time for section I and section II are shown in Fig. 5.

As can be seen, the support stress of each dynamometer in section I rapidly increased to its maximum value within 1 to 2 days. Then, with the reduction of excavation influence, the support stress gradually decreased and tended to stabilize. Specifically, dynamometers #20 and #22 located at the center of the roof directly bore the surrounding

rock stress, with their support stress reaching a maximum of 4.4 MPa and eventually stabilizing at 2–3 MPa. Conversely, the support stresses of dynamometers #16 and #18 on the high rib were relatively small, ultimately stabilizing at 0–1 MPa.

Similarly, the support stress of each dynamometer in section II rapidly increased to its maximum value within 1 to 2 days, followed by a gradual decrease and stabilization as the excavation influence faded. Dynamometers #13, #15, and #19 on the roof directly sustained the surrounding rock stress, exhibiting a maximum support stress of up to 5.8 MPa and eventually stabilizing at 3–5 MPa. Meanwhile, the support stresses of dynamometers #17 and #21 on the high rib were smaller, stabilizing at 0–2 MPa.

The overall data variation indicates that the roof support stress is significantly higher than that of the roadway rib, and the support stress rises rapidly after installation before gradually stabilizing.

The relationship between support stress and excavation distance reveals a clear two-dimensional spatiotemporal response. As the excavation face approaches, support stress rapidly peaks, reflecting the dynamic impact of mining-induced loading. This evolution confirms that the support system effectively manages the stress peak shift in the time-space domain as the face advances.

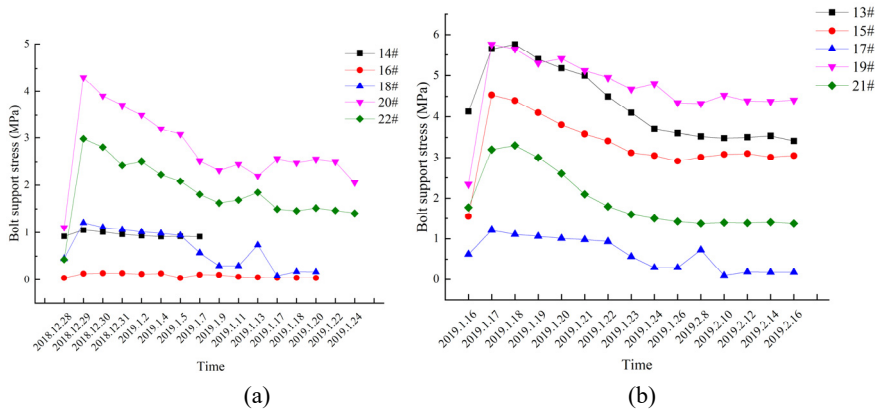


Fig. 5. Variation curves of bolt support stress over time: (a) Section I; (b) Section II.

4.3 Determination Results of Residual Gas Pressure and Content

The determination results of the residual gas pressure and residual gas content at measuring points W1–W5 are shown in Fig. 6.

It is apparent that the maximum residual gas pressure at measuring points W1–W5 does not exceed 0.12 MPa, and the maximum residual gas content does not exceed 7 m³. This is because the mine had previously conducted cross-bedding borehole gas drainage for the E0103 transportation roadway within the E06 gas drainage roadway. Consequently, regional outburst prevention measures had been implemented across the experimental section of the coal roadway tunneling face, ensuring that both the residual gas pressure and residual gas content remained below their respective critical values.

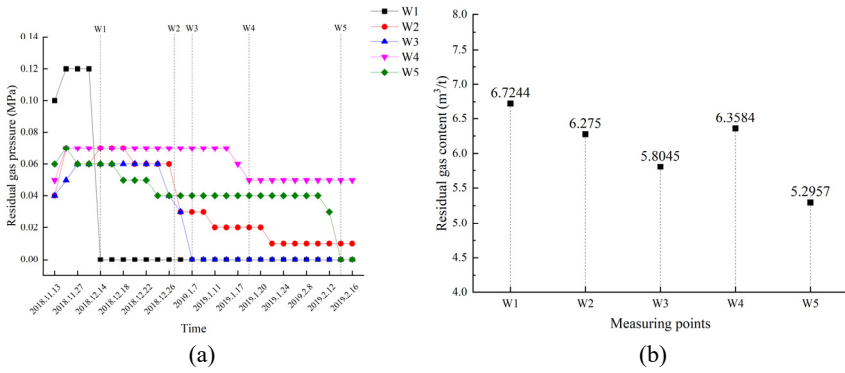


Fig. 6. Determination results of residual gas parameters: (a) Residual gas pressure; (b) Residual gas content.

4.4 Determination Results of Drilling Cuttings Gas Desorption Index

During the excavation of the experimental coal roadway section, the drilling cuttings gas desorption index K_1 and the drilling cuttings quantity S were employed to conduct local predictions of the coal seam outburst hazard for each operational cycle. The testing results are presented in Fig. 7.

It can be observed that during the excavation of the experimental section, the locally predicted K_1 values for each cycle were all below the critical threshold of 0.5. However, under the original support scheme, the drilling cuttings quantity S in several cycles exceeded the critical value of 6 kg/m, accompanied by gas dynamic phenomena such as rib spalling and coal popping in the high rib of the coal roadway. In contrast, when the optimized support scheme was adopted, the drilling cuttings quantity S for every cycle remained below the critical value of 6 kg/m, and no other anomalies were detected.

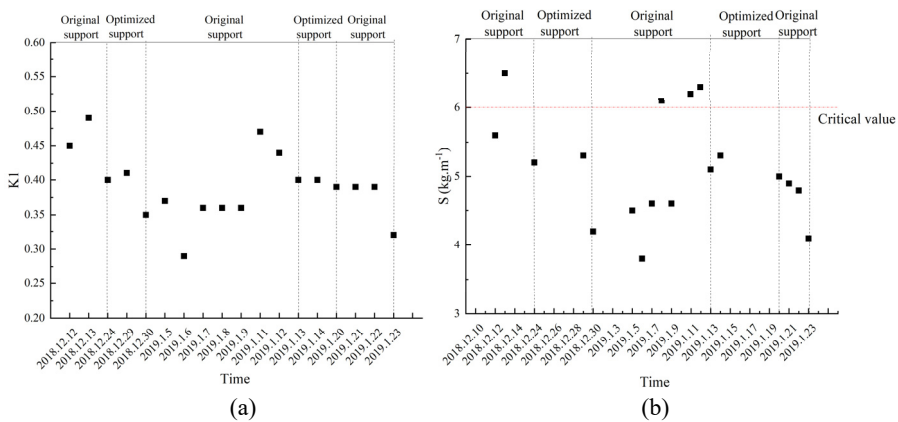


Fig. 7. Prediction results of local outburst indicators: (a) Drilling cuttings gas desorption index K_1 ; (b) Drilling cuttings quantity S .

4.3 Controlling Effect of Surrounding Rock Stress on Coal and Gas Outburst

The prestress from bolts and anchor cables forms an integrated compressive zone at the tunneling face. High prestress extends this compressive zone further into both the unsupported roof and the area ahead of the working face. Compared with standalone bolt support, standalone anchor cable support generates a smaller stress concentration zone ahead of the tunneling face and reduces both the area and intensity of energy concentration along the roadway ribs.

Mining stress redistributes the surrounding rock stress, leading to a highly non-uniform permeability in the coal seam. Within the 0–4 m pressure-relief zone from the coal wall, fractured coal dramatically increases permeability (by dozens to hundreds of times), rendering support stress negligible to gas flow. However, within the 5–10 m range, high support stress significantly suppresses coal permeability, whereas beyond 10 m, gas flow stabilizes due to minimal porosity changes.

The combined support of bolts and anchor cables establishes a high support stress field that transforms the support system and the surrounding rock into an active load-bearing structure. This configuration shifts the stress peak deeper into the coal mass, thereby reducing the gas pressure gradient. Consequently, this joint mechanism effectively improves the macroscopic stress conditions of the surrounding rock, thereby blocking the stress conditions required for outburst initiation, although the underlying microscopic mechanisms—such as microcrack propagation—warrant further verification.

5 Conclusions

(1) Field monitoring successfully captured the evolution laws of mining stress and support stress in the E0103 transportation roadway: the mining stress first increases to a peak value with the roadway excavation and then gradually stabilizes; the roof support stress is significantly higher than that of the roadway ribs; and the support stress reaches its maximum within 1 to 2 days before flattening out.

(2) The optimized combined support of bolts and anchor cables can form a high support stress field, transferring the horizontal stress ahead of the tunneling face deep into the coal mass. This reduces the concentrated stress on the coal wall, keeping both the drilling cuttings gas desorption index K_1 and the drilling cuttings quantity S below their critical thresholds, and eliminating dynamic phenomena such as rib spalling and coal popping.

(3) Mining stress governs the redistribution of surrounding rock stress and the modification of gas permeability. Support stress controls coal and gas outbursts effectively by enhancing the bearing capacity of the surrounding rock, transferring stress peaks, and reducing the elastic energy of the coal-rock mass, thereby blocking the stress and energy conditions required for outburst initiation.

(4) Roadway surrounding rock stress (comprising mining stress and support stress) serves as the critical controlling factor for coal and gas outbursts. Strengthening high-prestress support can optimize the distribution of stress and energy fields, providing a

feasible technical pathway for roadway support design and outburst disaster prevention and control in deep, highly outburst-prone mines.

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Disclosure of Interests

The authors have no competing interests to declare that are relevant to the content of this article.

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