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Editorial

Special Issue: Rock Burst Disasters in Coal Mines

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Rock bursts are often encountered in coal mines worldwide. In China, rock bursts have been a major safety concern in underground coal mines for more than 50 years. Due to the large uncertainty and variability in rock mass properties (e.g., rock mass strength) and boundary conditions (e.g., in situ stress), the unpredictable and recurrent nature of the phenomenon poses a major safety problem in many hard rock mines and coal mines. The highly frequent occurrence of rock burst disasters in coal mines still seriously restricts the safety and efficiency of mining coal resources deep underground. Thus, an in-depth understanding of the failure mechanism of rock bursts is of the utmost importance for the prediction, mitigation and control of geological disasters in deep underground mining and engineering projects.

Therefore, we established a Special Issue of *Energies* on the subject area of "Rock Burst Disasters in Coal Mines" (https://www.mdpi.com/journal/energies/special_issues/RBD_CM), which closed on the deadline of 31 May 2022. This Special Issue has published 13 high-quality papers. The details of these accepted papers in the Special Issue are summarized individually as follows:

(1) Wang, X. et al., Numerical Study on Strength and Failure Behavior of Rock with Composite Defects under Uniaxial Compression [1].

The very first accepted paper of the Special Issue on "Rock Burst Disasters in Coal Mines" presented a numerical study on seven combined models of single circular hole and double cracks with different angles. The results show that the combined defects reduce the strength of the model. Meanwhile, the distributions of the stress and displacement are changed by the cracks with different angles in the defective models.

(2) Wang, Z. et al., Longwall Top-Coal Caving Mechanism and Cavability Optimization with Hydraulic Fracturing in Thick Coal Seam: A Case Study [2].

This paper analyzed the failure mechanisms of the top-coal in a thick coal seam and cavability improvement induced by hydraulic fracturing. Based on the geological and geotechnical conditions of the Dongzhouyao coal mine, it is revealed that top-coal failure mechanisms are dominated by both compressive and tensile stresses. Ahead of the face line, shear failure initiates at the lower level of the top-coal and propagates to the upper level. Compressive-stress-induced damage leads to obvious deterioration in tensile strength, causing the onset of tensile failure in the top-coal behind the face line. Accumulated plastic strain (APS) is selected as a top-coal cavability indicator. The cavability degrades gradually at the higher elevation of the top-coal while it is greatly strengthened as the top-coal approaches closer to the face line. In a thick coal seam without hydraulic fractures, the maximum APS occurs at the middle section of the face length in the Longwall top-coal caving (LTCC) panel. After hydraulic fracturing, top-coal cavability is significantly enhanced. However, the spatial distribution of the APS transitions from being uniform to non-uniform due to the existence of hydraulic fractures, causing great variety in the cavability along the panel width. With increasing fracture intensity and fracture size, the

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failure zone expands significantly ahead of the longwall face, which means the cavability becomes increasingly favourable.

(3) Xiong, Y. et al., Instability Control of Roadway Surrounding Rock in Close-Distance Coal Seam Groups under Repeated Mining [3].

This paper studied the mechanism and control technology of surrounding rock instability under repeated mining via indoor testing, field testing, physical similarity simulation experiment, and numerical simulation. The results show that the surrounding rock of roadways has low strength, low bearing capacity, and poor self-stabilization ability, and it is vulnerable to engineering disturbances and fragmentation. Affected by the disturbance under repeated mining, the roadway-surrounding rock cracks are developed, the sensitivity is strong, and it is prone to large-scale loosening and destruction. The location of the roadway is unreasonable, and the maximum principal stress of the roadway is 3.1 times of the minimum principal stress, which is quite different. Thus, under a large horizontal stress, the surrounding rock undergoes long-range expansion deformation. On the basis of this research, the direction and emphasis of stability control of roadway surrounding rock under repeated mining of coal seam groups in close distance are shown. A repair scheme (i.e., long bolt + high-strength anchor cable + U-shaped steel + grouting) is proposed that reduces the risk of roadway instability.

(4) Zhong, K. et al., Experimental Study on the Mechanical Behavior and Failure Characteristics of Layered Coal at Medium Strain Rates [4].

This study systematically analyzes the uniaxial compressive strength (UCS), acoustic emission (AE) characteristics, failure pattern, and risk of rock burst on coal specimens with two bedding orientations under strain rates ranging from 10^{-4} s^{-1} to 10^{-2} s^{-1}. The results reflect that the bedding direction and the strain rates significantly affect the UCS and failure modes of coal specimens. The UCS of the coal specimens with loading directions perpendicular to the bedding planes (horizontal bedding) increases logarithmically with the increasing strain rate, while the UCS increases first and then decreases in coal specimens with loading directions parallel to the bedding planes (vertical bedding). The AE cumulative energy of the specimens with horizontal bedding is an order of magnitude higher than that of specimens with vertical bedding. However, it is independent of the strain rates. The energy release rates of these two types of bedded coal specimens increase in a power function as the strain rate increases. The coal specimens with horizontal bedding show violent failure followed by the ejection of fragments, indicating a high risk of rock burst. On the other hand, the coal specimens with vertical bedding exhibit a tensile splitting failure with a low risk of rock burst. Strain localization is a precursor of coal failure, and the concentration area of local principal strain is highly consistent with the initial damage area, and the area where the principal strain gradient is significantly increased corresponds to the fracture initiation area.

(5) Zhou, X. et al., An Approach to Dynamic Disaster Prevention in Strong Rock Burst Coal Seam under Multi-Aquifers: A Case Study of Tingnan Coal Mine [5].

This paper considers the 207 working faces in the Tingnan Coal Mine as a study example, combined theories with numerical simulation and field work technologies to analyze the key contradiction between rock burst and water inrush, proposed an integrated method (roof pre-splitting at a high position and shattering at a low position), and put the method into practice. Based on the lithology and predicted caving height of the roof, the contradiction between rock burst and water inrush was analyzed. In light of these analyses, an integrated method, roof pre-splitting at a high position and shattering at a low position, was proposed. According to the results of numerical modelling, pre-crack blasting at higher rock layers enables a cantilever roof cave in time, thereby reducing the risk of rock burst. In addition, pre-crack blasting at underlying rock layers helps increase the crushing degree of the rock, which is beneficial for decreasing the caving height of the rock layers above the goaf, thereby preventing the occurrence of water inrush. Finally, the proposed method was applied in an engineering case, and the effectiveness of this method for the prevention and control of multi-dynamics disasters was evaluated by field observations of the caving height of rock layers and micro-seismic monitoring. As a result, the proposed method works well to integrally prevent and control rock burst and water inrush.

(6) Guo, X. et al., Full-Stress Anchoring Technology and Application of Bolts in the Coal Roadway [6].

This paper proposed a full-stress anchoring technology for bolts. Firstly, a mechanical relationship model of a bolt-drawing, anchoring interface was established to obtain the equations of the axial force and obtain shear stress distribution, as well as the decreasing-load transfer law of the anchoring section of bolts. Through studying the prestress-loading experimental device of bolts, we found that increasing the initial preload could increase the axial force under the same conditions, and the retarded anchoring section could control the axial-force loss of bolts in the middle of the anchoring section. Under the full-stress anchoring mode, the effect of applying a pre-tightening force was better than that of applying a pre-tightening force under traditional anchoring methods. Moreover, FLAC3D numerical simulation calculation was performed. Under the full-stress anchoring mode of bolts, the increased anchoring length reduced the damage of the anchoring section, with a wider control range in the rock formation and higher strength in the compressive-stress anchoring zone. Based on the above research, four methods for applying the full-stress anchoring technology of bolts in engineering were proposed. The full-stress anchoring technology of bolts in the coal roadway has been applied in the support project of the return-air roadway at working face 3204 of the Taitou Coking Coal Mine of the Xiangning Coking Coal Group, Shanxi. The maximum moving distance of the roof and floor of the roadway was reduced from 200 to 42 mm, and the maximum moving distance on both coal sides was reduced from 330 to 86 mm. The full-stress anchoring technology of bolts was able to control the surrounding rock in the coal roadway.

(7) Yue, X. et al., Stability and Cementation of the Surrounding Rock in Roof-Cutting and Pressure-Relief Entry under Mining Influence [7].

The excavation of the 360,803 airway in Xinji Mine No. 1 is affected by intense mining of the 360,805 working face. Hence, to address the stability problem of surrounding rock in the 360,803 airway, rock mass blast weakening theory was used in this study to analyze the blasting stress of the columnar charged rock mass and to obtain the radii of crushed, fractured, and vibration zones under uncoupled charging conditions. The reasonable array pitch, length, and dip angle of boreholes were determined according to the pressure–relief range of the blasting fracture. The migration laws of roof strata were explored based on a mechanical model of overlying roof strata structure on the working face. Subsequently, the horizon, breaking span, and caving sequence of hard roof strata were obtained to determine the roof-cutting height of this entry. On the basis of the theory of key stratum, the number of sequences at the roof caving limit stratum and hanging roof length in the goaf were calculated, the analytical solution to critical coal pillar width was acquired, the evaluation indices for the stability of entry-protecting coal pillars were determined, and the engineering requirements for the 25 m entry-protecting coal pillars in the 360,803 airway were met. Moreover, various indices such as roof separation fracture, displacement of surrounding rock, and loose circle of surrounding rock in the gob-side entry were analyzed. The stability and cementation status of surrounding rock in the 360,803 airway were evaluated, and tunneling safety was ensured.

(8) Zhang, G. et al., Distribution Law of In Situ Stress and Its Engineering Application in Rock Burst Control in Juye Mining Area [8].

This paper presents an integrated approach for mathematical statistics, theoretical analysis, and a field test to investigate the distribution law of in situ stress and its engineering practice of rock burst control. The test site is located in the Juye mining area, Shandong Province, China. The main conclusions included the following: (1) There are two types of in situ stress states in the Juye mining area, $\sigma_H > \sigma_V > \sigma_h$ (42.42%) and $\sigma_H > \sigma_h > \sigma_V$ (57.57%), which are mainly caused by the tectonic stress of the Heze and Fushan faults (σ_H, $\sigma_{V,}$ *and* σ_h are the maximum principal stress, vertical principal stress or intermediate principal stress, and minimum principal stress, respectively). (2) The lateral pressure coefficients K_H, K_h, and K_{av} show a non-linear distribution with increased depth, approaching 1.32, 0.96, and 1.41, respectively. The variation range of the horizontal difference stress μ_d is 0.09 to 0.58. (3) The average value of the stress gradient is 3.05 MPa/100 m, and the main directions of the maximum horizontal principal stress are northeast–southwest and northwest–southeast. (4) A new combined supporting strategy, incorporating optimization of roadway layout, anti-impact support system design, and local reasonable pressure relief, was proposed for the rock burst control, and its validity was verified via field monitoring. All these design principles and support strategies for the rock burst control presented in this study can potentially be applied to other similar projects.

(9) Rajwa, S. et al., Numerical Simulation of the Impact of Unmined Longwall Panel on the Working Stability of a Longwall Using UDEC 2D—A Case Study [9].

The main goal of the paper is numerical simulation for investigating causes of damage in the working of a longwall located under the unmined longwall panel. The paper presents the results of model-based research on the stability of the roof of a longwall working in a zone subject to cave-in mining, taking into account the influence of mining conditions in the form of an unmined coal seam located 115 m above the exploited seam. It presents the geometry of the rock mass under study and the discretization area of the solution and gives an overview of the assumptions used to build the numerical model. The authors discuss the results of numerical simulations of the influence of mining phenomena on the formation of roof falls in the longwall. Based on the results of numerical simulations, the process of identifying the size of roof falls in a longwall working (loss of stability) was carried out through their appropriate classification. The case presented and analyzed in this paper occurred in one of Poland's coal mines.

(10) Li, L. et al., Study on Overburden Movement and Fissure Evolution Law of Protective Layer Mining in Shallow Coal Seam [10].

This study aims to effectively solve the problem of the destruction of the coal roof and floor overlying rock after mining the protective layer and determines whether the gas in the protected layer can be effectively released. We do this based on the engineering background of the Weng'an Coal Mine; research and analysis of the movement of the roof and floor overlying rocks; the evolution of cracks and the pressure relief characteristics of the protected layer after mining the protective layer; and through theoretical analysis and similar simulation experiments. Through numerical simulation, it was found that the protected layer was depressurized due to the mining of the protective layer, and the decompression rate of the protected layer was 0.2 to 0.8. In addition, the overall expansion rate of the protected layer was greater than the requirements of the "Detailed Rules for Prevention and Control of Coal and Gas Outbursts" for coal mines, and the investigation of the residual gas pressure and content of the protected layer revealed that the protected coal seam had been mined in the upper protective layer coal face. The gas pressure dropped to 50.7% of the original coal seam gas pressure, the rate of decrease was 49.3%, the residual gas content dropped by 68.67%, and the gas concentration in the return airway was 0.31% on average, meeting the national regulations that require its value to be less than or equal to 1%. The study comprehensively demonstrates that mining the protective layer is beneficial to the release of gas from the protected layer, and provides a practical reference for coal and gas outbursts in mines.

(11) Zhang, J. et al., Experimental Investigation of Failure Mechanisms of Granites with Prefabricated Cracks Induced by Cyclic-Impact Disturbances [11].

This work investigated rock deformation and failure characteristics through cyclic impact tests on granite samples with cracks of different angles. A Hopkinson bar was employed for uniaxial cyclic impact tests on granite samples with the crack inclination angles of 0 to 90°. The magnetic resonance imaging technique was used to determine rocks' porosity after cyclic impacts. The stress–strain curves, porosity, strength, deformation modulus, failure modes, and energy density of samples were obtained and discussed. Results showed that the crack inclination angles significantly affected the damage evolution and crack morphology of rocks. Under the constant cyclic impact, the dynamic deformation modulus and dynamic strength of rock samples first increased and then decreased with the increase in crack inclination angle. The failures of granite samples for inclination angles of 0 and 90° were dominated by tensile cracking, while those for the inclination angles of 30–60° were dominated by shear cracking. The energy density per unit time gradually decreased with the increase in impact cycles. The results can provide references for the stability analysis and cyclic-impact-induced failure prediction of fractured rock masses.

(12) Chen, J. et al., Studying the Bond Performance of Full-Grouting Rock Bolts Based on the Variable Controlling Method [12].

This paper studied the bond performance of full-grouting rock bolts with a theoretical analysis. The variable controlling method was used to study the effect of parameters on the load-carrying force of bars. The results showed that when the bar diameter grew from 15 mm to 25 mm, the maximum force of the bars rose from 194 kN to 349 kN, growing by 80%. As for the stiffness, it grew by 108%. Moreover, when the elastic modulus grew from 50 GPa to 200 GPa, the maximum force rose from 229 kN to 269 kN, only growing by 17%. As for the stiffness, it grew by 100%. When the grouting length increased from 2 m to 3 m, the maximum force rose from 269 kN to 364 kN, growing by 35%. However, the grouting length had almost no effect on the stiffness. Lastly, for the bond slip when the bond strength was reached, when it grew from 1 mm to 3 mm, the maximum force dropped from 281 kN to 258 kN, dropping by 8%. As for the stiffness, it dropped by 44%. This paper is conducive to enriching the base of knowledge.

(13) Chen, Y. et al., Rockburst Precursors and the Dynamic Failure Mechanism of the Deep Tunnel: A Review [13].

This paper presents a review of the current understanding of rockburst precursors and the dynamic failure mechanism of the deep tunnel. Emphasis is placed on the stability of the surrounding rock of the deep tunnel, the rockburst prediction method, and the dynamic failure characteristics of the surrounding rock of the deep tunnel. Throughout the presentation, the current overall gaps in understanding rockburst precursors and the dynamic failure mechanism of deep tunnels are identified in an attempt to stimulate further research in these promising directions by the research community.

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