Research Article

Study on Complex Theory Solution and Numerical Simulation of Fracture Mechanics of Surrounding Rock Stress and Energy Field in Fault Type Rock Burst Stope

Haidong Zhang,1,2,3 Siyu Dou,2 Guangchen Zhao,2 Haiqing Yang,2 and Chunyan Wu2

1School of Mines, China University of Mining and Technology, Xuzhou 221116, China
2Department of Civil and Construction Engineering, Shanxi Institute of Technology, Shanxi 045000, China
3University of Wollongong, NSW 2522, Australia

Correspondence should be addressed to Haidong Zhang; haidongzhang221116@163.com

Received 12 July 2022; Accepted 23 July 2022; Published 13 September 2022

Academic Editor: Depeng Ma

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Rock burst is one of the most sophisticated and threatening problems in the process of mining activities of underground coal mines, which can evolve into phenomena such as gas outburst, violent ground pressure behavior of surrounding rock of stope roadway, and high dynamic load energy event of roof overburden fracture through the interaction between various influencing factors. This paper deeply studies the causes, consequences, and disaster mechanism of “2–2 rock burst accident” in Xinjulong coal mine, gives the network diagram of the relationship between the main influencing factors of fault type rock burst and high-energy events, establishes a mechanical model on this basis, and explores the influence and evolution law of fault influencing factors on the elastic strain energy field of surrounding rock of stope roadway.

1. Background Introduction

On February 22, 2020, Xinjulong coal mine had a large rock burst accident in the upper roadway of 2305 s working face in the south wing of −810 m horizontal second mining area, resulting in four deaths and huge economic losses. Due to the jumping impact energy characteristics caused by its unique fault distribution at the mining area boundary, this case has become a research hot topic of experts and scholars at home and abroad in the field of rock burst for several years. At the same time, due to the complexity of its disaster causing mechanism and the overall systematicness of the stope, the factors that directly lead to the violent impact can only be delineated in the following three aspects: 1) fault stress concentration; 2) hard composite sandstone roof accumulating a large amount of elastic strain energy; 3) regional tectonic stress being affected by mining disturbance. This is a disadvantageous factor for the research on the means of impact prevention and control. Thanks to the detailed data support of Xinjulong coal mine of Shandong energy group, this paper only studies the formation, accumulation, and appearance process of impact energy for the interruption layer of this case and takes this as the basis to provide a theoretical basis for the impact prevention and control of mines under similar conditions.

2. Introduction

In the mining industry, transportation industry, underground space engineering construction, water conservancy, and hydropower engineering and other underground rock mechanics engineering fields, more and more large section roadways have been excavated and serve large-scale deep rock mechanics engineering in recent years [1]. With the average buried depth moving towards the geocenter at the speed of nearly 10 m/A, the surrounding rock state of deep roadway is subject to more complex mechanical boundary conditions, and a series of new dynamic load failure and...
instability phenomena also appear [2]. In 1933, the phenomenon of strong dynamic pressure in rock burst mine, which occurred in Fushun Shengli coal mine for the first time in China and was officially statistically defined in 1959, also appeared unprecedented “jumping” disaster causing characteristics after the mining entered the deep stage, which undoubtedly launched a more arduous task for rock burst research experts and scholars, and even the emergence of this case challenged some previous research results on the starting and appearing mechanism of rock burst. Some theories that only apply to continuous coal seam impact have been questioned. Therefore, the in-depth study of Xinjulong 2–2 rock burst case is of great significance even after the stoppage of the mining area.

Rock burst is a violent and instantaneous throwing out of coal and rock mass in the process of deep coal seam mining. When the release energy is serious, it may cause the damage of roof and floor and even destroy the roadway, resulting in casualties and equipment damage [3].

At present, in the initiation and accumulation mechanism of rock burst, the theories that have a detailed classification and definition of the influencing factors of fault structure include dividing the rock burst phenomenon from the stope abutment pressure acting on the working face and near the stope roadway into “gravity type” rock burst. The rock burst events of the principle stope appearing in the advance position of the driving roadway and the upper and lower drift are divided into “structural” rock burst [4]. Then, the structural rock burst is divided into broken line type (rock burst in the area near the fault line, which is easy to burst when the roadway is under the influence of this stress due to the residual in situ stress in the fault zone) and section type (it occurs in the section within a certain range from the fault structure and is subject to the combination and superposition of mining stress and in situ stress). Tai et al. divided the rock burst induced by mining activities into coal compressive stress type, top and bottom plate tensile stress type, and fault strike slip shear type according to the force source [5]. The fault strike slip shear type rock burst takes the energy released by the sudden dislocation between bedding and fault discontinuity as the source of input energy of rock burst. In 2003, Pan further divided rock burst into coal compression rock burst, roof fracture rock burst, and fault dislocation rock burst and deeply expounded its mechanism: it is considered that the process of fault dislocation rock burst is due to the propagation, superposition, and amplification of mining stress in the form of energy through the rock material medium between faults when mining activities are close to the fault. Thus, the fault is suddenly staggered and damaged, with the magnitude often reaching 3–4 and the dominant frequency reaching 1–6 Hz [6]. Fault type rock burst is often not due to the impact damage of the energy of fault dislocation directly caused by natural tectonic stress on the stope roadway, but due to the disturbance stress caused by mining activities, it breaks the rock structure of the fault plane that has accumulated higher than the original rock stress level, resulting in the dislocation damage and energy release of the fault structure position, and then transfers the energy to the roadway surrounding rock, coal pillar, or stope coal wall. It leads to the sudden destruction of the bearing structure of the surrounding rock of the stope which is already bearing the peak stress, the release of energy to the free surface, and the violent rock burst accident.

In view of the prediction and early warning of the precursor of fault tectonic rock burst in coal mine, a microseismic monitoring station was established for the first time in the deep mine of Johannesburg, South Africa, at the early stage of Alhilton seismological society to monitor the microseismic activity of the mine [7]. More than 200 microseismic energy events caused by mining disturbance have been successfully monitored. Simple capture and analysis of microseismic event frequency and energy are difficult to combine its characterization characteristics with impact prediction. Therefore, B. Brady and others further studied microseismic energy events and roof collapse caused by mine mining and matched them from the characterization level [8]. Since entering the 21st century, Professor Dou et al.’s team [9] of China University of mining and technology has integrated monitoring equipment such as Aramis, ESG microseismic monitoring system, and electromagnetic radiometer into the rock burst microseismic monitoring system and used big data to build a multiparameter rock burst early warning cloud platform to achieve accurate and quantitative prediction of coal mine impact.

In addition, as per Pan et al. [10, 11], the roadway anti-impact support equipment is deeply explored according to the impact energy absorption mechanism. Therefore, after the in-depth study of the mechanism of fault type rock burst and the establishment of quantitative discrimination formula, it is expected to realize the safe mining of resources with earthquake and without disaster under the guarantee of the above microseismic monitoring technology and anti-scour equipment.

In this paper, on the one aspect, the mechanical modeling of the fault round the roadway surrounding rock in the case is carried out, and the plate construction of elastic mechanics is adopted for analysis. In the assumption of boundary conditions and the selection of constitutive relationship, reference is made to the classical theories of fracture mechanics, material mechanics, and rock mechanics. In order to obtain the theoretical calculation formula of the stress field around the infinite crack plate, preestimate the stress concentration position and energy accumulation area of the surrounding rock of the stope in the case and similar boundary conditions and realize the formulation of quantitative measures for the scour prevention of the stope working face and roadway on this basis. Finally, the accuracy of the theoretical formula is verified by the simulation experimental results of FLAC3D numerical simulation software and field measured data records.
3. Rock Burst Occurrence Process and Geological Details of Typical Fault Type Rock Burst

The accident occurred at 6:17 a.m. in the upper drift of −810 n level 2305 s fully mechanized top coal caving face. The average thickness of the main mining 3# coal seam is about 8.82 m, and its elastic energy index wet is calibrated as 1.2 by laboratory rock mechanics experiment. It belongs to the coal seam with medium risk of impact tendency. There are 6 faults in the working face with impact, of which FD8 and FD11 have a great impact on the mining operation. The FD8 fault penetrating the impact appearance roadway and connecting roadway is selected for in-depth study. The fault location and the damage degree of the working face roadway by impact are shown in Figure 1.

As shown in Figure 1, it is shown that FD8 fault passes through the intersection of 2305s upper drift and associated roadway, and the damage degree is the most serious. During the roadway excavation, no obvious features of the upper and lower drift and the surrounding plastic area of the combined roadway are found to be affected by the fault. However, after the accident site investigation, it is found that the roadway near the FD8 fault is the most severely damaged, the two sides move relatively close, the roof bolts, anchor cables, and roof beams break off, and only 1.0 m² of the minimum roadway section is left.

Therefore, modeling and analysis are mainly carried out for the relative spatial position of FD8 fault and 2305 s working face stope and service roadway and the geological conditions of rock mechanics.

The coal seam passes through the fault structure, and the horizons move and slip with each other. Affected by the regional tectonic stress of the plate, the coal seam reaches a stable state after long-term movement. Before the influence of artificial mining activities, it is difficult to form a strong source of elastic strain dynamic force by itself. Generally, it is only shown that the regional stress rise exceeds the original rock stress. When the mining disturbance is near the fault, the leading stress caused by mining and the regional elevated stress caused by tectonic stress are mutually superimposed, resulting in the breaking of the original stable state between faults, as shown in Figure 2. The movement law between rock layers shows that the fault rock layer near the mining free surface may sink, slide, rotate, or composite displacement. The support of the rock layer at the fault plane where the original rock is squeezed and engaged is changed from surface load to point load, resulting in the stress concentration of the rock layer at the fault plane. As a result, a large amount of elastic strain energy is accumulated in the rock layers on both sides, which is more dangerous. The trend surface of elastic strain energy release tends to the working face near the horizontal angle and the free end of the roadway.

In fracture mechanics, Volterra’s [1] dislocation model [12, 13], and dislocation theory [14] and for the calculation of the normal stress field of the infinite crack surface, the mechanical model of the influence factors such as cohesion is introduced.

Among them, the Volterra dislocation model assumes that the relative position staggering amount between the two sides of the fault is equal everywhere, which is a constant, and the displacement discontinuity value can be easily obtained by taking the average value in the drill hole.
inspection map. However, the Volterra dislocation model has its major defect that when a specific value is obtained in the X direction, its shear stress $\tau_{xy}$ is greater than another certain value, and some singular values will appear in the calculation results of the stress field and displacement field on the crack surface, which will seriously affect the confidence of the calculation results.

The other two fault mechanics models are not suitable for calculating the displacement around the crack on the infinite plane because of their mechanical assumptions. Therefore, the linear elastic fracture mechanics model is introduced [15].

The final result of the calculation of the complex stress function in the complex variable function is

$$
\begin{align*}
\tau_{xy} &= \text{Im}[Z\phi''_1(z) + \psi'_1(z)], \\
\sigma_x &= 2\text{Re}[\phi'_1(z)] - \text{Re}[Z\phi''_1(z) + \psi'_1(z)], \\
\sigma_y &= 2\text{Re}[\phi'_1(z)] + \text{Re}[Z\phi''_1(z) + \psi'_1(z)], \\
\psi'_1(z) &= \frac{1}{2\pi(1+\kappa)} \sum_{k=1}^{m} (x_k + iY_k)\ln (z - z_k) + \phi_{1s}(z),
\end{align*}
$$

where $\phi_{1s}(z)$ is the coordinate variable for calculating Westergaard function.

Assume that the total stress component around the fault is $\sigma_{XX}, \sigma_{YY}, \tau_{XY}$, disturbance stress field of fault is $\sigma^{c\infty}$ ($i = 1, 2$) [16], and stress function of direct conversion is as follows [16]:

$$
Z^{c}_1(z) = \left( \frac{z}{\sqrt{z^2 - a^2}} - 1 \right) \Delta \tau.
$$

The disturbed stress component around the fault is

$$
\begin{align*}
\text{Im}Z^{c}_1 &= \left( \frac{\sigma^{c\infty}_{xx} - \sigma^{c\infty}_{yy}}{2} \right), \\
\text{Im}Z^{c}_1 + y\text{Re}Z^{c}_1 &= \left( \frac{\sigma^{c\infty}_{xx} - \sigma^{c\infty}_{yy}}{2} \right), \\
\text{Re}Z^{c}_1 - y\text{Im}Z^{c}_1 &= \tau^{c}_{xy},
\end{align*}
$$

where $\text{Im}Z^{c}_1$ is the real part of the complex variable function of the disturbed stress component; $\text{Re}Z^{c}_1$ is the imaginary part of the complex variable function of the disturbed stress component; $Z^{c}_1$ is called the Westergaard function of mode II crack, which can be taken as

$$
Z^{c}_1(z) = \frac{zt^{\infty}}{\sqrt{z^2 - a^2}} + D,
$$

where $C = D = t^{\infty}$ take the boundary condition at infinity; $A$ selects the center position for calculation; $Z$ is the coordinate variable for calculating Westergaard function. Then, the displacement component around the fault is obtained as

$$
\begin{align*}
2\mu' &= \left( \frac{\kappa + 1}{2} \right) \text{Im}Z^{c}_1 + y\text{Re}Z^{c}_1, \\
2\nu' &= -\left( \frac{\kappa + 1}{2} \right) \text{Re}Z^{c}_1 - y\text{Im}Z^{c}_1.
\end{align*}
$$

The full field formula of mode II crack stress is

$$
\begin{align*}
\sigma_{xx} &= 2\text{Im}Z^{c}_1 + y\text{Re}Z^{c}_1, \\
\sigma_{yy} &= -y\text{Re}Z^{c}_1, \\
\tau_{xy} &= \text{Re}Z^{c}_1 - y\text{Im}Z^{c}_1 - C.
\end{align*}
$$

Figure 2: Suspension curve of fault stress and mining stress.
By substituting formulas (5) into (6) of elastic strain energy [17, 18], the calculation formula of elastic strain energy field (scalar) around the crack can be obtained:

\[
U^e = \frac{1}{2E} \left[ (\Delta \sigma_x + \sigma_H) + (\Delta \sigma_y + \sigma_v) \cos \alpha \right] \sin \theta + \left[ (\sigma_h + \Delta \tau_{xy}) + (\Delta \sigma_y + \sigma_v) \sin \alpha \right] \cos \theta
\]

\[
= \frac{1}{2E} \left[ (\Delta \sigma_x + \sigma_H) + (\Delta \sigma_y + \sigma_v) \cos \alpha \sin \theta + (\sigma_h + \Delta \tau_{xy}) + (\Delta \sigma_y + \sigma_v) \sin \alpha \cos \theta \right]
\]

\[
+ \Delta r_{xy} \left( \sigma_h + \Delta \tau_{xy} + (\Delta \sigma_y + \sigma_v) \sin \alpha \right) \sin \theta
\]

\[
+ \Delta r_{xy} \left( \sigma_h + \Delta \tau_{xy} + (\Delta \sigma_y + \sigma_v) \cos \alpha \right) \cos \theta
\]

(7)

The disturbed stress contour of linear elastic fault stress model is calculated, as shown in Figure 3.

It can be clearly seen that around the type II fracture, the maximum disturbed compressive stress around the fault initiation direction reaches 1.04 MPa, and the maximum disturbed tensile stress in the middle of the fault reaches 23.7 MPa. This shows that the stress state of the rock stratum near the fault structure is sophisticated, there is a sudden change in the stress direction, and the stress difference is obvious. This stress state of the surrounding rock of the fault makes it easier for the surrounding rock in the dominant stress direction to displace or transfer elastic strain energy to the surrounding rock in the weak stress direction under the influence of the external mining stress. As a result, the disturbance caused by mining causes further damage when the stress difference near the fault structure exceeds the strength limit of the surrounding rock under the stress state at that time. There is a 15.1 m thick hard rock near the FD8 fault with a length of about 240 m, accumulating a large amount of elastic strain energy. When it is subjected to a tensile stress of 23.7 MPa under its own confining pressure, it will be damaged, and the released energy will exceed 0.4393 kJ. It is far beyond the upper limit threshold of one-day total energy involved in micro earthquake shutdown warning of the coal mine by 0.27 kJ.

Therefore, through the analytical calculation of rock fracture mechanics theory, it is not difficult to find that the energy accumulated in the surrounding rock of the fault is enough to cause the occurrence of rock burst. In addition, through the analysis of the direction and contour map of the disturbed stress field around the fault fracture in Figure 3 obtained from the calculation results, it can be concluded that for the stope roadway and working face threatened by fault type rock burst, through large-diameter pressure relief drilling and hydraulic fracturing, the pressure relief and erosion prevention of high stress surrounding rock by means of deep space blasting and surface fracturing are not only required to create a certain range of plastic zone of fault surrounding rock, but also important to reasonably regulate the stress field of fault surrounding rock according to the stress direction and dominant weak stress relationship of fault surrounding rock obtained from theoretical calculation and field measurement, so as to reduce the stress around the fault and reduce the stress difference at the same time.

4. Numerical Simulation Experiment of Sudden Change Model of Surrounding Rock Stress Disturbed by Fault in Rock Burst Stope

According to the field survey and measured borehole histogram, the physical and mechanical properties of the top and bottom slate of 3# coal seam near FD8 fault in 2305S working face are obtained, and the column chart is shown in Figure 4. It can be seen that the direct roof of 1.5 m fine sandstone with moderate impact tendency and the basic roof of 15.45 m thick hard medium grained sandstone are unfavorable factors for the risk of rock burst in the stope and roadway of 2305 s working face. Firstly, the thick and hard basic roof of medium grained sandstone accumulates a large amount of elastic strain energy under the action of high in situ stress with −980 m buried depth, and because the direct roof is thin and close to the basic roof, it is difficult to fully protect the stope, so the dynamic pressure appears obviously in the process of fracture and instability of the basic roof. In addition, the 3# coal seam has a moderate impact tendency, and a large amount of elastic strain energy is bound to
accumulate in the surrounding rock energy field of the stope. The following modeling and analysis are carried out for 2305 s working face in combination with geological conditions.

Flac3D 6.0 numerical simulation software is used to model the stope roadway, working face overburden strata, and baseplate of 2305 s working face. At the same time, FD8 fault is divided and constructed in the rock stratum. The spatial geometric position relationship is shown in Figure 5.

Based on the actual mining conditions of 2305 s working face, the experimental scheme and parameter selection mainly discuss the stage far away from the fault during working face and roadway excavation (stage I); approaching fault stage (stage II); the stress field distribution and the accumulation of elastic strain energy field in the overlying strata and floor of the stope during the four stages of passing through the fault (stage III) and leaving the fault (stage IV) to verify the accuracy of the energy field distribution in the calculation results of the elastic strain energy analytical formula around the fault derived above.

It can be seen in Figure 6 that the development shape of the stope collapse zone is similar in the mining influence stages I, II, and III. When passing through the fault, the height of the roof collapse zone is reduced from 15.4 m to 12.65 m due to the increase of the horizontal tectonic compressive stress and the decrease of the tensile stress in the surrounding rock stress area of the fault. During the period when the mining space crosses the fault (stages II and III), the stress of the surrounding rock of the roadway increases significantly and the stress of the overlying strata of the stope decreases to a certain extent, which is good news for the working condition of the support in the mining process of the working face. However, for the surrounding rock of the roadway, the existence of the fault further increases the stress of the surrounding rock of the roadway and increases the risk of rock burst accidents in the roadway.

In addition, by comparing the calculation results of the above rock fracture mechanics on the disturbed stress field around the fault in phase IV of Figure 5, it can be seen that the isoline distribution law of the stress state around the fault basically conforms to the conclusion of numerical simulation.

The displacement and stress data of the node element are postprocessed by Tecplot2021 software, and the stress tensor is calculated according to formula (6). The result is the elastic strain energy UE of each node. As shown in Figure 7, the concentration and accumulation position of elastic strain energy changes with the advance of the fault crossing stage.
At stage I, the elastic strain energy is mainly accumulated in the coal seams, and the concentration degree is very weak. There is little difference between the elastic strain energy accumulated in the fine sandstone of the roof and siltstone of the floor, and the elastic strain energy of the node is less than 0.02 kJ as a whole. When the roadway near the fault reaches stage II, the bending moment of the roof increases, and the accumulated elastic strain energy in the fine sandstone layer increases significantly. Due to the influence of mining liberation, the accumulated elastic strain energy in the coal seam decreases. However, when the stope roadway and working face pass through the fault to reach stage III, a large amount of elastic strain energy accumulation occurs in the roof and floor strata at the same time. At this time, the elastic strain of the node element as a whole is close to 0.06 kJ, which is about three times higher due to the influence of the fault.

Therefore, it is not difficult to see that the presence of faults has a negative and dangerous impact on coal mining when theoretically explaining the mechanism of the initiation of rock burst caused by faults from the perspectives of stress source and elastic strain energy [19]. In order to reduce the threat of faults to a certain extent and the risk of rock burst in mines with faults around or inside the mining area [20], the antiscour process parameters shall be quantitatively formulated in combination with the analytical formula (7), and the relevant influence parameters of fault stress energy field are revealed by the numerical simulation test results.

5. Discussion and Conclusions

Based on the established mechanical model, combined with the calculation formula of stress field and energy field around type II structural fault derived from rock fracture
mechanics and material mechanics, and the numerical simulation experiment of FD8 fault model, the influence of strata fault structure on the impact risk of stope surrounding rock from the perspective of force source and energy is obtained when the stope experiences different mining stages and spatial geometric position relationship of the fault.

In practice, the conclusion of this paper is as follows.

(1) Through the calculation formula of stress field and energy field around type II structural fault, the state of surrounding rock in different stages of mining is preliminarily calculated and estimated. It is a new effective auxiliary means to predict and estimate the impact tendency of mine threatened by fault type rock burst [21]. It can calculate the accumulation degree coefficient of elastic strain energy \( U_e \), the high-energy position of the roof and floor of the surrounding rock, and the high-stress and high-energy horizon that should be paid attention to in different mining stages.

(2) The parameters in the analytical formula explain that, in the prevention and control means and measures of type II structural fault type rock burst [22], while considering the pressure relief construction of high stress concentration in thick and hard basic roof strata, attention should also be paid to the phenomenon of high stress and high energy in the floor. The surrounding rock of the stope is an integral support and bearing system. When the floor stress is at a high level, the pressure relief of the basic roof strata may not be effective.

(3) During stage II and stage III, attention should be paid to reducing the advancing speed of the working face, using large-diameter pressure relief drilling and hydraulic fracturing to cut the hard basic roof strata to shorten the fracture layout and basic roof thickness. In addition, reducing the mining height to protect the floor and opening pressure relief grooves for the roadway floor are also theoretically feasible and effective on-site impact prevention measures [23]. What is more noteworthy is that, in the process of mining thick coal seams, the staggered roadway layout method [24] can be selected, and the roadway can be arranged along the coal seam roof [25]. Under the dual protection of the thin immediate roof protection and the roadway floor coal seam, the roadway surrounding rock can be slightly vibrated without obvious impact [26, 27].

**Data Availability**

The experimental and analytical calculation results data used to support the findings of 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.

**Acknowledgments**

This research project was funded by the Science and Technology Project of Shanxi Institute of Engineering and Technology (2022HX-05), the National College Students’ Innovation and Entrepreneurship Training Project (s202214527014), and Ecological Restoration of Mining Areas and Solid Waste Recycling Provincial and Municipal Joint Construction of Shanxi Provincial Key Laboratory Cultivation Base Project. This dissertation was funded by the Shanxi Province Overseas Students Management Committee.

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