Experimental and numerical investigation of energy dissipation of roadways with thick soft roofs in underground coal mines

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Abstract
The roadway deformation process is closely associated with the accumulation and dissipation of energy. Understanding the characteristics of energy dissipation in surrounding rock is critical to revealing the development process and the disaster mechanisms of the roof accidents in underground coal mines. This paper presents a triaxial compression experiment to analyze energy characteristics of mudstone taken from Huangyanhui Coal Mine. A numerical study is performed to study the energy distribution under different roof strengths. The results suggest that the instability of the roadway is an irreversible energy dissipation process with high nonlinearity and complexity. The roadway support principle, based on the energy balance theory, is proposed to explain the energy distribution. The safety control of roadways with thick and soft roofs should consider the following three aspects: (a) optimize the layout of the roadway, avoid placing it in the stress concentration zone, and reduce the strain energy of the surrounding rock accumulation from the source; (b) improve the ability of the supporting structure to adapt to the deformation of the surrounding rock and avoid the failure of the supporting structure because of excessive force; and (c) set weak structures in the surrounding rock, that dissipate part of the energy, thus reducing the load acting on the support. The results of field application monitoring show that roadway severe deformation was effectively controlled using pressure relief boreholes (PRB), and the average floor heave speed of the roadway decreased from the original 2.0 to 0.98 mm/d. Therefore, PRB provides a new approach for roadway control.

Keywords
energy balance, energy dissipation, pressure relief, roadway, thick soft roof
1 | INTRODUCTION

The immediate roof of coal seams in China is generally carbonaceous mudstone, sandy mudstone, and carbonaceous shale, and the thickness of the soft strata is usually more than 8 m.1 Roadways with extra thick soft roofs are difficult to support and are prone to roof accidents owing to the long-lasting roof convergence and wide range of the broken rock zone during mining period.

Research on the instability mechanism and control technology of roadways with thick soft roof, mainly focused on the evolution law of surrounding rock fracture,2,3 roof separation mechanism,4,5 and support technology.6-8 Although much effort has been made in studies of the mechanism, prediction, and prevention of roadways with thick soft roofs, our understanding is still far from mature. Theoretical and experimental studies confirm that energy plays an important role in rock deformation and failure. Dissipated energy from external forces produces damage and irreversible deformation within rock and decreases rock strength over time.9 Therefore, the study of the characteristics of energy dissipation in the surrounding rock is important to reveal the formation conditions, development process, and disaster mechanism of roof accidents in underground coal mines.

Salamon10 proposed that the failure of rock mass is assumed to be a process of strain energy accumulation, dissipation and release inside the rock. Researchers including Wang and Park,11 Jiang et al,12 Mansurov,13 Zhang,14 Kidybinski,15 Yang et al,16 and Winton17 conducted extensive works to predict rockburst using energy analysis. Gao et al18 proposed the design method of supporting parameters for roadway with rock burst risk based on the principle of energy balance. It is believed that the support system should be able to absorb the impact of kinetic energy generated in the process of rock burst in time. Numerical investigation associated with strain energy density is conducted to simulate the energy accumulation and dissipation characteristics of the roadway failure process in Linglong gold mine.19 Ma et al20 discussed the cross-section shape optimization of deep roadways from the perspective of energy release and provided a reference for the design of the deep high-stress roadway. Based on the theory of energy dissipation, Zhang et al21 analyzed the energy characteristics of roadway lateral surrounding rock and revealed the mechanism of pressure relief in a staggered arrangement roadway. He22 developed an energy absorber with a constant resistance by using a negative Poisson’s ratio material to solve the control problem of large deformation roadways. Dai et al23 designed an axial splitting metal circular tube for anchor bolts and successfully enhanced its load level and energy absorption efficiency. In addition, Roofex bolt,24 D bolt,25 and Yield-Lok bolt26 were also designed to adapt to a large deformation of the surrounding rock in deep roadways. In the aforementioned examples, each type of bolts maintains the stability of the roadway by absorbing energy. However, there is no systematic support principle based on energy balance theory for roadways with thick soft roofs.

In this study, we investigate the energy dissipation of roadways with thick soft roofs in underground coal mines. A triaxial compression experiment was conducted to analyze energy characteristics of mudstone taken from Huangyanhui Coal Mine, and a numerical study was performed to study the energy distribution with different roof strengths. The roadway support principle based on the energy balance theory is also proposed. The research results provide a reference for the safety control of roadways with thick soft roofs.

2 | ENERGY DISSIPATION CHARACTERISTICS OF MUDSTONE

2.1 | Geological condition

Huangyanhui Coal Mine is located in Qinshui coalfield in the Shanxi Province, China. Figure 1 presents the general layout of the mine. The main coal seam of Huangyanhui Coal Mine is #15 coal seam with a 4-18° dip and thickness of 4.3-6.3 m. The immediate roof of #15 coal is formed by black mudstone and sandy mudstone, and the thickness of the immediate roof ranges from 10 to 12.5 m (Figure 2).

The support system of the tailgate of panel 15 107 consisted of anchor cables, bolts, steel belt, and steel mesh during excavation (Figure 3). The anchor cable beams were constructed on the roadway rib during the mining period to ensure the stability of the roadway. Field monitoring showed that the roadway roof to floor convergence was more than 1.0 m during the mining period. The roadway should be repaired in the 20 m section ahead of the working face to ensure the normal use of the roadway.

2.2 | Energy dissipation characteristics of mudstone

The sand mudstone taken from the 15 111 highly located drainage roadway of Huangyanhui Coal Mine was compressed under different confining pressures by using the TAW-2000 electro-hydraulic servo rock mechanics test system, and the energy dissipation characteristics of the samples during the failure process were analyzed.

Taking a unit volume coal unit as an example, it is assumed that the test is a closed system without heat exchange with the outside world. In the test process, the external work done on the sample is equal to the change of internal energy of the samples \((U)\), which can be calculated as

\[
U = U^d + U^e, \tag{1}
\]
where \( U_d \) is the dissipative strain energy, used to form internal damage and plastic deformation of the sample, and \( U_e \) is the elastic strain energy.

Under the uniaxial compression test, there is only axial stress and strain, and \( U \) can be obtained using the following formula:

\[
U = \int \sigma_1 d\varepsilon_1 = \sum_{i=1}^{n} \frac{1}{2} \left( \sigma_{1i} + \sigma_{1i-1} \right) \left( \varepsilon_{1i} - \varepsilon_{1i-1} \right) \ , \quad (2)
\]

where \( \sigma_1 \) and \( \varepsilon_1 \) are the axial stress and axial strain, respectively; and \( \sigma_{1i}, \varepsilon_{1i} \) are the axial stress and axial strain at each point on the axial stress-axial strain curve.

The unloading elastic modulus can be replaced by the elastic modulus of the prepeak elastic section, so the elastic strain energy at any point on the stress-strain curve is calculated with the following formula (Xie et al, 2009):

\[
U_e = \frac{\sigma_{1i}^2}{2E} \ . \quad (3)
\]

Under the condition of triaxial compression, there are both axial stress (\( \sigma_1 \)) and hoop stress (\( \sigma_2 = \sigma_3 \)), and the energy actually absorbed by the samples is calculated according to formula (4):

\[
U = \int \sigma_1 d\varepsilon_1 + 2 \int \sigma_3 d\varepsilon_3
= \sum_{i=1}^{n} \frac{1}{2} \left( \sigma_{1i} + \sigma_{1i-1} \right) \left( \varepsilon_{1i} - \varepsilon_{1i-1} \right) - 2 \sum_{i=1}^{n} \sigma_3 \left( \varepsilon_{3i} - \varepsilon_{3i-1} \right) \ . \quad (4)
\]

The elastic strain energy (\( U_e \)) includes two parts: axial strain energy and annular strain energy. According to relevant studies, the annular elastic strain energy is small and negligible compared with axial strain energy, so the elastic strain energy (\( U_e \)) is still calculated according to Equation (3).

Figure 4 shows the energy evolution curve of mudstone samples. It can be seen from the figure that there are similar characteristics of each energy characteristic curve, and the energy evolution process can be divided into two stages, with peak stress as the demarcation point.

Stage I corresponds to the part of the stress-strain curve before the peak stress, and most of the energy absorbed by the samples is converted into elastic strain energy, accounting for 77.2%-87.9% of the total absorbed energy. Confining pressure and accumulated elastic strain energy of the samples are directly related to elastic strain energy being approximately linear with the confining pressure, as shown in Figure 5. When the confining pressure was 3 MPa, the elastic
strain energy was 118.37 kJ/m^3, and when the confining pressure was 7 MPa, the elastic strain energy was 249.93 kJ/m^3, which was an increase of 111.1%. The elastic strain energy is stored in the deformation of the sample skeleton, and the strain energy is dissipated in the friction of the closed fracture and the initiation and expansion of the microcracks.

Stage II corresponds to the postpeak phase of the stress-strain curve, as shown in Figure 4. At this stage, the dissipative strain energy assumes a dominant position. After reaching the peak strength, a substantial amount of the elastic strain energy is released and transformed into dissipative strain energy, which is dissipated in the shear slip of
the macrofracture. In this stage, the dissipative strain energy accounted for a maximum of 96.7% of the total dissipative strain energy during the test.

The increment of strain energy per unit time is the strain energy conversion rate, and the magnitude of the conversion rate of dissipative strain energy reflects the rate of deformation damage of the samples. The dissipative strain energy conversion rate of mudstone samples under different confining pressures is shown in Figure 6, which illustrates that higher confining pressure corresponds to a higher dissipative strain energy conversion rate, especially for the maximum strain energy conversion rate. When the confining pressure ($\sigma_3$) is 0 MPa, the maximum strain energy conversion rate is 20.83 MJ/m$^3$/h. When the confining pressure was 3 MPa, then the maximum strain energy conversion rate was 62.39 MJ/m$^3$/h. When the confining pressure was 5 and 7 MPa, the maximum strain energy conversion rate was 308.01 and 461.74 MJ/m$^3$/h, respectively. The maximum dissipative strain energy conversion rate of the mudstone samples occurs at the moment when the axial stress suddenly drops, and it is very sensitive to the confining pressure. Higher confining pressure equates to greater elastic strain energy accumulation in stage I, a higher dissipative strain energy conversion rate in stage II, and a higher rate of deformation damage of the samples.

**FIGURE 3** Support layout of the tailgate in panel 15107

**FIGURE 4** Energy evolution curve of mudstone samples failure process
3.1 UDEC numerical simulation model

A numerical model (model size 30 m × 40 m) was established, as shown in Figure 7. The horizontal displacement of the left and right sides of the model is restricted, zero vertical displacement is set at the base of the model, and the lateral pressure coefficient is 1.5. The mechanical parameters of strata and joints are shown in Table 1.

3.2 Energy balance in UDEC

The total energy balance can be expressed in terms of the released energy ($W_r$), which is the difference between the work done at the boundary of the model and the total stored and dissipated strain energies:
where \( W_r \) is the released energy; \( W \) is the total boundary loading work; \( U_c \) is the total stored strain energy in rock mass; \( U_b \) is the total change in potential energy of the system; \( W_j \) is the friction work done on joints; and \( W_p \) is the total dissipated work in plastic deformation of intact rock.

The total boundary loading work \( W \) is calculated from the boundary gridpoint forces and displacements. The total change in potential energy of the system \( U_b \)
is calculated from the gridpoint gravitational forces and the displacements of the gridpoints. The total stored strain energy in rock mass \( U_c \) is composed of two parts: the energy stored in the blocks and that stored in the joints. Each is calculated, and the total stored energy, \( U_c \), is determined as the sum of these two components. The friction work done on joints \( W_j \) is calculated by summing frictional energy at the contact during a time step. The total dissipated work in plastic deformation of intact rock \( W_p \) is the difference between the total strain energy and the elastic strain energy in the model. The detailed calculation formula of these parameters can refer to UDEC program specification.27

3.3 | Analysis of simulation results

Roof strength is an important factor affecting roadway stability. Table 2 shows the energy distribution under different roof strengths. Roof strength greatly influences the size of plastic dissipation energy. When the elastic modulus of roof strata decreased from 10 GPa to 1.25 GPa, the plastic dissipative energy increased from 5.312 to 18.17 MJ, increasing by 242.1%, as shown in Figure 8. The variation trends of roof subsidence and plastic dissipation energy under different roof strengths are also shown to be consistent in Figure 8. The plastic dissipative energy is a reflection of the range of the loose circle and the rupture degree of surrounding rock. Higher plastic dissipative energy coincides with a larger circle of surrounding loose rock, and expansion and deformation of the surrounding rock, resulting in greater deformation of the roadway.

Figure 9 shows the distribution curve of plastic dissipative energy when the elastic modulus of roof strata is 2.5 GPa, and the plastic zone distribution maps of 200, 300, 400, 1000, and 2500 steps are used to analyze the expansion law of the plastic zone in the process of plastic dissipative energy evolution. As the plastic dissipative energy increases, the plastic zone of the roadway gradually expands from the roof and floor to two ribs and then tends to be stable. The plastic zone and fracture evolution of the surrounding rock penetrate the whole process of deformation and instability of the roadway. The instability of the roadway is an irreversible energy dissipation process with high nonlinearity and complexity.
4 | ROADWAY SUPPORT PRINCIPLES BASED ON ENERGY BALANCE THEORY

The rock mass is in a balanced energy state before roadway excavation, and stress is redistributed after roadway excavation. Where the stress concentration and energy accumulation occur, the surrounding rock presents plastic deformation and local destruction, releasing energy, which is inelastic and irreversible. The energy involved in the roadway excavation process mainly includes the blasting energy of the explosive, mechanical energy of the rock drill and the roadheader, strain energy of the rock, thermal energy, plastic dissipative energy, and kinetic energy during the impact damage.

The various energies involved in energy conversion are mainly released by deformation energy, which is the product of pressure and deformation. Total energy released is about constant for the roadway. Therefore, there is more pressure acting on the support when there is little surrounding rock deformation. Conversely, substantial surrounding rock deformation occurs when there is substantial pressure acting on the support. Compared with a hard rock roadway, this kind of roadway has a large range of loose circles and a large amount of deformation. Therefore, the support should be adapted to the deformation characteristics of the roadway, which allows the surrounding rock to produce deformation and release part of the energy.

From the perspective of the energy balance principle, the safety control of roadways with a thick and soft roof should be realized from three aspects, as shown in Figure 10. First, optimize the layout of the roadway, avoid placing it in the
stress concentration zone, and reduce the strain energy of the surrounding rock accumulation from the source. The commonly used methods include upward mining, gob side entry driving, and stagger arrangement. The second is to improve the ability of the supporting structure to adapt to the deformation of the surrounding rock and avoid the failure of the supporting structure from excessive force. This mainly includes improving the elongation of supporting materials, rationally arranging the anchor cable position and delayed installation of the anchor cable. The third is to set a weak structure in the surrounding rock, which dissipates part of the energy, thus reducing the load acting on the support. This includes borehole pressure relief, loose blasting, and cutting groove.

5 | IN SITU EXPERIMENT

5.1 | Geological conditions

The headgate of 2927 coal face was excavated along the 9# coal seam floor (Figures 11 and 12), and U29 steel support with the row spacing of 600 mm was adopted. Field monitoring shows that the average deformation speed of rib and floor exceeded 2.0 mm/d, so the roadway needed maintenance to ensure normal production.

5.2 | Numerical simulation of support schemes

The collaborative control technology combined pressure-relieved boreholes with U-steel support was proposed
based on energy balance theory. Numerical analysis was carried out using FLAC5.0 software to analyze the support effect of each scheme. Based on the geological conditions of 2927 coal face in Luling Coal Mine, a numerical model with a length of 30 m and a height of 40 m was established, as shown in Figure 13. The left, right, and lower boundaries of the model are displacement fixed constraints, and the initial in situ stress acting on the upper boundary of the model was 15 MPa. The lateral pressure coefficient was set to be 0.8 based on extensive in situ measurements in the Huaibei mining area. Mohr–Coulomb model was adopted using the mechanical parameters shown in Table 3.

Three models were set up for comparative analysis in numerical simulation (Figure 14). U-shaped steel support was adopted in the 1# model, and leg-locked bolts were added in the 2# model to improve the stability of the U-shaped steel support. The pressure relief borehole with a diameter of 100 mm and a length of 4000 mm was adopted in the 3# model. In FLAC5.0, the beam unit was used to simulate the U-steel support, and the yield bending moment of the support was 42 966 N·m.

The bending moment of the U-steel in the ribs of the roadway was much higher than that of the vault in 1# model, and the maximum bending moment of the U-steel support was 48 254 N·m, which was greater than the yield bending moment of the bracket of 42 966 N·m. The roadway roof to floor deformation was 606 mm, and the floor heave accounted for 82% of the roof to floor deformation. The deformation of two ribs of the roadway reached 270 mm, as shown in Figure 14.

The maximum bending moment of the U-steel in the ribs of the roadway after the implementation of the leg-locked bolts was reduced from 48 254 to 38 563 N·m in the 2# model, which was reduced by 20.08%. Displacement monitoring indicated that the roadway roof–floor deformation was reduced from 606 to 541 mm, which was a 10.7% reduction. The rib–rib deformation of the roadway went from 270 to 192 mm, which was a 28.9% reduction. Overall, structural compensation using the leg-locked bolts can improve the load-bearing capacity of the U-steel support, but the maintenance effect of the roadway has not been substantially improved, especially with no change observed for the floor heave of the roadway.

The maximum bending moment of the U-steel support was 30 436 N·m in the 3# model, which is 36.9% lower than that of the 1# model, and the bearing capacity of the U-steel support was further improved. The surrounding rock vertical stress distribution of the roadway rib before and after the pressure relief borehole implementation is shown in Figure 15. After the pressure relief drilling, the distance from the vertical stress peak to the surface of the roadway increased from 1.6 to 4.2 m, and the whole roadway was in the stress reduction zone. At the same time, the large-diameter pressure relief boreholes also provide a certain deformation compensation space for the broken coal rock mass, and reduce the pressure of surrounding rock acting on the support.

5.3 Support parameters and effects

Combined with the geological conditions and numerical simulation results, the headgate of the 2927 coal face was
and avoid the failure of the supporting structure from excessive force; and (c) set weak structure in the surrounding rock, which dissipates part of the energy, thus reducing the load acting on the support. The results of field application monitoring show that severe deformation of the roadway in Luling Coal Mine was effectively controlled during mining using pressure relief boreholes. The average floor heave speed of the roadway decreased from the original 2.0-0.98 mm/d, so the roadway section can meet the production requirements.

CONFLICT OF INTEREST
The authors declare no conflicts of interest.

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