Research Article
Evolution of Mining-Induced Stress in Downward Mining of Short-Distance Multiseam

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The strong disturbance of upward mining of the short-distance multiseam results in frequent fractures in the coal seam to be mined, jeopardizing the operation safety due to the instability of the surrounding rock of the overlying stope and risks of coal and gas outburst. A lucrative alternative is the downward mining of multiseam, which wide implementation is limited by the lack of reliable data on stress evolution characteristics. In this paper, the evolution, nonlinear interaction, and superposition characteristics of mining-induced stress after multiple pressure relief in the short-distance multi-seam were investigated by the numerical simulation and physical simulation, taking the endowment and mining technology characteristics of group B short-distance multiseam in Pan’er Coal Mine in Anhui Province of China as an example. The numerical and physical models of multiple (2—5 times) mining activities in pressure relief multiseam (with five main coal layers) under downward mining are constructed and implemented with the FLAC3D software and 1:100 scaling, respectively. The results showed that a pressure relief arch, which bears and transfers stresses, is formed on the coal seam’s roof and floor. In downward mining, the local arch expands to the long arch, which bearing capacity within the influence range of multiple mining disturbances is weak, and the arch structure is prone to instability. If a coal pillar is left in the upper coal seam after its mining, the lower coal seam mining will switch from the large-scale low-stress area under the goaf to the small-scale high-stress area of nonlinear interaction of the multiple mining stresses with the stress under the coal pillar.

1. Introduction

With an annual growth of coal mining depth worldwide, multiseam mining encounters such technical problems as the easy occurrence of rockburst, difficulty preparing roadway support, controlling roof and floor, gas drainage, and setting up coal pillars [1—4]. In particular, such problems became very topical in several mining regions of China, including the Datong, Xishan, Xinwen, Pingdingshan, Huainan, and Guizhou Shuicheng mining areas [5]. A lucrative solution to these problems is downward mining, which can improve the productivity and efficiency of multiseam coal mining activities [6]. However, downward mining may jeopardize the operation safety due to excessively high mining-induced stress concentration at the working face [7]. Therefore, the robust substantiation of rock strata stress evolution and interaction induced by longwall mining activities under multiple coal seams is crucial for the wider implementation of downward multiseam mining. It is also topical to ensure mining safety by assessing the supporting efficiency, deformation of surrounding rock, and gas drainage parameters.

The study of stress redistribution and evolution caused by the single coal seam mining is the basis for studying those caused by longwall mining under multiseam. The mining of coal seams violates the in situ stress balance of the strata, causing continuous deformation, movement, and fracture of the rock strata near the working face [8, 9]. The fissures in the adjacent coal and rock masses initiate and expand,
generating new channels for the flow and accumulation of free gas in the coal seams. This phenomenon is used in the simultaneous extraction of coal and gas [10–12]. At present, there are quite mature theoretical and experimental methods to study the evolution characteristics of mining-induced stresses caused by mining activities in a single coal seam [13, 14]. Generally, scholars use theoretical analysis, in situ/field testing, numerical simulation, physical simulation, and other methods to study the mining-induced stress field and the development and evolution characteristics of mining fissures in coal seams [15–18]. They have achieved significant results on preventing coal and rock dynamic disasters, derived the relationship between the stress distribution in the mining space and mining disturbance, and revealed the influence of hard roof hydraulic fracturing on mining-induced stress. These studies are of great significance to ensure the safe and efficient mining of a single coal seam working face. However, the extension of these findings to multiseam is quite problematic due to the following challenges.

According to the interaction between the caving, broken, and deformation zones induced by the downward mining activities of multiple coal seams, they can be subdivided into short-distance, medium-distance, and long-distance multiseam. When the caving zone induced by the mining activity of the middle and lower coal seams is connected with the failure zone of the upper coal seam floor, such multiseam is considered a short-distance one [19]. This study focuses on this type; so, the description of the other two cases can be found elsewhere [6]. At present, there are several publications on the relationships between mining stress and coal deformation failure, gas pressure changes, the laws of overlying rock migration, and gas accumulation in the mining of multiseam [20, 21]. Most of these studies analyzed the breaking of the single roof structure and the evolution of mining-induced stress. However, the upper coal mining space and the remaining coal pillars have a linkage response to the stress of the lower coal seam, which aggravates the breakage of the roof and floor structures of the working face [22–24]. Therefore, it is necessary to study the stress transfer process caused by the coupled development of the multiseam roof and floor’s fractured structure, the mining-induced stress evolution, superposition, or/and coupling.

This paper proposes numerical and physical models for assessing the stress interaction and evolution under multiple mining conditions of a particular mine in China based on multisource data verification. The geological exploration data, experimental data, in situ stress test data, and the actual exposure data in excavation and mining are used to elaborate the numerical model run via the FLAC3D software package. Under the same conditions, the physical model is used to verify the mining-induced stress evolution characteristics of downward mining of the short-distance multiseam and test the applicability of the dynamic mining-induced stress superposition effect. It provides a theoretical basis for the further study of gas analysis and emission, migration law, and the simultaneous extraction of coal and gas. Also, it has important reference significance for engineering practice under similar mining conditions.

2. Geological Conditions of the Mine under Study

This work considered the occurrence characteristics and mining technology features of group B coal in the short-distance multiseam of the Pan’er Coal Mine located in the Huainan Mining Area of Anhui Province, China, as shown in Figure 1(a).

The West 4th mining district borehole statistical data and the average thickness distribution of the coal and rock layers were used as the input data for the numerical and physical models. The above geological structure was quite complicated and characterized by numerous faults, with drops exceeding the coal layers by thickness. This implied significant discontinuities in the lithology, roof, and floor of coal seams at different levels. Group B coal resources in the Pan’er Coal Mine contained five main coal seams with an average inclination of 15°, namely, 8-1, 7-1, 6-1, 5-2, and 4-1, numbered from top to bottom in Figure 1(b). In addition, the above stratigraphic column contained four more secondary coal layers, namely, 7-2, 6-2, and 4-2 (with thickness ranging from zero to 1.22 mm), as well as 5-2 (with a thickness range from 0.49 to 2.48 m). The roofs and floors of the main coal seams were mostly composed of mudstone and sandy mudstone with small shares of sandstone and siltstone. The latter two rock types were characterized by high compressive strength, in contrast to mudstone and sandy mudstone, which were easily broken and prone to water leakage. Thus, the roof and floor rocks had relatively low strength and poor stability.

Insofar as the coal seams in this mine had a high gas concentration (12–25 m³/t) and high gas pressure (1.5–4 MPa), the downward mining method was adopted to reduce the stress and gas pressure in the coal and rock mass above and below the goaf and eliminate the risk of coal seam outburst.

3. Methods

The mining-induced stresses in multiseam form an exceptionally complex stress field due to repeated mining activities, which damage or even break the surrounding rocks of particular coal seams. Since the respective rock strata displacements are nonlinearly related to respective strains and stresses, the superposition principle, which implies that elastic stresses and strains can be directly summed up, is no longer applicable. The mining-induced stress field results from complex spatial–temporal variations of the nonuniform and unstable stress-strain states. Therefore, this study adopted the numerical simulation method for the stress distribution analysis. Besides, physical modeling with 1:100 scaling and respective physical simulation materials was performed to verify the numerical simulation results.

The adopted approach’s flowchart is presented in Figure 2.
Firstly, based on the existing geological data and experimental data, a three-dimensional numerical model conforming to the real coal-bearing strata’s distribution characteristics and geological structure was constructed. Secondly, the coal and rock constitutive relationships were derived, the model constraints and boundary loads were preset, and the numerical simulation was performed. In parallel, the physical simulation model with a 1:100 scaling ratio was realized, with stress measurement via installed strain sensors. The physical parameters of the mining response at each measured point in the model were extracted, and the errors were compared with the previously measured data. Then, the concept of combined error was adopted. After determining the weight of each factor, the combined error was calculated, and if the latter exceeded the limiting value, the initial simulation constraints and boundary loads were automatically adjusted. The numerical simulation was restarted, and the above procedure was repeated until the combined error fell within the acceptable range. At this point, the numerical simulation was terminated, and the respective constraints and boundary loads were treated as optimal ones. They were incorporated into the numerical model for stress calculation and stability analysis. The numerical model under the key simulation conditions was verified by the physical simulation model, yielding the multi-information database under different conditions.

### 3.1. Numerical Simulation

#### 3.1.1. Determination of Model Parameters

The numerical simulation was performed with the explicit finite difference method via the FLAC3D 5.00 commercial software package, which is widely used in mining engineering. A numerical model of the multiseam with the actual coal strata was established. The length × width × height was 300 m × 300 m × 250 m. According to the boundary conditions, the horizontal displacement of the four vertical planes of the model was limited, and the vertical displacement at the bottom of the model was defined as zero. The coal pillar adopted the strain-softening model, the goaf adopted the double-yield...
model, and the rest of the model adopted the Mohr-Coulomb model, as shown in Figure 3.

Accurate surrounding rock parameters must be obtained to ensure reliable numerical simulation results. The physical and mechanical parameters of coal and rock strata were determined via a comprehensive statistical analysis of the mechanical test data on field coring and laboratory coal and rock samples combined with data provided by the mining producer. The surrounding rock parameters obtained in the laboratory must be reduced to some extent when implemented in the numerical simulation calculation, and determination of the reduction factor is particularly important. According to previous studies, the modulus of elasticity, cohesion, and tensile strength is 0.2 times the test value, and Poisson’s ratio is 1.2 times the test value. The main parameters are listed in Table 1. As seen in Table 1, only one secondary coal layer 5-2 was considered, while three thinner secondary coal layers, namely, 7-2, 6-2, and 4-2 (with thickness ranging from zero to 1.22 m), were disregarded.

3.1.2. Calculation of Model’s Boundary Loads. The following approximations were made to simulate the multiseam stress state.

The model’s upper surface is as follows: the overburden gravity force was assumed to be equivalent to the uniform load applied to the model’s upper surface. Assuming the overburden’s approximate average density $\rho = 2.5 \times 10^3$ kg/m$^3$ and
the burial depth of the model’s top surface model $H = 430$ m, the equivalent uniform stress in the model’s top was derived as $q = \rho g H = 10.75$ MPa $q = \rho g H = 10.75$ MPa.

The model’s side surface is as follows: the maximum principal stress orientation in the West 4th mining district in Pan’er Coal Mine was NE65°–NE95°, and the maximum and minimum principal stress-vertical stress ratios were 1.4 and 0.8, respectively. Therefore, it was assumed that the overlying stratum gravity force multiplied by 1.4 was applied to the model’s front and back side surfaces, simulating the maximum horizontal principal stress action. Similarly, the overlying stratum gravity force multiplied by 0.8 was applied to the left and right model, simulating the minimum horizontal principal stress action and the corresponding stress gradient in a positive trapezoidal loading mode along the vertical direction of the model.

The strata self-weight is as follows: the coal and rock mass-induced gravity forces were derived by multiplying the density of each coal and rock layer by gravity acceleration.

3.2. Physical Simulation. To validate the numerical model feasibility and analyze the evolution and interaction of mining-induced stress fields in the multiseam, coal’s mechanical, and mining characteristics in the West 4th mining district of the Pan’er Coal Mine were used to elaborate a physical simulation model with a scaling ratio was 1:100. Thus, the model’s length, width, and height were 4.0, 0.4, and 1.3 m, respectively. The details of the physical simulation model and the layout of pressure sensors for stress measurements are shown in Figure 4.

3.3. The Combined Numerical Simulation and Physical Simulation Procedures. Considering the actual length of the working face, the inclined mining length of the numerical model was set to 200 m, and the advance length of each layer was 200 m, with an advance step of 8 m. The following procedures were realized to assess the distribution of mining-induced stresses in the simulated multiseam under downward mining conditions.

(1) Numerical and physical simulation models of stress distribution during mining of a single 8-1 coal seam were constructed, and their results were compared to verify their reliability and accuracy

(2) Numerical simulation was performed to simulate the mining-induced stress distribution characteristics near the working face during the downward mining of 8-1, 7-1, 6-1, 5-2, and 4-1 coal seams

(3) Based on the physical simulation model with respective simulation materials, the distribution and evolution of stresses induced by the mining of 8-1 to 6-1 coal seams were assessed near the working face
**Table 1**: Physical and mechanical parameters of rock and coal samples used in the numerical simulation.

| No. | Lithology         | Thickness/m | Density/(kg/m³) | Bulk modulus/GPa | Shear modulus/GPa | Cohesion/MPa | Friction angle(°) | Tensile strength/MPa |
|-----|-------------------|-------------|-----------------|------------------|-------------------|--------------|------------------|----------------------|
| 1   | Roof strata       | 50.0        | 2600            | 3.33             | 2                 | 0.85         | 30               | 1.22                 |
| 2   | Fine sandstone    | 8.5         | 2794            | 20.14            | 16.27             | 3.8          | 43               | 6.75                 |
| 3   | Sandy mudstone    | 2.5         | 2520            | 10.76            | 5.7               | 1.18         | 35               | 1.17                 |
| 4   | 8-1 coal          | 3.0         | 1420            | 1.9              | 0.93              | 0.2          | 20               | 0.28                 |
| 5   | Sandy mudstone    | 9.4         | 2446            | 10.76            | 5.7               | 1.18         | 35               | 1                    |
| 6   | Sandy mudstone    | 6.6         | 2417            | 10.76            | 5.7               | 1.18         | 35               | 1.64                 |
| 7   | 7-1 coal          | 2.6         | 1370            | 2.8              | 1.2               | 0.6          | 27               | 0.4                  |
| 8   | Sandy mudstone    | 13.5        | 2549            | 10.76            | 5.7               | 1.18         | 35               | 1.31                 |
| 9   | Mudstone          | 2.5         | 2437            | 4.3              | 2.8               | 0.7          | 30               | 1.8                  |
| 10  | 6-1 coal          | 2.5         | 1390            | 2                | 0.88              | 0.42         | 24               | 0.3                  |
| 11  | Mudstone          | 4.1         | 2545            | 5.8              | 3.2               | 1.2          | 30               | 3.25                 |
| 12  | Fine sandstone    | 2.0         | 2800            | 16.04            | 12.02             | 3.47         | 43               | 4.96                 |
| 13  | Coarse sandstone  | 5.9         | 2700            | 7.35             | 6.63              | 3.04         | 40               | 4.34                 |
| 14  | Fine sandstone    | 3.0         | 2800            | 16.04            | 12.02             | 3.47         | 43               | 4.96                 |
| 15  | 5-2 coal          | 1.6         | 1410            | 1.73             | 0.82              | 0.18         | 20               | 0.2                  |
| 16  | Fine sandstone    | 3.0         | 2597            | 15.28            | 11.2              | 3.1          | 42               | 3.48                 |
| 17  | 5-1 coal          | 1.1         | 1410            | 1.73             | 0.82              | 0.18         | 20               | 0.2                  |
| 18  | Fine sandstone    | 3.6         | 2586            | 18.02            | 14.02             | 3.8          | 43               | 5.13                 |
| 19  | Sandy mudstone    | 3.8         | 2520            | 4.9              | 3.2               | 1.18         | 35               | 1.8                  |
| 20  | Mudstone          | 2.8         | 2567            | 4.3              | 2.8               | 0.7          | 30               | 1.68                 |
| 21  | 4-1 coal          | 3.1         | 1460            | 2.12             | 0.93              | 0.5          | 24               | 0.35                 |
| 22  | Mudstone          | 2.8         | 2463            | 3.94             | 2.6               | 0.68         | 30               | 0.98                 |
| 23  | Floor strata      | 20.0        | 2463            | 3.94             | 2.6               | 0.68         | 30               | 0.98                 |
The evolution and redistribution characteristics of mining-induced stresses in the multiseam were studied through a comprehensive analysis of numerical simulation and physical simulation results.

4. Results

4.1. Numerical Simulation Results. Figure 5 shows the numerical simulation results on the vertical stress field evolution at four different advance steps of the single 8-1 coal seam mining, namely, at advance distances of 24, 56, 120, and 200 m.

When the stress exceeds the yield stress level of the particular material, plastic deformation occurs. Depending on the strain-hardening or strain-softening characteristics of the material, the stress-strain curve changes from linear to nonlinear one, with possible time-dependent creep and stress relief (relaxation) phenomena.

In the early stage of mining 8-1 coal, the in situ stress in the approximate equilibrium state was observed, while a small-scale arch-shaped stress-relief area was formed in the vertical direction of the roof and floor of the coal seam (inverted arch in the floor). In the horizontal direction, different degrees of stress concentration around the goaf were observed. With an increase in the mining distance, the...
height and depth of the stress-relief area increased, and the respective arch expanded in the horizontal direction. However, after the main roof collapse, the stress-relief area extension rate first dropped and then remained almost unchanged until the mining area approximated a square.

With a continuous advance of the working face, under the action of self-weight and overburden load, when the suspension span of the main roof reached a specific length, periodic weighting and collapse occurred. The periodic weighting phenomenon is known to occur during the retreat of the face under certain geologic conditions: strong strata in the immediate and main roof tend to cantilever over the gob, weighting the supports periodically. The rear goaf was gradually compacted, and the fractures and separation were gradually closed. Hence, the overburden load, which was transferred downward and led to the goaf area restoration, exceeded the original-state in situ stress. In addition, the distribution of mining-induced stress was synchronized with the periodic weighting, and the maximum height and depth of the stress-relief area remained approximately unchanged. The stress-relief areas of the roof and floor gradually shifted forward as the advanced distance increased. As shown in Figure 5, during the 8-1 coal seam mining, at the advance distances of 24, 56, 120, and 200 m, the vertical heights of stress relief in the roof strata were 20, 43, 62, and 100 m, respectively, while the stress-relief area depths within the floor strata were 15, 33, 50, and 73 m, respectively.

To study the impact of the 8-1 coal seam mining on the maximum vertical stresses of different coal seams, five monitoring lines were set at certain positions on each main coal seam floor in the numerical simulation process. The stress data were then extracted, and the corresponding curves were plotted in Figure 6.

As shown in Figure 6, with a continuous increase in the distance between the monitoring line and the 8-1 coal seam, the mining-induced stress disturbance was reduced, exhibiting different distribution patterns. Thus, a W-shaped distribution of the vertical stress was observed on the 8-1 coal floor. The peak value on the right side of "W" was the front abutment pressure of the working face; the median peak value of "W" was the maximum value of stress restoration in the goaf due to roof caving; the peak value on the left side was the stress-concentration area at the coal pillar behind the setup entry. The peak value on the left side was approximately 31 MPa and a stress concentration factor of 2.4. The latter was reduced to 2.3 on the 7-1 coal seam floor and to 2.1 on the 6-1 coal seam floor. The vertical stress concentration factor near the 5-2 and 4-1 coal seams’ floors dropped approximately to 1.5. This implies that with an increase in the distance between 8-1 coal seam and other coal seams, the mining-affected vertical stress of each seam gradually decreases, and the stress distribution curve gradually approaches a horizontal straight line, becoming uniform. The stress peak and stress difference gradually attenuate in the process of downward transmission, and the mining-induced stress distribution changes from "W" type to "U" type with increased depth.

According to the downward mining sequence adopted in the numerical model, the mining-induced stress field distributions after the successive exploitation of 8-1, 7-1, 6-1, 5-2,
and 4-1 coal were simulated and analyzed. The results are depicted in Figure 7.

After mining each coal seam, the scope of the roof of the coal seam area and the surrounding stress-concentration area showed band-like distribution characteristics. In particular, the stress gradient at the boundary of the goaf and the stress-relief area on the bottom plate had a ring shape. However, with an increase in the relative distance from the floor, the annular stress-relief area was reduced.

4.2. Physical Simulation Results. Considering the actual test conditions, the stress evolution and release during the downward mining process of only two coal layers (namely 8-1 and 6-1) were assessed using physical simulation. Following on-site mining conditions, the coal pillar mining method was adopted for the upper coal seam. Boundary pillars of 60 cm were left at both ends of the coal seam. The upper coal seam was mined at 50 cm and left at 30 cm pillars, and the mining was subsequently continued for 90 and 20 cm pillars. The final mining was at 90 cm. After the 8-1 coal seam mining was completed and stabilized, the 6-1 coal seam was gradually mined at 280 cm. Each mining step had an interval of 5 cm. Black rock formations marked the exploited 8-1 and 6-1 coal. In the disregarded coal seams, no mining activities were conducted.

As indicated by the physical simulation results on the overburden pressure in Figure 8, when the 6-1 coal seam was mined under the uncompacted goaf of 8-1 coal seam, the main roof did collapse after advancing 75 m from the setup entry, and a large area of unfilled goaf was observed. This can be attributed to the fact that the interval between 8-1 and 6-1 coal seams was only 22–27 m. The mining-induced high stress in the upper coal seam was effectively reduced by the pressure relief effect, while the joint action of the front abutment pressure and high stress below the pillars promoted the stress concentration in the area below the pillars and contributed to a significant energy accumulation in front of the working face. After mining the advance length of 75 m below the 8-1 coal pillar, the pressure on the working face was suddenly increased, and the rock formation between 6-1 and 8-1 coal seams’ floors experienced the overall collapse. This result was due to the shear failure of the local bearing structure of the voussoir beam caused by the combined effect of the self-gravity of the rock above the 6-1 coal seam, high stress in front of the mined 6-1 coal seam, and the stress concentration in the coal pillar. The goaf was restored to the in situ stress area by stress restoration in the stress-relief area. High-stress release in the concentrated zone below the coal pillar and redistribution of the stress field were achieved. The results also indicated that the distribution of mining-induced stress at different floor depths after 8-1 coal mining was the same in the numerical and physical simulation models. The mining-induced stress distribution regularities under the coal pillar or solid coal in front of the working face were similar.

The stress distribution of the 8-1 coal seam’s floor is depicted in Figure 9.

After mining the upper 8-1 coal seam with the coal pillar, the stress in the lower part of the goaf located in the stress-relief area was smaller than that in the original rock. The stress below the coal pillar on both sides of the working face was higher than the original rock stress, and the stress concentration below the middle pillar and that below the side coal pillar became noticeable. Meanwhile, the stress concentration below the 20 cm pillar was larger than that below the 30 cm pillar. This implies that the pillars not only bore the load but also transferred the stress. The stress concentration was high in the inner and lower coal seams. With an increase in coal pillar width, the stresses increased. Still, the stress concentration coefficient was reduced, indicating that the coal pillar width on the energy distribution in the coal seam inside and below the coal pillar was weakened.

A comparison of the stress distributions at 3.5 and 16.5 m below the floor showed that the amplitudes of all mining-induced stress distribution curves were gradually reduced with depth. The stress concentration values corresponded to the original rock stress values, thereby proving a close correlation of the numerical simulation and physical
simulation modeling results. That is, as the relative depth increased, the energy accumulation in the floor rock was reduced.

5. Analysis of Mining-Induced Stress Evolution and Redistribution

After the upper (8-1) coal seam was mined, the processes of roof caving, bed separation, and bending settlement occurred from bottom to top. With the stress transfer of the overlying strata, the roof and floor of the coal seam produced pressure relief due to the formation of a free surface after mining, which jointly formed the roof stress arch, which could bear the overlying strata load and transfer the stress downward, as shown in Figure 10.

When the stress was transferred to the coal pillars on both sides, the annular stress-concentration area is formed. With an increase in the mining distance, the roof and floor pressure relief arch ranges also increased. After the main roof fracture, the stress in the goaf area behind the working face had fluctuating characteristics. With an increase in the floor depth, the fluctuation range was gradually reduced.

Compared with the single 8-1 coal mining, the stress distribution characteristics of the strip in the descending sequential mining of 7-1, 6-1, and 5-2 coal seams were more pronounced, which reflected the high-stress generation effect in the descending mining process. Affected by the multiple mining of several coal seams, the disturbance of the stress field at the roof and floor of the 4-1 coal seam in the lowest part of the coal seam was enhanced, the stress-relief area was significantly increased, and the regionalized distribution of the strip was no longer obvious.

As shown in Figure 11, the local stress arch range between 7-1 and 8-1 coal seams increased with the mining distance in the sequential mining of the lower coal seams. After entering the affected area, the roof stress arch height increased sharply to form an entire stress arch, which bearing capacity was weaker than that of the local one, and the arch structure exhibited instability. The overlying stratum stress was rapidly transferred downward, and the bearing effect of this type of arch structure was gradually weakened with the disturbance of multiple coal mining.

Due to the remaining coal pillars in the 8-1 coal seam, the weight of the coal and rock mass on both sides of the goaf was borne by the coal pillars. Under the coal pillar, due to the short distance between the two coal seams, the stress concentration area was produced. In the affected area of the coal pillar, the advance abutment pressure of 7-1 coal face overlapped the concentrated stress area under the coal pillar. The stress concentration factor evolution exhibited a nonlinear stress accumulation trend. The load-bearing structure of the stress arch was unstable, and a large amount of energy accumulated in the working face was dissipated in a short period from the stress-concertation area to the stress-relief area. On the one hand, the bracket had to bear the immediate stress impact. On the other hand, the release of
Figure 10: Evolution characteristics of the vertical stress field during the single 8-1 coal seam mining.

Figure 11: Evolution characteristics of the stress field during coal seam mining.
the concentrated stress in the range of the coal pillar subjected the coal wall to higher shear force, which would damage and break the “masonry beam” structure, forcing the support to bear the weight of irregular rock blocks with large span and height. Therefore, when mining the upper coal seam, the coal pillar should not be left as far as possible to prevent the dislocation of strata, large amount of separation, asymmetric fracture development, and other phenomena caused by the long-term action of high stresses.

In the actual production, the mining in the area with generated high stresses should be avoided as far as possible, or the roof caving measures should be taken to prevent frame-crushing accidents caused by extreme weighting in the area stope.

6. Discussion

The comparative analysis of Figures 7 and 11 revealed that in the multiseam short-distance mining, the mining activities in each coal seam caused changes in the in situ stress field of nearby coal strata. Therefore, the mining-induced stress field of multiseam short-distance mining was much more complex than a single coal seam. Using the proposed numerical simulation model, the mining-induced stresses on the monitoring line were assessed when each coal seam was mined separately. Then, the difference between the mining-induced stress value and the in situ stress at this location was treated as the disturbance stress caused by the coal seam mining. Finally, by superimposing the disturbance stress caused by each coal seam mining with the in situ stress at this location, we can assess the interactive effect of mining-induced stress in the process of coal seam mining compared with the mining-induced stress in the numerical model of downward mining. Figure 11 shows the change of mining-induced stress at different positions in the short-distance multiseam according to different mining methods. For brevity sake, the results are presented only for three monitoring points with representative positions, namely, (1) 60 m above the 8-1 coal seam roof, (2) the 6-1 coal seam floor and roof, and (3) 60 m below 4-1 the coal seam floor within 5 m depth of the cut coal pillar.

It can be seen from Figure 12 that the individual mining activities in each coal seam will cause obvious disturbance stresses. The degree of disturbance mainly depends on the distance between the monitoring point and the coal seam. With an increase in the latter distance, the mining-induced stress effect is weakened, but its multiple interaction effects cannot be ignored. In addition, the mining-induced stresses in monitoring points No. 1 and No. 3 during the multiseam mining satisfied the stress superposition principle because their stresses and strains were elastic. In contrast, the calculation error of monitoring point No. 2 was relatively large because this monitoring point was in the middle of the multiseam, closed to several coal seams, and influenced by all the mining activities. Under the influence of several strong factors, the coal pillar at the monitoring point went beyond the elastic deformation stage and entered the plastic yielding or even postpeak residual deformation stage. Then, a part of the high stress was transferred to another coal and rock mass, reducing the actual mining-induced stress below the value obtained via the stress superposition.

7. Conclusions

The results obtained made it possible to draw the following conclusions.

(1) In the process of coal mining, a pressure relief arch, which bears and transfers stresses, is formed on the coal seam’s roof and floor. In downward mining,
the local arch expands to the long arch, which bearing capacity within the influence range of multiple mining disturbances is weak, and the arch structure is prone to instability. The stress of the overlying strata quickly transfers downward, resulting in a stress concentration effect, which is weakened with the stress transfer depth.

(2) If a coal pillar is left in the upper coal seam after its mining, the lower coal seam mining will switch from the large-scale low-stress area under the goaf to the small-scale high-stress area of nonlinear interaction of the multiple mining stresses with the stress under the coal pillar. As a result, the load-bearing arch effect will be eliminated, and the mining deformation values will quickly change from small-scale to large-scale ones, leading to the redistribution of strain and stress fields. Given this, necessary roof caving or scour prevention measures should be taken to mitigate the additional mine pressure caused by the rapid energy release in the stress concentration zones.

(3) In the process of downward mining, with an increase in the width of the coal pillar left in the upper goaf, the stress concentration area increases while the stress concentration factor decreases. Therefore, the increased coal pillar’s width is conducive to reducing stress concentration in mining. The backfilling method can mitigate the weighting effect induced by high stress concentration in coal pillars.

Data Availability

The data used for conducting classifications are available from the corresponding author upon request.

Conflicts of Interest

The authors declared no potential conflicts of interest with respect to the research, authorship, and/or publication of this article.

Authors’ Contributions

All authors contributed to this paper. Ke yang prepared and edited the manuscript. Qinjie Liu and Qiang Fu made a substantial contribution to the data analysis and revised the article. Xiang He and Xin Lyu reviewed the manuscript and processed the investigation during the research process. Ke Yang and Qiang Fu provided fund support.

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