

Research Article

Evolution Characteristics of Overburden Instability and Failure under Deep Complex Mining Conditions

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Aiming at the problems of intensified deformation and serious stress damage of stope overburden during mining in deep typical geological environment, taking the deep mining face of a mine in the northwest as the research background, the characteristics and influencing factors of overburden collapse in deep geological environment are analyzed, a comprehensive evaluation system affecting overburden instability and deformation is established, and the temporal and spatial evolution law of deep stope roof failure is obtained by using the "three-zone" distribution theory of overburden, analytic hierarchy process, and numerical simulation method. The results show that high ground stress has the most significant impact on the geological environment of deep stope, followed by the properties of surrounding rock, hydrogeological conditions, and unloading disturbance. The damage degree of the plastic zone at the top of the overburden is greater than that at the bottom. With the increase of mining time, the damage range of the roof and floor gradually increases. In the process of coal seam mining, the increasing range and sinking trend of roof stress are larger near the working face, and the farther away from the working face, the increasing range decreases continuously. The research conclusion provides an important reference for coal seam safety mining and surrounding rock control in complex environments such as deep mining, high stress, and strong disturbance.

1. Introduction

With the upgrading of coal mining technology and the progress of production mode, shallow coal resources are increasingly depleted due to large-scale mining, and the development and utilization of underground space resources gradually extend to the deep. In the process of deep development of underground mining, the geological environment around the stope also changes with the increase of depth, which brings a series of geological disasters. These problems will lead to uneven distribution of overlaying rock pressure and excessive deformation of surrounding rock. It is easy to cause rock burst, coal and gas outburst, and rockburst in mining engineering operations, which seriously threatens the personal safety of construction personnel and affects the safety of mine production.

Around the evolution law of geological and mechanical environment in deep mining, academic circles and engineering technicians have carried out fruitful research. Chen et al. [1] studied the characteristics of surrounding rock in the process of roadway excavation. Systematic experimental analysis was carried out on the factors related to rock mass characteristics such as different distributions of surrounding rock fractures, surrounding pressure, and temperature changes. It was concluded that the fracture extension zone formed at different positions of rock mass and the corresponding rock mass failure characteristics. Ma et al. [2] proposed the support technology of the combination of truss anchor cable and anchor cable network. Combined with the mechanical conditions and environmental characteristics of deep roadway, the high-strength anchor cable support technology can control the deformation of surrounding rock in a certain range. By quantitatively analyzing the constitutive damage curve model of metamorphic marble and metamorphic granite with large impact stress failure and combining the constitutive damage elastic model and constitutive elastic deformation model, Shan et al. [3] established a basic constitutive damage elastic model of metamorphic



FIGURE 1: Distribution map of overlying strata "three zones."



FIGURE 2: Three-dimensional schematic diagram of overlying rock roof collapse.

covered rock without deformation after large impact stress failure. Academician Qian [4] said that the interaction between the high initial ground stress in the deep and the unloading surface of the roadway formed during excavation will cause partition fracture. Yao et al. [5] applied field measurement, theoretical analysis, numerical simulation, and other methods to conduct in-depth analysis of coal pillar in the working face, obtained the variation law of overburden pressure in fully mechanized mining, and summarized the movement law and development characteristics of coal pillar in overburden in a fully mechanized mining face. Wang et al. [6] constructed a structural model of coal seam mining through similar materials, which was used to observe the collapse law of the overlying strata under different excavation stages. The infrared thermal imager and highprecision speckle system were used to study the temporal and spatial characteristics of infrared radiation during the movement of the overlying strata. Based on the study of rock movement process, Xu et al. [7] regard the change of mining-induced rock as the dynamic process of unloading expansion and recompaction, revealing the cumulative effect of unloading expansion of mining-induced rock and its influence mechanism on rock movement law. According to the characteristics of rock failure and deformation, Pan et al. [8] increased the parameters that actually affect the surrounding rock of the roadway. Therefore, a rock model with full stress-strain curve segmentation was established, and the



FIGURE 3: Two-dimensional schematic diagram of overlying rock roof collapse.

mechanical properties and elastic-plastic solution of circular roadway were deduced. Finally, the distribution characteristics of elastic zone, plastic residual zone, and plastic softening zone of deep circular roadway under the action of nonuniform stress field were analyzed. Azim [9] established a numerical model of pore elasticity by a finite element method coupled with water flow, pressure, and other related environmental factors and studied the main influencing

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FIGURE 4: Evaluation factor structure.

TABLE 1: Index hierarchical structure model.
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	C ₁ groundwater seepage
B ₁ geohydrologic environment	C_2 gas pressure
	C_3 ground temperature
	C ₄ mining stress
B_2 high ground stress	C_5 structural stress
	C_6 rock elastic modulus
	C_7 shear modulus of rock
B ₃ surrounding rock property	C_8 rock cohesion
	C_9 internal friction angle of rock
	C_{10} rock integrity
	C_{11} roadway layout
B_4 disturbance effect	C_{12} form of construction work
	C_{13} mining cycle
	 B₁ geohydrologic environment B₂ high ground stress B₃ surrounding rock property B₄ disturbance effect

TABLE 2: Nine-stage scale method.

Scale	Implication
1	Represents that two elements are equally important
3	Represents that the former is slightly more important than the latter in two elements
5	Represents that the former is obviously more important than the latter in two elements
7	Represents that the former is more strongly important than the latter in two elements
9	Represents that the former is more extreme important than the latter in two elements
2, 4, 6, 8	Represents the intermediate value of the above judgment
Reciprocal	If the judgment of factors <i>i</i> and <i>j</i> is a_{ij} , the judgment of the comparison of factors <i>j</i> and <i>i</i> is $a_{ji} = 1/a_{ij}$

factors of hydraulic fracturing expansion when hydraulic fracturing technology was applied to tight sandstone reservoirs. Valliappan et al. [10] studied the variation law of hydraulic fracturing in anisotropic media. The twodimensional numerical model of hydraulic fracturing is established by using the finite element method to couple the physical fields such as pore elasticity and local pressure. The influence of anisotropic parameters on crack propagation is summarized. Huang [11] used numerical simulation and mechanical analysis to study the water inrush mechanism of stope roof under the combined action of dynamic and static loads, deduced the mechanical basis of instability and water inrush of floor water-resisting layer, and discussed the influence of roof collapse on dynamic load and the influence of floor damage depth and confined water elevation. Jiang and Song [12] analyzed the influence of deep mining

TABLE 3: Sort B_1 , B_2 , B_3 , and B_4 .

A - B	B_1	B_2	<i>B</i> ₃	B_4	W _i
b_1	1	1/4	1/3	2	0.1216
b_2	4	1	2	6	0.5069
b_3	3	1/2	1	4	0.2987
b_4	1/2	1/6	1/4	1	0.0729

TABLE 4: B_1 layer index list sorting.

B_1	C_1	<i>C</i> ₂	<i>C</i> ₃	C_4
C_1	1	1/3	5	0.2828
C_2	3	1	7	0.6434
C_3	1/5	1/7	1	0.0738

TABLE 5: B_2 layer index list sorting.

<i>B</i> ₂	C_4	C_5	W _i
C_4	1	1/2	0.3333
C_5	2	1	0.6667

TABLE 6: B_3 layer index list sorting.

<i>B</i> ₃	C_6	<i>C</i> ₇	C_8	C_9	C_{10}	W_{i}
<i>C</i> ₆	1	1	1/6	1/4	1/3	0.0619
C_7	1	1	1/6	1/4	1/3	0.0619
C_8	6	6	1	3	4	0.4777
C_9	4	4	1/3	1	1/3	0.1792
C_{10}	3	3	1/4	3	1	0.0194

production on water inrush from the roof and floor of the erosion rock and pointed out that the deformation and failure of the erosion rock roof and floor caused by mining process would increase the connectivity of the floor. The erosion rock fracture surface was reactivated, and the thickness of the water barrier was weakened, resulting in the rupture of water from the roof and floor. Zhang et al. [13] studied the influence of water pressure on the water quantity of aquifer in deep rock and the water inrush of roof and floor of working face. It is found that pure water pressure is difficult to penetrate the complete thick rock to form water inrush, indicating that when the confined water is softened, dissolved, and eroded, it will have a great influence on the water resistance ability of the top and bottom water-resisting layer. Wang et al. [14–16] established a comprehensive evaluation system of surrounding rock instability and failure; obtained the space-time evolution characteristics of surrounding rock fracture field, stress field, and displacement field; and discussed the mechanical evolution law of roof fracture field, displacement field, and surrounding rock stress field in deep stope. The deformation characteristics and stress state of key block structure model before and after excavation are simu-

TABLE 7: B_4 layer index list sorting.

B_4	<i>C</i> ₁₁	<i>C</i> ₂	C ₁₃	W _i
<i>C</i> ₁₁	1	3	6	0.6393
C_{12}	1/3	1	4	0.2737
C ₁₃	1/6	1/4	1	0.0869

TABLE 8: Overall ranking of influencing factors.

	B_1 0.1216	<i>B</i> ₂ 0.5069	<i>B</i> ₃ 0.2987	$B_4 \\ 0.0729$	Weight	Rank
C_1	0.2828	_	_	_	0.0344	7
C_2	0.6434	_	_	_	0.0782	4
C_3	0.0738	_	_	_	0.0090	11
C_4	—	0.3333	—	—	0.1689	2
C_5	—	0.6667	—	—	0.3380	1
C_6	—	—	0.0619	—	0.0185	9
C_7	_	_	0.0619	_	0.0185	10
C_8	_	_	0.4777	_	0.1427	3
C_9	_	_	0.1792	_	0.0535	5
C_{10}	—	—	0.0194	—	0.0058	13
C_{11}	—	—	—	0.6393	0.0466	6
C_{12}	_	_	_	0.2737	0.0200	8
<i>C</i> ₁₃	—	—	—	0.0869	0.0063	12

lated and analyzed by FLAC3D, and the mechanical properties and instability mechanism of key block failure under three spatial structure modes are obtained. Using the similar material model, the structural mechanical properties of the key block model on the top of the roadway before and after excavation and the temporal and spatial evolution law of failure deformation were studied. Li et al. [17] analyzed the movement law and stress evolution characteristics of rock structure in step area of mining face in Yangcheng Coal Mine and revealed the induced mechanism of rock burst from the perspective of dynamic load and energy release. Hao et al. [18] studied the restrictive conditions and mechanical roots of surrounding rock triggering dynamic disasters in isolated island working face and put forward the evaluation criteria of impact disaster prevention and control and safe and efficient mining.

At present, the understanding of deep mines is not as deep as that of shallow mines, there is an overall lack of engineering experience, and the deep theoretical system is not perfect. These problems have become an inevitable obstacle to deep mining. Therefore, the research on the evolution law of the deformation of deep stope coating rock becomes more and more urgent. In this paper, taking a deep mine under typical geological environment as the engineering background, according to the spatial evolution characteristics of rock deformation and failure in the mining process of the working face, the coupling evolution law of displacement field and stress field of rock in deep stope under the Geofluids

Classification	Thickness (m)	B (GPa)	$d (\text{kg/m}^3)$	C (MPa)	T (MPa)	f (°)	S (GPa)
Siltstone	8	6.54	2550	7.5	3.6	40	3.31
Coal seam	5	0.42	1300	2.4	2	18	0.13
Mudstone	25	2.45	3530	3.5	2.4	29	1.19
Gravel rock	30	2.48	2560	3.6	3	30	1.25
Sandy mudstone	6	2.53	2540	4.5	3.2	36	1.26
Fine sandstone	5	6.22	2520	7.2	3.7	38	3.48
Middle sandstone	10	7.56	2590	8.7	3.9	46	3.67

TABLE 9: Numerical simulation rock mechanics parameter table.



FIGURE 5: Vertical stress distribution diagram of YOZ surface during the advancing process of the working face.



FIGURE 6: Vertical stress distribution diagram of XOY face 5 m above the working face.

	-2.3960E + 06 -3.0000E + 06 -4.0000E + 06 -5.0000E + 06 -6.0000E + 06 -7.0000E + 06
	$\begin{array}{c} -3.0000\pm +00\\ -9.0000\pm +06\\ -1.0000\pm +07\\ -1.2000\pm +07\\ -1.3000\pm +07\\ -1.3712\pm +07\end{array}$

FIGURE 7: Vertical stress distribution diagram of XOY surface 20 m above the working surface.



FIGURE 8: Maximum vertical stress during advancing process at different overburden heights.



FIGURE 9: Horizontal stress distribution diagram of YOZ surface during the advancing process of working face.



FIGURE 10: XOY horizontal stress at 5 m above the working face.

combined action of high stress and strong disturbance is explored, which provides important reference for deep resource mining and stope surrounding rock prevention and control under complex conditions.

2. Analysis of Rock Collapse Characteristics in Deep Typical Geological Environment

2.1. Distribution Theory of "Three Zones" of Covering Rocks. In the process of coal seam mining, the overlying strata on the mining face are divided into three zones from the bottom up, namely, caving zone, fracture zone, and bending subsidence zone, as shown in Figure 1.

In general, for deep stope, with the development of coal seam mining, the strata in caving zone will collapse rapidly, and there will be occasional delay when caving. The height of the caving zone is an important index to measure the failure range and degree of stope rock. Through the statistics and mathematical analysis of the height of the caving zone in each deep mining area, the traditional calculation formula is modified and improved; the formula for calculating the height of the caving zone suitable for deep



FIGURE 11: Horizontal stress on XOY surface 20 m above the working surface.



FIGURE 12: Distribution diagram of minimum principal stress on YOZ surface during advancing process of the working face.



FIGURE 13: Minimum principal stress on XOY surface 5 m above the working surface.



FIGURE 14: Minimum principal stress of XOY surface 20 m above the working surface.



FIGURE 15: The maximum value of the minimum principal stress during the advancing process at different overburden heights.



FIGURE 16: Distribution diagram of maximum principal stress on YOZ surface during advancing process of the working face.



FIGURE 17: Maximum principal stress of XOY surface 5 m above the working surface.

mining area is [19]

$$H_1 = \frac{65\sum M}{2.1\sum M + 16} \pm 2.5. \tag{1}$$

In formula (1), H is the height of the caving zone (m) and M is the cumulative mining thickness (m).

The predicted mining thickness is 8 m, the height of the caving zone is 23.05~28.05 m, and the caving ratio is 2.88~3.51. When the caving ratio is less than 1.4, the rock



FIGURE 18: Maximum principal stress of XOY surface 20 m above the working surface.



FIGURE 19: The maximum value of the maximum principal stress during the advancing process at different overburden heights.



FIGURE 20: Distribution diagram of vertical displacement of YOZ surface during advancing process of the working face.

in the caving area cannot reach the filling effect of the goaf, the stratum pressure is strong, and the roof on the working surface is difficult to control, resulting in the pressure frame accident. With the increase of caving ratio, caving rock can effectively reduce the roof weighting strength and meet the roof control requirements of the mining face. For the rock stratum that breaks but does not collapse, it will squeeze together with the original rock, bite each other, form a masonry beam, and support the pressure of the upper coating rock. This pressure is transmitted to some supports

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FIGURE 21: The sinking amount of the roof at different positions of the working face along with the advancing process.

when breaking, which is called hinged beam transmission, which also has a great influence on the collapse of coating rock. Through the statistics and mathematical analysis of the height of fracture zone in each deep mining area, the traditional calculation formula is modified and improved. The calculation formula suitable for the height of the fracture zone in deep mining area is

$$H_2 = \frac{198\sum M}{1.6\sum M + 3.6} \pm 5.6.$$
(2)

The predicted mining thickness is 8 m, and the height of the fracture zone is 110.87 m~122.07 m.

2.2. Characteristics of Caving of Stope Rock. In the study of stope rock caving characteristics, it is based on the Pearson III formula method. The prediction formula of Pearson type III formula method in the main section of surrounding rock inclination is [20]

$$W_{(x)} = a_1 W_{\max} Z^{a_2} \exp(-a_3 Z),$$

$$Z = \frac{x}{L_{\alpha}},$$

$$L_{\alpha} = H_1(\cot \lambda_0 + \cot \beta_0) + \frac{M}{\sin \alpha},$$

$$W_{\max} = K_{\alpha} \frac{\Delta H}{\sqrt{H_0}} M \sqrt{n_3} \cos \alpha,$$

$$n_3 = \frac{k_3 l}{H_0}.$$
(3)

The formula of Pearson III formula method for the



FIGURE 22: Vertical displacement of *XOY* surface 5 m above the working surface.

probability integral method of the main section is

$$W_{(y)} = \frac{1}{2} W_{\max} \left[\operatorname{erf} \left(\frac{\sqrt{\pi}}{r} y \right) + 1 \right] - \frac{1}{2} W_{\max} \left\{ \operatorname{erf} \left[\frac{\sqrt{\pi}}{r} (y - D) \right] + 1 \right\},$$
$$r = \frac{H_1}{\tan \beta}.$$
(4)

The calculation formula of subsidence value at any point of overlying strata is

$$\begin{split} C_x &= \frac{W_{(x)}}{W_{\max}},\\ C_y &= \frac{W_{(y)}}{W_{\max}},\\ W_{(x,y)} &= C_x C_y W_{\max}. \end{split}$$
(5)

In the above formula, n_3 is toward mining degree coefficient; k_3 valued by rock type, hardness 0.7, medium hardness 0.8, and soft hardness 0.9; n_3 valued 1.0 when $n_3 > 1.0$; l is length of the working face (m); α is the dip angle of coal seam (°); M is coal seam thickness (m); W_{max} is maximum subsidence (m); H_1 is depth of the lower border (m); H_0 is average mining depth (m); ΔH is vertical height at a certain stage (m); λ_0 is bottom plate boundary angle (°); β_0 is roof boundary angle (°); and $W_{(x,y)}$ is the subsidence of any point in the weathered rock stratum (m).

According to the calculation formula and the threedimensional data format of goaf range unified rock, the simulated rock strata are substituted into the fit function in Matlab, and then, the surface fitting is carried out by using the function. Finally, the fitting result function is combined with the predicted parameters to calculate the roof collapse of the rock, and the three-dimensional and twodimensional plane maps of the roof subsidence value of the rock are drawn, as shown in Figures 2 and 3.

From Figure 3, it can be found that with the progress of mining in the working face, the roof of goaf gradually collapsed. Because the whole mining process was dynamic, the caving form of coating rock roof was basically consistent at each point. The height of coating rock roof caving increased with the progress of mining, and the caving rock mass showed a bowl shape as a whole. The maximum caving value was in the middle position from the mining starting



FIGURE 23: Vertical displacement of XOY surface 20 m above the working surface.

point to the mining reaching point. In the direction of the strike surface, the caving of coated rock roof is basically U-shaped. The simulation mining length is 100 m. It can be seen from the graph that the roof caving reaches the maximum value at 50 m of the goaf strike surface, and the maximum value is 1.16 m. In the direction along the dip, the rock collapse shows a wide U shape; the dip length is 40 and fixed. It can be found from the map that the roof collapses rapidly between 5 m in the front and back of the dip, reaching the maximum value, and the 10 m in the middle of the dip plane is basically maintained at the maximum.



FIGURE 24: Distribution of horizontal displacement of *YOZ* surface during advancement of the working face.

3. Analysis on Influencing Factors of Displacement Deformation of Deep Stope Rock

3.1. Influencing Factor Evaluation. There are many factors affecting the movement and deformation of cover rock, and there is a connection between these factors. To screen out the main influencing factors from many influencing factors, it is necessary to start with quantitative or qualitative analysis whether using traditional screening methods or fuzzy mathematics method. This paper takes the influencing factors of rock displacement in a deep mine in northwest as the research object and selects the fuzzy analytic hierarchy process to analyze the weight of the influencing factors, so as to obtain the proportion of the influencing factors of rock displacement in the coal mine.

The factors leading to displacement deformation of deep stope mainly include the following: hydrogeological environment, high ground stress, surrounding rock properties, and disturbance effect. The analytic hierarchy process (AHP) is used to divide the importance of event correlation for multifactors affecting rock stability, and the key influencing factors are screened out to provide reference for deep mine mining.

On the basis of the above analysis, establish the evaluation factor system as shown in Figure 4; the first layer is the evaluation target layer; the second layer is the middle layer, and the middle layer factor is the parent node; the third layer is the basic factor layer, and the basic factor is the refinement of the intermediate factor. The order of fuzzy comprehensive evaluation is from basic factors to intermediate factors, from bottom to top, and from the next layer to the top layer by layer calculation and analysis.

3.2. Building Hierarchical Progressive Structure Model. According to the evaluation factor structure and index system shown in Figure 4, a hierarchical structure model is established in Table 1.

3.3. Development of Judgment Matrix. The importance of each factor is obtained by constructing a judgment matrix. The judgment matrix is formed by comparing the factors that belong to the same layer of indicators.

The importance of index elements is scaled by a 9-level scaling method, as shown in Table 2.

Combined with engineering geological conditions and mathematical analysis, the following judgment matrix P is obtained:

$$P = \begin{bmatrix} 1 & \frac{1}{4} & \frac{1}{3} & 2\\ 4 & 1 & 2 & 6\\ 3 & \frac{1}{2} & 1 & 4\\ \frac{1}{2} & \frac{1}{6} & \frac{1}{4} & 1 \end{bmatrix}.$$
 (6)

A new judgment matrix P_1 can be obtained by



FIGURE 25: Shear stress distribution diagram of YOZ face during advancing process of working face.



FIGURE 26: XOY shear stress at 5 m above the working face.

normalizing matrix P.

$$P_{1} = \begin{bmatrix} \frac{2}{17} & \frac{3}{23} & \frac{4}{33} & \frac{2}{13} \\ \frac{8}{17} & \frac{12}{23} & \frac{24}{33} & \frac{6}{13} \\ \frac{6}{17} & \frac{6}{23} & \frac{12}{33} & \frac{4}{13} \\ \frac{1}{17} & \frac{2}{23} & \frac{3}{33} & \frac{1}{13} \end{bmatrix}.$$
 (7)

The judgment matrix after the above normalization is added by row.

$$W_i = [0.5231 \quad 2.1811 \quad 1.2851 \quad 0.3136].$$
 (8)

The normalized vector \overline{W}_i is shown as follows:

$$\begin{split} \bar{W}_i &= \begin{bmatrix} 0.5231 & 2.1811 & 1.2851 & 0.3136 \end{bmatrix}, \\ W_1 &= 0.1216, \\ W_2 &= 0.5609, \\ W_3 &= 0.2987, \\ W_4 &= 0.3136. \end{split} \tag{9}$$

The consistency test is carried out to determine that the maximum eigenvalue λ_{max} of the matrix is

$$\lambda \sum_{i=0}^{n} \frac{(PW_i)}{nW_i} \frac{1}{4} \frac{0.4937}{0.1216} \frac{2.0281}{0.5069} \frac{1.2086}{0.2987} \frac{0.2929}{0.0729} 4.0313_{\text{max}}.$$
 (10)

The consistency index C_I is

$$C_I = \frac{\lambda_{\max}}{n - 1(4.0313/(4 - 1))0.0104}.$$
 (11)

According to the random consistency index, the value of R_I is 0.90, and finally, the test coefficient C_R is

$$C_R = \frac{C_I}{R_I} = \frac{0.0104}{0.90} = 0.0116 < 0.10.$$
(12)

The judgment matrix is reasonable and consistent by calculation. Thus, the hierarchical single ordering of criterion layer B_1 , B_2 , B_3 , and B_4 is obtained, and the hierarchical single ordering of element *C* of each criterion layer is calculated.

According to Table 2, the judgment matrixes A-B, $B_1 - C$, $B_2 - C$, $B_3 - C$, and $B_4 - C$ are established, respectively, as shown in Tables 3–7.



FIGURE 27: XOY shear stress at 20 m above the working face.



FIGURE 28: Changes in the plastic zone of the YOZ surface during the advancement of the working face.

According to the relative weights between all decision layers and criterion layers, the total order of influencing factors can be obtained, such as Table 8.

3.4. Analysis of Evaluation Results. In the evaluation analysis of the influencing factors of deformation and failure of deep cover rock, it can be found that the internal stress change of rock has the greatest influence on the stability of rock, and its weight value is 0.5069, ranking first in the criterion layer. The weight value of rock structural stress is close to twice of mining stress, indicating that the stress environment of rock mass itself has the greatest influence on the stability of cover rock. With the deepening of mining depth, the horizontal stress increases continuously, and the original rock stress value of the whole buried rock increases accordingly, and the stability of buried rock decreases continuously. Because of this, the change of high geostress in deep buried rock will obviously affect the deformation of buried rock.

In addition to the stress suffered by the cover rock, the physical and mechanical properties of the surrounding rock itself have an overall impact on its failure in the second place. Among them, the cohesion has the greatest impact, with the weight value of 0.1427, followed by the internal friction angle, with the weight value of 0.0535. The deformation modulus, including elastic modulus and shear modulus, has relatively small control on its stability, but they also reach 0.0185. For deep cover rock, the cover rock is generally noncontinuous, so the mechanical properties of rock have a great influence on the strength of rock mass. It can be seen from the evaluation that the coating rock is most sensitive to the change of cohesion, and less sensitive to the deformation modulus.

Although the influence of hydrogeological environment on the deformation and failure of coated rock ranks third, the influence of groundwater seepage and gas pressure on the stability of coated rock ranks seventh and fourth, respectively, and the weight values are 0.0782 and 0.0334, indicating that the interference of hydrogeological environment is also particularly important for deep cover rock. Due to the ultradeep destruction of the coal seam floor and the effect of high Ordovician limestone water pressure in deep coal seam mining, there is no structural water inrush in the coal seam floor, which not only destroys the floor but also affects the stability of the coating rock due to the erosion and softening of the water on the cover rock. The influence of thermodynamic heat transfer on the stability of coated rock is the smallest, but it cannot be ignored.

The influence of disturbance on the stability of cover rock cannot be ignored, and the weight value in the criterion layer is 0.0729.

4. Spatiotemporal Evolution Law of Erosion Rocks in Deep Typical Geological Environment

4.1. Building Numerical Models. Taking a mine in northwest as the engineering geological background, FLAC3D software was used to study the failure law of surrounding rock plastic zone in the process of coal seam excavation in the working face of the mine. According to the geological conditions of the working face and the physical and mechanical parameters of the rock layer, a numerical model was established. The vertical height of the model is Z = 100 m, the strike length of the working face is X = 160 m, and the inclination length is Y = 100 m, which is composed of 560000 units.

In the process of mining, the mechanical properties of different strata are different. The general compressive

strength is higher than the tensile strength, and the stress change is nonlinear. Correct constitutive model should be selected in mining of deep strata. This time, Mohr-Coulomb plastic constitutive model and Mohr-Coulomb yield criterion are adopted. According to the data, the rock mechanical parameters of this numerical simulation are shown in Table 9.

Coal mining is considered a dynamic process. From the initial stage of mining to the final stable stope roof and roadway surrounding rock which are in dynamic changes, the changes of stress field and displacement field of working face are recorded. Based on the working face of the mine, the original data model was established. In order to prevent the boundary effect, the excavation roadway is 30 m away from the left and right boundary of the model as the protective coal pillar. The model is to be excavated five times, and the excavation distances are 20 m, 40 m, 60 m, 80 m, and 100 m, respectively. In order to collect data in the simulation process, three observation lines are set in the model. The first line is set along the strike direction in the middle of the working face, and the other two monitoring lines are set in the overlying mudstone at 5 m and 20 m from the roof of the working face.

The mining of deep coal seam makes the original stress redistribute, and the part where the stress decreases gradually becomes the pressure relief area, while the stress concentration occurs at the part where the stress increases. Through the establishment of the rock stratum model, the FIAC3D software was used to mine the model and study the evolution law of erosion rock movement. FALC3D is used to analyze the stress of deep coal seams with different mining depths, and the relevant stress at 20 m, 40 m, 60 m, 80 m, and 100 m can be obtained.

4.2. The Stress Evolution Law of Deep Rock

4.2.1. Vertical Pressure Distribution. The buried rock mass in the deep mine is in a natural equilibrium state without the influence of mining, and the deep mining activities will break the original equilibrium in an instant, resulting in the release of ground stress, which leads to the imbalance between the deformation of buried rock mass and the stability of free surface, and the stress near the buried rock mass is redistributed. The vertical stress distribution of surrounding rock above the working face after coal mining is shown in Figures 5–8.

The vertical stress distribution Figures 5–8 above the working face show that with the advancement of the working face, the stress concentration area first appears on both sides of the left and right sides, and then, the stress concentration area gradually moves forward and backward on both sides. The formation of the overlying strata in the whole goaf is proportional to the tensile stress area of the working face. The maximum vertical stress values of 5 m and 20 m above the working face at 100 m are 18.87 MPa and 13.71 MPa, respectively. By comparing the analysis figures of the two positions, it can be seen that the farther away from the working face, the faster the transfer speed of the stress concentration point. The maximum stress value decreases slowly with

the increase of the height of the intrusive rock, and the range of the middle tensile stress zone decreases slowly with the increase of the height.

4.2.2. Horizontal Stress Distribution. The horizontal stress distribution of the overlying strata is shown in Figures 9-11. At 5 m away from the working face, with the progress of the working face, the first stress was mainly concentrated on the excavation side, and the stress at the other end gradually increased. After a certain distance of excavation, the stress values at both ends were similar. At 20 meters away from the working face, unlike below, the initial stress is concentrated above the central goaf. With the advancing process, the stress moves forward and backward on both sides, and the tensile stress zone gradually forms in the central area, and the stress concentration zone forms in the area where the stress value on both sides increases. For horizontal stress, the stress value on both sides of the goaf first increases and then decreases, showing a convex shape. When the working face advances to a certain progress, the stress reaches equilibrium.

The stress distribution map shows that after excavation, the stress changes continuously, and the stress distribution of the lower plate on the working face of the goaf is redistributed. With the increase of working face advancing distance, the vertical stress also increases, but it does not break its distribution law.

In the development of working face, the two-way stress has an increasing trend. The two-way stress peaked for the first time when the working face reached 60 m and reached the "encounter." At this time, the erosion rock activity intensified, and affected by the rock strata activity, the dynamic pressure continued to rise, gradually far exceeding the static pressure. Over the next time, the surrounding rock stress continues to increase.

4.2.3. Minimum Principal Stress and Maximum Principal Stress. By observing the three minimum principal stress diagrams from Figures 12-14, it can be found that during the mining process, the tensile stress area of the overlying strata above the goaf is rectangular, and the stress value gradually decreases from outside to inside. The minimum principal stress is mainly distributed around the overlying strata around the goaf. With the advancing process, the distribution area is mainly concentrated on the left and right sides of the goaf, and the stress value is also increasing. By comparing the minimum stress values at three locations from Figure 15, it is found that the minimum principal stress value around the working face is the largest, which is 25.23 MPa when advancing 100 m. In the overlying strata, the values at 5 m and 20 m above the working face are 19.38 MPa and 13.71 MPa, indicating that the stress value will gradually decrease with the increase of height.

According to the observation of the three maximum principal stresses from Figures 16–19, during the mining process, the tensile stress area of the overlying strata above the goaf is also rectangular, and the stress value gradually decreases from outside to inside. The higher the distance from the working face, the middle of the tensile stress zone gradually increases and finally tends to be stable. The maximum stress is mainly distributed in the peripheral corner of the goaf, and its value increases with the increase of the advancing distance. The maximum principal stress around the working face is 10.03 MPa, and the maximum stresses at 5 m and 20 m above the working face are 5.98 MPa and 3.95 MPa, respectively. It can be seen that the maximum principal stress also decreases with the increase of the coating rock height.

4.3. Evolution Law of Cover Rock Migration

4.3.1. Vertical Displacement. When the working face starts mining, the displacement of cover rock roof and floor is small. When the mining work continues to advance, the influence range of cover rock movement increases, and the vertical and horizontal movement also gradually increases. After the mining reaches a certain point, the vertical subsidence value of roof and floor tends to be stable, showing a symmetric model, and the vertical subsidence value of roof and floor reaches the maximum value in the distance center along the strike. According to the above distribution Figures 20–23, it can be seen that when the working face mining distance is from 20 m to 100 m, the vertical migration of the overlying rock roof increases continuously, from the initial 33.83 cm to 127.04 cm, and gradually tends to be stable. Similarly, the vertical displacement of the floor increases from 27.46 cm to 46.28 cm and then remains stable. It can be found that when reaching the "encounter" of the mining face, the vertical displacement of the floor reaches the maximum and tends to be stable, because with the continuation of mining, the caving rock in the goaf enters the pressure state, the roof is gradually compacted, and the maximum vertical displacement of the roof and floor is gradually reduced and tends to be stable. Finally, the mined-out area is gradually compacted, and the strata are expanded and squeezed to form cracks.

By observing and comparing the vertical displacement of 5 m and 20 m above the working face, it can be found that the vertical displacement of the square strata on the mining face is basically the same. With the advancement of the working face, the displacement increases gradually, and all reach the maximum displacement in the center, symmetrical, and bilateral, showing a "U" type. Through the simulation results, it can be seen that at 5 m above the working face, the vertical displacement of the grazing rock gradually increased from 28.89 cm to 122.60 cm, and at 20 m above the working face, the vertical displacement of the grazing rock gradually increased from 19.78 cm to 110.26 cm. It was found that the maximum displacement was 4.44 cm and 16.78 cm different from the roof ratio of the working face, respectively. The higher the distance between the grazing rock and the working face was, although the overall subsidence law was the same, the vertical displacement would gradually decrease. The farther the distance from the working face was, the smaller the subsidence was.

4.3.2. Horizontal Displacement. According to the analysis of Figure 24, in the horizontal direction, with the increase of

mining progress, the horizontal migration value in the middle of the goaf is 0, and the increase in the distance between the depressions on both sides of the goaf is approximately the same. At the beginning, the displacement increases continuously. After the working face advances a certain distance, the horizontal value-added amplitude becomes smaller and gradually tends to be the maximum.

4.4. Evolution Law of Fracture Field Failure in Cover Rock. It can be seen from the distribution Figures 25–28 of surrounding rock plastic zone after deep roadway excavation that shear-n and shear-p represent the shear stress, and shear-n represents the yield shear stress, which is converted from elastic deformation to plastic failure. Tension-n and tension-p represent tensile stress, and tension-n represents yield tensile stress, and rock mass will be subjected to tensile failure.

After coal mining, the overlying strata are affected by shear stress and shear stress and begin to deform and destroy. At this time, the shear stress has the maximum value and is mainly concentrated at both ends of the surrounding rock. In the process of working face mining, the failure height of the roof and the failure depth of the floor of the coal seam increase with the mining work and tend to be stable when mining reaches a certain point; since then, with the development of mining, the deformation of rock tends to be stable, but the scope of damage expands with the mining process.

5. Conclusions

- (1) The comprehensive evaluation system of the evolution of deep stope rock is constructed, which is composed of hydrogeological environment, high ground stress, surrounding rock properties, and disturbance factors. It is concluded that the change of high ground stress and surrounding rock properties of the rock have the most obvious influence on the evolution of deep stope rock. The weight of high ground stress on the evolution of rock is 0.5069, the weight of surrounding rock properties on the evolution of rock is 0.1427
- (2) The variation characteristics of coal wall stress in the goaf during mining are proved. During the mining process, the maximum stress appears near the coal wall on both sides of the goaf, and the maximum stress increases gradually with the increase of excavation process. However, after a certain mining distance, the variation range of rock stress near the coal wall is gradually stable with the mining process
- (3) The variation characteristics of vertical displacement of roof strata in the working face during mining are revealed. In the mining process, the vertical deformation of the surrounding rock of the working face is larger than that of the floor rock, and the subsidence value of the mined-out area increases gradually with the increase of the mining distance, and

the maximum subsidence value appears at the middle position of the mined-out area

- (4) In the process of working face mining, the plastic zone damage range of the roof and floor of the erosion rock continues to expand, and the expansion area and damage degree of the roof are significantly greater than those of the floor. With the continuous mining work, the failure range of the roof and floor increases gradually, but the failure depth has not changed much. At the same time, due to the change of vertical displacement of roof surrounding rock, cracks appear in the rock mass in the plastic zone, resulting in gas flow and emission, and the pore pressure of surrounding rock begins to release
- (5) By comparing the stress and displacement variation characteristics of overlying strata at different positions from the working face, it is found that the stress increase amplitude and subsidence amplitude of overlying strata are larger near the working face, and the farther away from the working face, the smaller the growth amplitude is, and the stress of surrounding rock shows a decreasing evolution law from near to far

Data Availability

The data used to support the findings of this study are included within the article.

Conflicts of Interest

The authors declare that they have no conflicts of interest regarding the publication of this paper.

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References

- Y. J. Chen, Y. Jing, and F. Q. Yin, "Study on multi-factor influence of rock fracture propagation," *Journal of Shandong Jianzhu University*, vol. 34, no. 4, pp. 32–37, 2019.
- [2] N. J. Ma, X. D. Zhan, and Z. Q. Zhao, "Stability analysis and control of panel in deep mining roadway," *Journal of China Coal Society*, vol. 40, no. 10, pp. 2287–2295, 2015.
- [3] R. L. Shan, Y. S. Xue, and Q. Zhang, "Time dependent damage constitutive model for dynamic failure of rock," *Chinese Journal of Rock Mechanics and Engineering*, vol. 22, no. 11, pp. 1771–1776, 2015.
- [4] Q. H. Qian, "The corresponding characteristic scientific phenomena of deep rock engineering and the definition of ' deep

'," Journal of Donghua Institute of Technology, vol. 27, no. 1, pp. 1–5, 2004.

- [5] B. H. Yao, H. F. Zhou, and L. Chen, "Simulation study on fracture development law of rock under repeated mining," *Journal of Mining and Safety Engineering*, vol. 27, no. 3, pp. 443–446, 2010.
- [6] J. Wang, D. Y. Zhu, and W. L. Gong, "Physical simulation experiment on rock movement law of automatic roadway formation under roof cutting and pressure relief," *Journal of Rock Mechanics* and Engineering, vol. 37, no. 11, pp. 2536–2547, 2018.
- [7] J. L. Xu, W. Qin, and D. Y. Xuan, "Cumulative effect of unloading expansion of mining-induced erosion rock," *Journal of China Coal Society*, vol. 45, no. 1, pp. 35–43, 2020.
- [8] Y. Pan, G. M. Zhao, and X. R. Meng, "Elastic-plastic analysis of roadway surrounding rock under non-uniform stress field," *Journal of China Coal Society*, vol. 36, no. 1, pp. 54–57, 2011.
- [9] R. A. Azim, "A poroelastic numerical model for simulation of hydraulic fracture propagation: application to Upper Safa formation-western desert-Egypt," *Petroleum Research*, vol. 5, no. 1, article S2096249519300651, pp. 39–51, 2020.
- [10] V. Valliappan, J. J. Remmers, A. Barnhoorn, and D. M. Smeulders, "A numerical study on the effect of anisotropy on hydraulic fractures," *Rock Mechanics and Rock Engineering*, vol. 52, no. 2, article 1362, pp. 591–609, 2019.
- [11] Q. H. Huang, "Study on water inrush mechanism of deep stope floor induced by roof caving dynamic load," *China University* of *Mining and Technology*, vol. 2018, no. 5, pp. 143–150, 2018.
- [12] J. Q. Jiang and Z. Q. Song, "Study on floor activity of mining face and its influence on water inrush," *Journal of Shandong Mining Institute*, vol. 6, no. 4, pp. 1–11, 1987.
- [13] H. R. Zhang, W. Q. Zhang, and X. L. Wen, "The role of karst confined water in the process of mine floor water inrush," *Chinese Journal of Geological Disasters and Prevention*, vol. 10, no. 1, pp. 88–92, 1999.
- [14] X. F. Wang, W. B. Luo, and Z. D. Wu, "Mechanical evolution law of macroscopic deformation and failure of surrounding rock in deep minning," *Modern Tunnelling Technology*, vol. 56, no. 1, pp. 99–106, 2019.
- [15] X. F. Wang, Y. He, and M. Y. Lu, "Study on deformation and failure characteristics of surrounding rock of deep roadway under excavation unloading disturbance," *China Safety Science*, vol. 31, no. 8, pp. 83–90, 2021.
- [16] X. Wang, M. Lu, Y. Gao, W. Luo, and W. Liu, "Structural mechanical characteristics and instability law of roof key block breaking in gob-side roadway," *Advances in Civil Engineering*, vol. 2020, Article ID 6682303, 12 pages, 2020.
- [17] Y. Y. Li, S. C. Zhang, and L. Q. Gao, "Mechanism of induced rock movement and prevention in bench area of unequal length working face," *Rock and Soil Mechanics*, vol. 37, no. 11, pp. 3283–3290, 2016.
- [18] J. Hao, M. W. Xu, and W. B. Wu, "Study on dynamic disaster prevention and control method of island working face," *China Earthquake Engineering Journal*, vol. 39, no. 5, pp. 976–980, 2017.
- [19] P. Xiang, L. H. Sun, H. G. Ji, Y. Gao, Y. J. Liu, and Y. F. Wu, "Dynamic distribution characteristics and determination method of caving zone in large mining height working face," *Journal of Mining & Safety Engineering*, vol. 34, no. 5, pp. 861–867, 2017.
- [20] B. Zhao, N. S. Liang, and C. Wu, "Mining subsidence prediction system of steeply inclined coal seam in mountainous area based on Matlab," *Metal Mine*, vol. 2020, no. 3, pp. 159–167, 2020.