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Research Article

Instability Mode and Control Technology of Surrounding Rock in Composite Roof Coal Roadway under Multiple Dynamic Pressure Disturbances

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In view of the problems of composite roof coal roadway under multiple dynamic pressure disturbances such as strong stratification, large deformation of the roadway surrounding rock, and poor safety and reliability, this paper analyzed the deformation and instability characteristics of composite roof coal roadway under different influencing factors using numerical simulation and expounded the stress distribution pattern of composite roof coal roadway under multiple dynamic pressure disturbances. The geological conditions of gas control roadway in the 22301 working face of Tunlan coal mine are taken as examples, and this study is carried out based on the analysis of roof instability conditions of composite roof coal roadway. The progressive crack development characteristics in composite roof coal roadway and the instability failure mode of surrounding rock under multiple dynamic pressure disturbances were revealed, and the control mechanism and support scheme were put forward. According to the result, it is difficult to form a stable bearing rock beam due to the occurrence characteristics of the weak interlayer of the composite roof. The deflection of the roadway roof and the extent of the plastic zone are negatively correlated with the distance from the soft rock layer to the roadway but positively correlated with the thickness of the soft rock layer. As the lateral pressure coefficient λ increases, the plastic zone of the roadway surrounding rock first decreases and then increases. With the increasing influence of dynamic pressure, the coal roadway with composite roof suffers from greater damage under the superposition of high ground stress, lateral abutment pressure, and advance abutment pressure, which is characterized by significant zonal failure of the roof and asymmetric failure of surrounding rock. The control method of high-strength thick anchorage + differential support can construct the thick layer stable rock beam of the roof and equivalent support of the side, realizing continuous stress transfer and geometrically coordinated deformation. The support scheme was finally applied to the on-site industrial test, and the mine pressure monitoring results showed that the deformation of the roadway surrounding rock was effectively controlled under the new support scheme.

1. Introduction

Affected by the formation age, the coal measure strata are mainly sedimentary strata, which determines the widespread existence of composite roof in the coal roadway. The composite roof coal roadway is subject to multiple dynamic pressure disturbances during the whole service period and is prone to uncoordinated deformation under multiple stresses or groundwater due to the large difference in rock properties [1–3]. It further induces stratification and even leads to overall roof rupture, support failure, and roof collapse, bringing a great challenge to roadway safety maintenance and control [4, 5]. Therefore, in order to ensure the safety and reliability of the roadway surrounding rock in the full service period, it is of great practical significance to reveal the laws of instability and failure of the composite roof coal roadway under multiple dynamic pressure disturbances and to design the parameters of the support scheme according to local conditions.

A shared consensus in the academic community is that the composite roof is a multilayer structure of soft rock layers and hard rock layers mixed with each other, of which the former is characterized by low strength, joint development, and penetration of cracks. Firstly, due to the difference in mechanical properties, the soft rock layer and hard rock layer are prone to uncoordinated deformation when the roadway is deformed under pressure, resulting in roof separation [6–10]; in addition, the dynamic pressure disturbances during the full service cycle of the roadway will lead to continuous loading and unloading of the internal stress of the roadway roof structure, which exacerbates the deformation of the soft rock strata inside, resulting in serious roadway deformation. Moreover, multiple maintenance is needed during use, which not only threatens coal safety production but also greatly increases the safety cost [11–13]. According to the comprehensive analysis of roof failure characteristics and field application, one of the reasons for the instability of the composite roof of rectangular coal roadway is the compression failure or shear failure of the anchored rock stratum [14]. Secondly, other factors such as excavation disturbance stress, self-weight stress, tectonic stress, groundwater, weak rock formation, and rock formation interface have a greater impact on roof separation [15–17]. The difference in strength and stiffness between different rock strata in a composite roof is the main factor affecting the roof stability [18, 19]. The closer the soft rock layer is to the roadway surface, the greater the compressive stress on the roof rock layer will be [20]. Horizontal compressive stress is the dominant factor leading to stratification and bending failure of composite roof [21]. Under the action of horizontal stress, the buckling failure or shear failure of composite roof can be determined according to the relationship between the thickness-span ratio and critical stress, and the failure process of composite roof dynamic pressure roadway is summarized as structural load adjustment ⟶ structural stiffness weakening ⟶ structural instability (or stability) [22]. According to related studies on failure mechanism, scholars believed that it is feasible to use the anchor net beam support for the composite roof coal roadway. To sum up, many scholars have analyzed the damage mechanism and control technology of surrounding rock of composite roof mining roadway from different angles, and the results have great practical significance. However, most of them focus on the evolution characteristics of delamination failure of composite roof coal roadway under specific stress environment; only few systematic studies focused on the failure characteristics of composite roof rock beams with different structures and the entire failure mode and control of composite roof coal roadway under multiple dynamic pressure disturbances during the whole service period.

Therefore, taking the typical composite roof coal roadway in Tunlan Mine under multiple dynamic pressure disturbances as the engineering background and based on the deflection formula of the roof rock beam, this paper systematically studied the instability law of coal roadway composite roof under the distribution and thickness of weak rock strata and different lateral pressure coefficients. In addition, the stress distribution law, progressive fracture development characteristics, and instability failure mode of surrounding rock under multiple dynamic pressure disturbances are further clarified. This paper put forward the control concept of high-strength thick anchorage of the roof + differential support of the side to improve the stress homogenization and deformation coordination of surrounding rock of composite roof coal roadway in the mining process. Moreover, the new support scheme is applied in the field, which effectively controls the deformation of the roadway surrounding rock. In summary, this paper provides the corresponding theoretical basis and technical reference for the stability control of composite roof coal roadway under multiple dynamic pressure disturbances.

2. Laws of Instability and Failure of Coal Roadway with Composite Roof under Multiple Dynamic Pressure Disturbances

2.1. Roof Instability Conditions of Composite Roof Coal Roadway. Here, the coal seam with composite roof is regarded as a beam model, and the roadway is assumed to be a symmetrical rectangle. When the roadway is excavated, both ends of the roof are fixed by surrounding rock, so it is regarded as a fixed beam [23–26]. Here, the thickness of the beam is h, the length is l, the span is l_1, the distance between the roadway side and the end of the beam is l_2, and the roadway height is b. With the roadway excavation, the roof stress is redistributed, the shallow stress of surrounding rock is partially released with the completion of deformation, and the stress gradually transfers to the deep. The simplified image of the roadway roof beam model is shown in Figure 1. Under the condition of uniform stress distribution, that is, $q_2(x) = q_1(l-x), q_4(x) = q_3(l-x)$, the deflection of roadway roof beam under load can be calculated by the following formula. When the distance from the left end of the beam is t, a small segment dt is taken at t, and the force of this segment is $q_1(t)\,dt$; the deflection generated by this small segment force is as follows shown in equations (1)–(7) [27]:

$$F(t) = \begin{cases} -\dfrac{q_1(t)(l-t)^2}{6EIl^2}\left(-\left(1+\dfrac{2t}{l}\right)x^3 + 3tx^2\right)dt, & (0 \le x \le t), \\ -\dfrac{q_1(t)t^2}{6EI}\left[\left(3-\dfrac{2t}{l}\right)\dfrac{x^3}{l^2} - 3\left(2-\dfrac{t}{l}\right)\dfrac{x^2}{l} + 3x - t\right]dt, & (t \le x \le l), \end{cases} \tag{1}$$

where E is the elastic modulus (GPa) and I is the moment of inertia of cross section to bending neutral axis (m^4).

Then,

$$F_1(t) = -\frac{q_1(t)(l-t)^2}{6EIl^2}\left(-\left(1+\frac{2t}{l}\right)x^3 + 3tx^2\right), \tag{2}$$

$$F_2(t) = -\frac{q_1(t)t^2}{6EI}\left[\left(3-\frac{2t}{l}\right)\frac{x^3}{l^2} - 3\left(2-\frac{t}{l}\right)\frac{x^2}{l} + 3x - t\right]. \tag{3}$$

Then, the deflection generated by $q_1(x)$ can be expressed

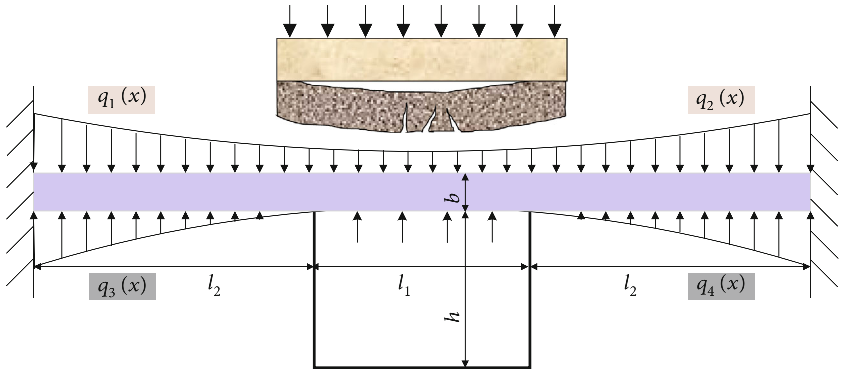


Figure 1: Stress diagram of roof rock beam.

by the following formula:

$$y_1 = \begin{cases} \int_0^x F_2(t)dt + \int_x^{l_2+l_1/2} F_1(t)dt, & \left(0 \le x \le l_2 + \dfrac{l_1}{2}\right), \\ \int_{l_2+l_1/2}^{l_2+l_1/2} F_2(t)dt, & \left(l_2 + \dfrac{l_1}{2} \le x \le l_1 + 2l_2\right). \end{cases} \tag{4}$$

Similarly, according to the thought of integral, the deflection generated by $q_2(x)$, $q_3(x)$, and $q_4(x)$ can be expressed, respectively, by the following formulas:

$$y_2 = \begin{cases} \int_{l_2+l_1/2}^{2l_2+l_1} F_3(t)dt, & \left(0 \le x \le l_2 + \dfrac{l_1}{2}\right), \\ \int_{l_2+l_1/2}^{x} F_4(t)dt + \int_x^{2l_2+l_1} F_3(t)dt, & \left(l_2 + \dfrac{l_1}{2} \le x \le l_1 + 2l_2\right), \end{cases} \tag{5}$$

$$y_3 = \begin{cases} \int_0^x F_6(t)dt + \int_x^{l_2} F_5(t)dt, & (0 \le x \le l_2), \\ \int_0^{l_2} F_6(t)dt, & (l_2 \le x \le l_1 + 2l_2), \end{cases} \tag{6}$$

$$y_4 = \begin{cases} \int_{l_1+l_2}^{2l_2+l_1} F_7(t)dt, & (0 \le x \le l_1 + l_2), \\ \int_{l_1+l_2}^{x} F_7(t)dt + \int_x^{2l_2+l_1} F_8(t)dt, & (l_1 + l_2 \le x \le l_1 + 2l_2). \end{cases} \tag{7}$$

Therefore, the deflection formula of the composite roof can be expressed as $y = y_1 + y_2 + y_3 + y_4$. The stability of the surrounding rock is closely related to the deflection of the roof. When the maximum tensile stress at the upper part exceeds the ultimate strength of the beam, the tensile failure will first occur on the roof surface, and then, the vertical cracks will further develop and expand to the deep, eventually resulting in the failure and instability of the beam and roof (see Figure 1 for the schematic diagram). Therefore, the initial stability of the roadway surrounding rock is determined by the strength of the roof rock beam and the stress distribution of the surrounding rock. It is difficult to form a stable rock beam structure due to multilayer occurrence, large differences in rock mass properties, and weak interlayer bonding force. Additionally, the poor stress environment of the roadway surrounding rock under multiple dynamic pressure disturbances results in maintenance and control difficulties. Therefore, the stability of composite roof with different structures, the stress distribution law, and the failure mode of surrounding rock under repeated mining disturbance will be further studied to lay a foundation for the control principles and support schemes.

2.2. Analysis on the Influencing Factors of Composite Roof Instability in Coal Roadway. As mentioned above, the composite roof is usually a combination of soft rock stratum and hard rock stratum in a staggered form. The spatial relationship and thickness change of the two affect the forms of roadway deformation and failure, and the horizontal stress is an essential factor inducing the separation and bending failure of rock strata. Accordingly, in this section, we select three factors, i.e., the location and distribution of weak rock stratum, the thickness of weak rock stratum, and lateral pressure coefficient, to carry out a numerical simulation analysis on the composite roof coal roadway. The FLAC3D numerical simulation software is adopted to analyze the roof deformation and plastic zone distribution of the composite roof coal roadway under the changes of the three factors. Generally, the total thickness of the composite roof is greater than 2 m, and the overall stability of the roadway depends on the stability of rock mass within half the roadway span above the roof according to the traditional theory. However, compared with the stable rock roadway, the loosening range of the composite roof coal roadway is relatively large. Therefore, in the numerical simulation model, the thickness of the roadway direct top is selected as 4 m, which is divided into five layers, including two soft layers and three hard layers. The roof above 4 m is composed of fine sandstone, siltstone, and medium sandstone, and the floor is composed of sandy mudstone, fine sandstone, siltstone, and medium sandstone; the model size is 50 ∗ 50 ∗ 10 m, the roadway section size is 5 ∗ 3.5 m, and the vertical stress at the top is 12.5 MPa. The simulation scheme of three factors is shown in Figure 2.

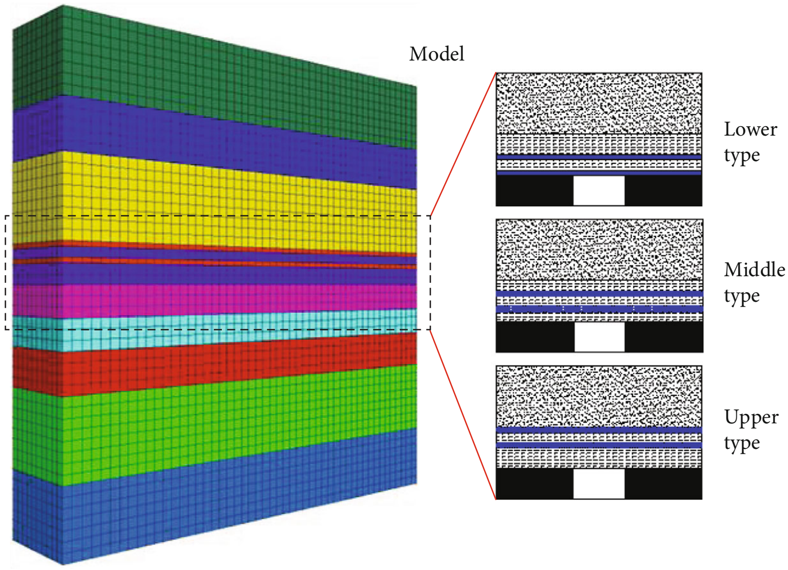


FIGURE 2: Composite roof models with different structures.

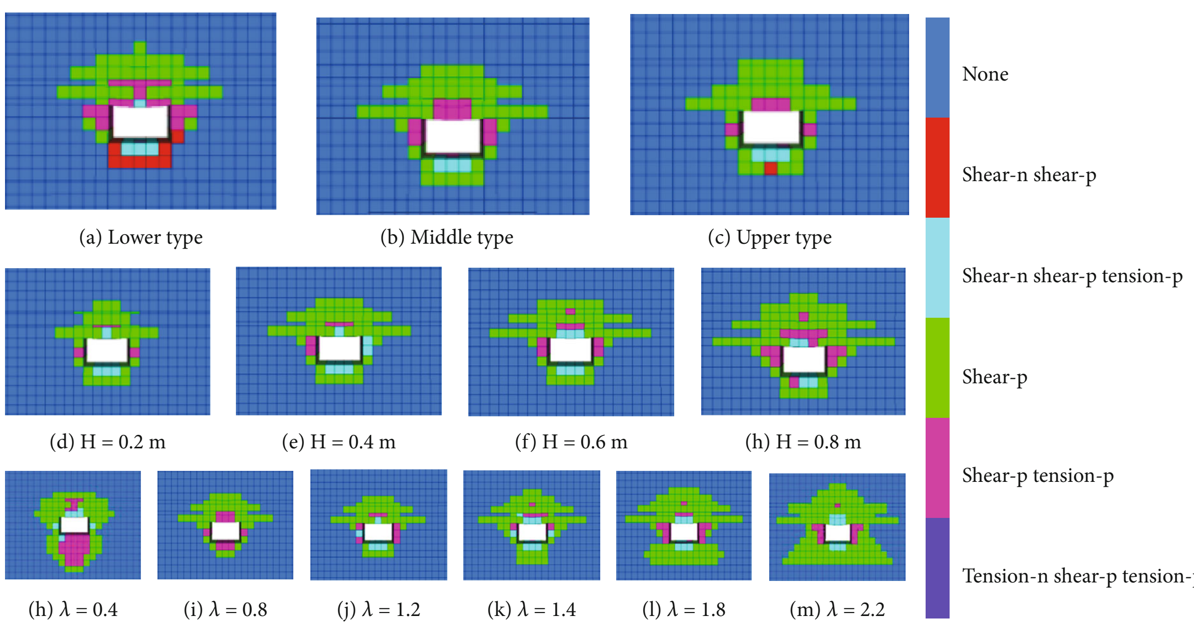
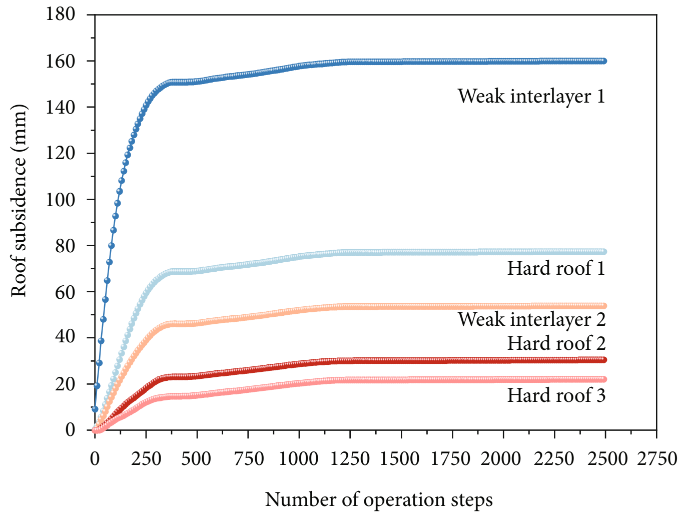


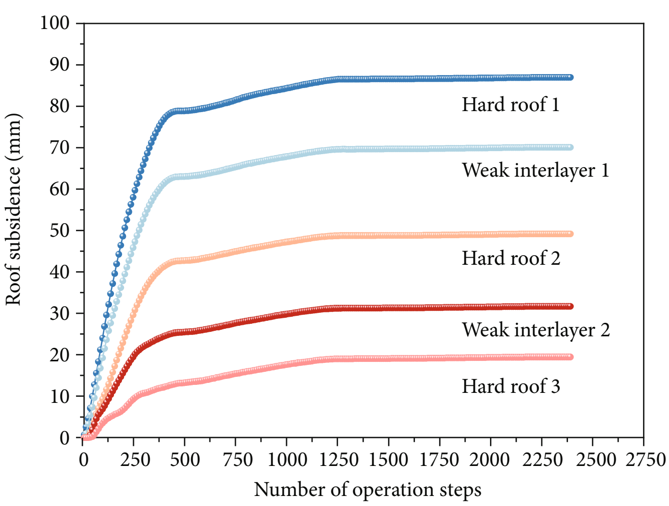
FIGURE 3: Distribution of plastic zone in coal roadway with composite roof under different influencing factors.

Here: (1) location of weak rock stratum: the thickness of weak rock stratum is 0.5 m; the thickness of hard rock stratum is 1 m; lateral pressure coefficient $\lambda = 1$. The spatial relationship is divided into lower, middle, and upper types. (2) Thickness of weak rock stratum: the roof rock structure is selected as the middle type, and the lateral pressure coefficient $\lambda = 1$. The thickness of weak rock stratum is 0.2 m, 0.4 m, 0.6 m, and 0.8 m, respectively; the corresponding thickness of hard rock stratum is 1.20 m, 1.06 m, 0.93 m, and 0.8 m, respectively. (3) Different lateral pressure coefficients λ: the roof rock structure is selected as the middle type; the thickness of weak rock layer is 0.5 m and that of hard rock layer is 1 m; the lateral pressure coefficient ranges from 0.4 to 2.2.

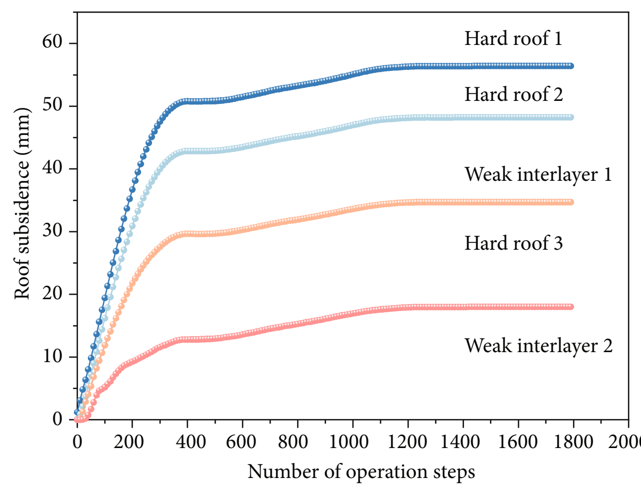
As shown in Figures 3 and 4, the plastic zone distribution is mainly concentrated in the surface surrounding rock of the roadway and the weak rock stratum of the roof. In specific, the plastic zone of the lower type composite roof roadway is the largest, which gradually expands to both sides. As the weak rock layer creeps up, the distribution range of the plastic zone is decreasing and the roof subsidence is also decreasing, indicating that the closer the weak rock stratum in the composite roof is to the roadway, the more likely it is to cause large deformation. On the other hand, as the weak rock layer creeps up, the deformation gap of each rock stratum in composite roof is also changing. In specific, the deformation gap between the lowest weak



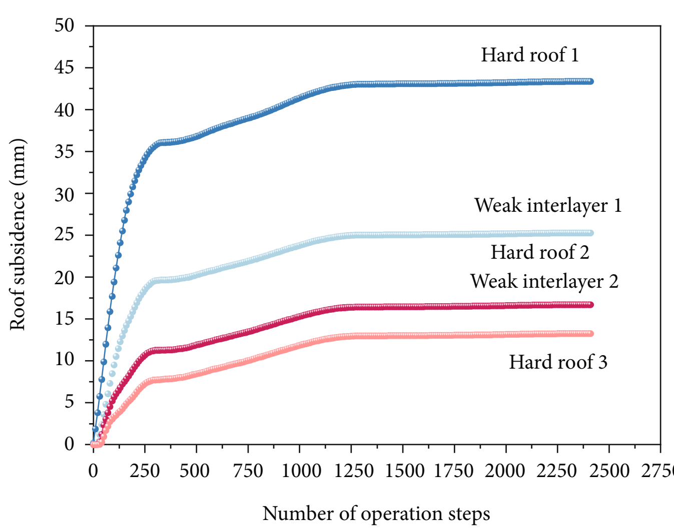
(a) Lower type



(b) Middle type

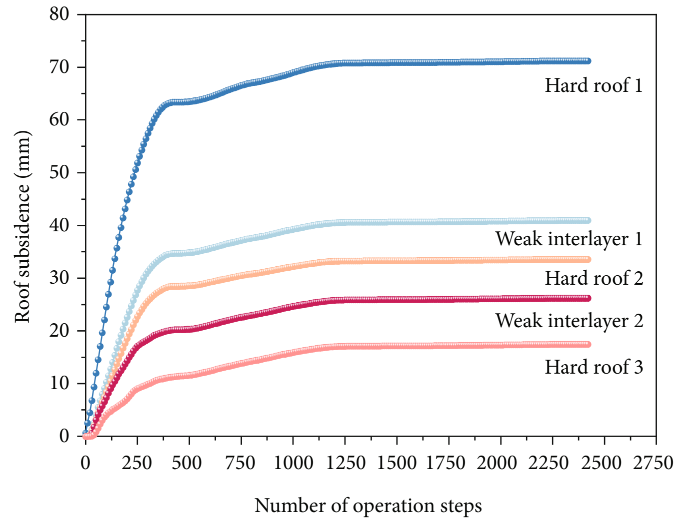


(c) Upper type

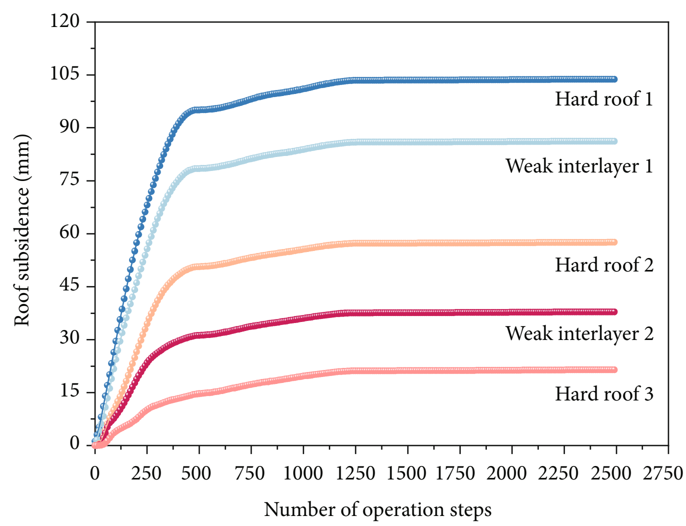


(d) $H = 0.2$ m

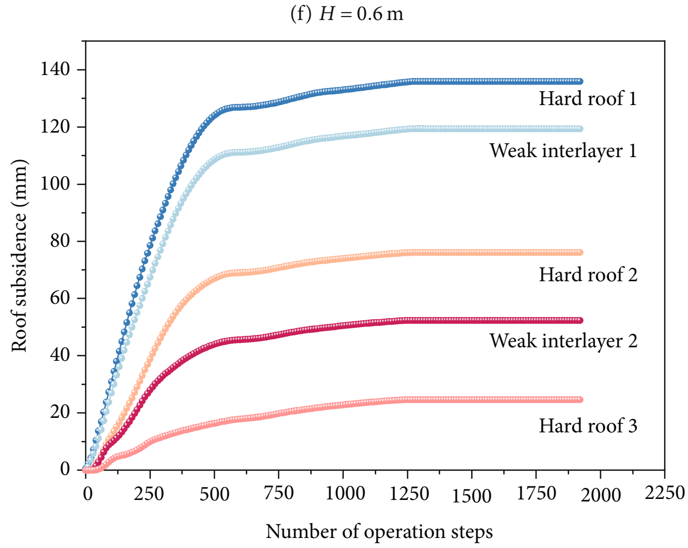
FIGURE 4: Continued.



(e) $H = 0.4$ m

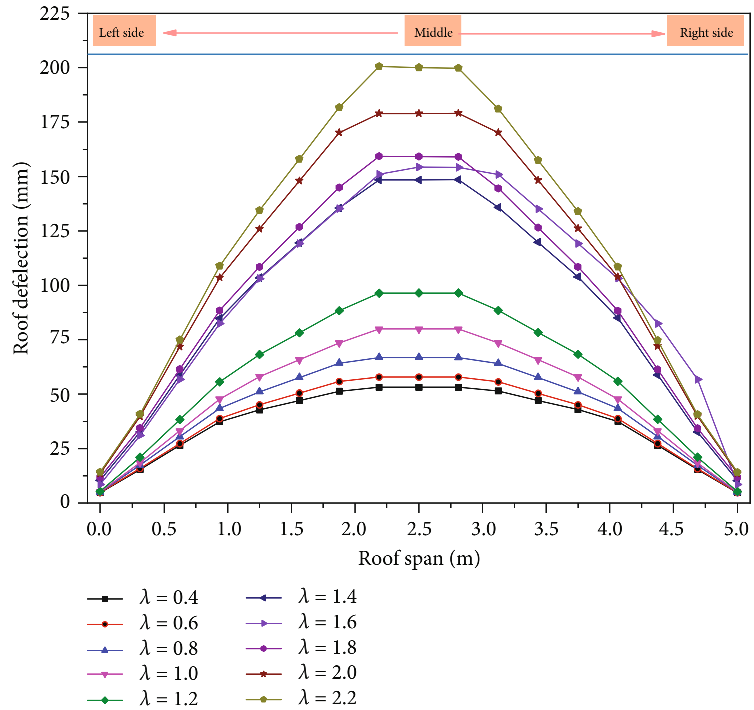


(f) $H = 0.6$ m



(g) $H = 0.8$ m

Figure 4: Continued.



(h) Different lateral pressure coefficient

Figure 4: Roof subsidence of compound roof coal roadway under different influencing factors.

rock stratum and the upper hard rock stratum is 83 mm in the lower type composite roof roadway, while the interlayer displacement gap between the middle and upper type composite roof roadway decreases gradually, indicating the existence of weak rock strata in composite roof greatly weakens the interlayer strength of the rock strata. The lower the weak rock stratum is, the more prone the interlayer uncoordinated deformation occurs, resulting in interlayer separation and obvious zoning characteristic.

The greater the thickness of weak rock stratum, the more serious the roof subsidence, the wider the distribution range of plastic zone, and the more serious the deformation of the roadway surrounding rock. When the thickness of the weak rock stratum is 0.2 m, the plastic zone of roadway roof extends obviously at the weak rock stratum, and the roof subsidence is about 45 mm. When the thickness of the weak rock stratum is 0.8 m, the plastic zone increases significantly, which has extended to 6-7 m above the roof; at the same time, there is a large deformation in the lower strata of roadway roof, and the roof deformation reaches about 140 mm, indicating that the deformation of surrounding rock and the size of plastic zone in the composite roof are positively correlated with the thickness of weak strata.

As the lateral pressure coefficient λ increases, the roof deflection increases, as shown in Figure 4. When $0.4 < \lambda < 1.2$, the roadway roof deflection is relatively small, and the growth rate is slow. When $\lambda > 1.2$, the roof deflection increases sharply; at this time, the roadway roof bears strong horizontal extrusion force, indicating that the bending force of the rock stratum exceeds its own bending capacity, thus the deflection increases sharply. Owing to the low cohesive force of composite roof and large difference in interlayer strength, when the horizontal extrusion force gradually increases, the weak rock stratum takes the lead in failure but it fails to reach the critical failure point of the hard rock stratum; thus, the uncoordinated deformation between the two causes interlayer separation, further leading to continuous stress transmission failure. The cracks in the weak rock stratum expand upward directly from the dominant direction of the tips on both sides, resulting in roadway instability. When $\lambda < 1$, the range of the plastic zone decreases with the increase of λ, and the horizontal stress is small. The horizontal tensile failure occurs in each rock stratum in the composite roof, especially the weak rock, under large vertical stress, resulting in the decrease of the overall bearing performance of the roadway surrounding rock and the increase of the roadway damage range. When the horizontal stress gradually approaches the vertical one, the overall stress becomes relatively uniform, in which the horizontal stress plays a particular role in clamping and balancing. When $\lambda > 1$, the larger λ means the larger range of plastic zone of the roadway surrounding rock, which indicates that when the lateral pressure coefficient increases to a certain value, the characteristics of large strength difference of each rock layer in composite roof coal roadway are gradually prominent. In this process, the weak rock layer will be further damaged with the plastic zone which will extend and expand along. Additionally, the increase in lateral pressure coefficient will lead to continuous plastic zone expansion of the bottom plate, especially in two bottom corners.

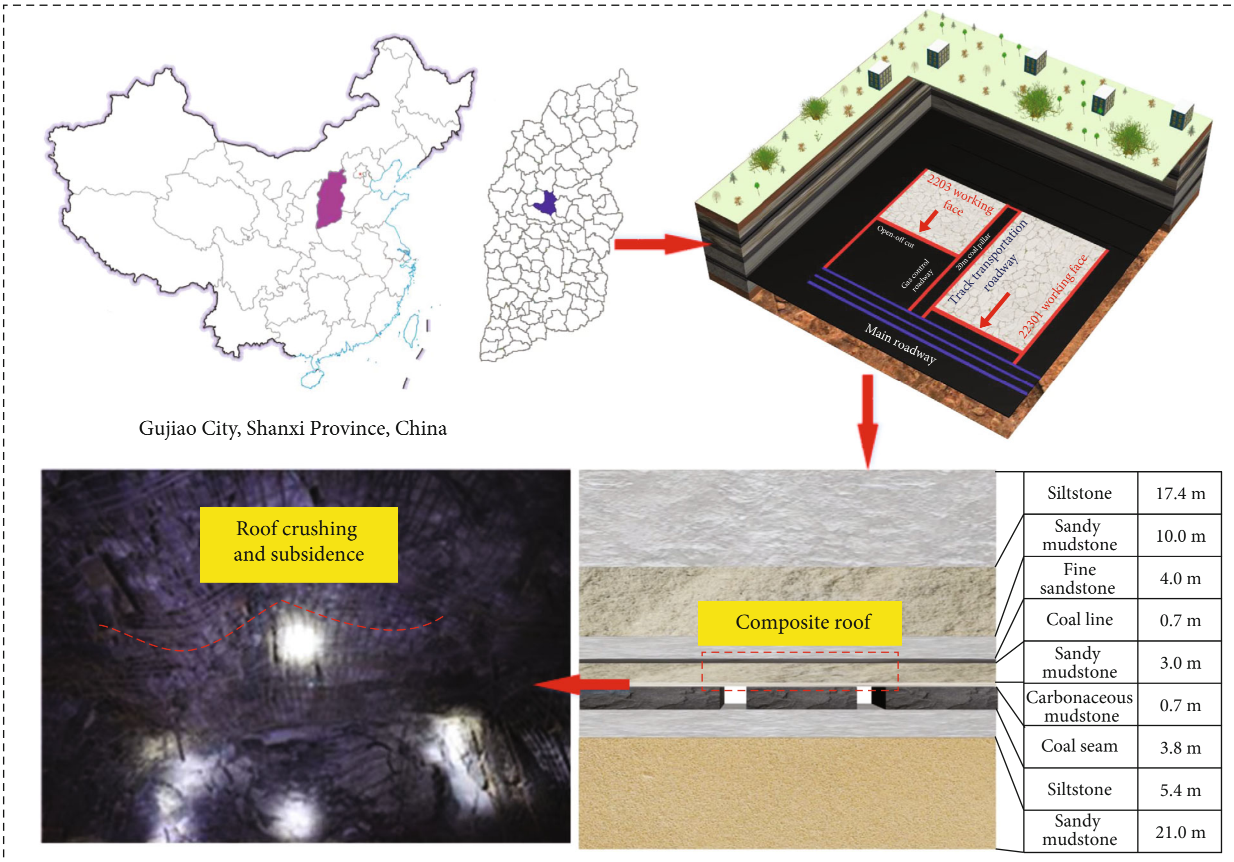


FIGURE 5: Schematic diagram of geological conditions of gas control roadway in 22301 working face.

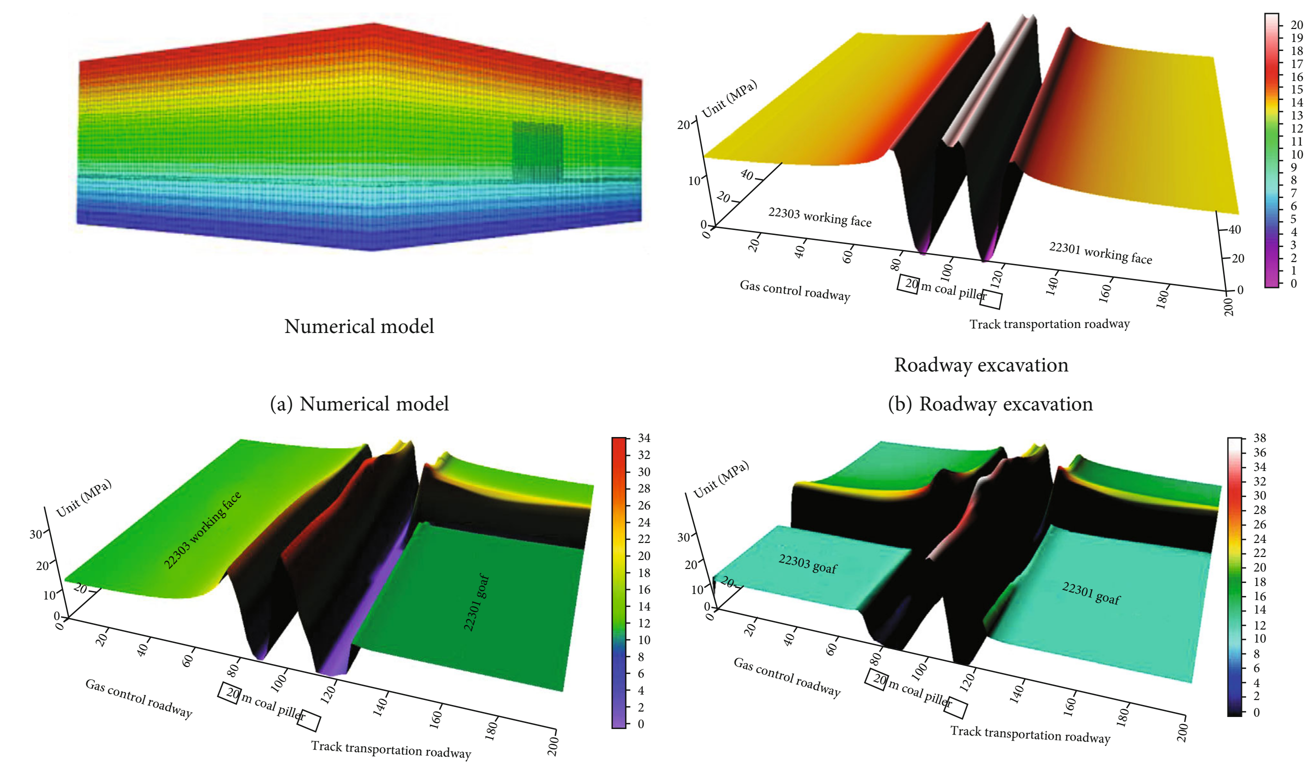
2.3. Stress Distribution and Failure of Surrounding Rock of Composite Roof Coal Roadway under Multiple Dynamic Pressure Disturbances. Located in the southwest of Gujiao City, Taiyuan, the Tunlan coal mine is a large state-owned coal mine affiliated to Shanxi Coking Coal Xishan Coal Power Co. Ltd., with an approved production capacity of 4.5 million tons/year. The roadway studied in this paper is the gas control roadway in the 22301 working face of Tunlan Mine, with a buried depth of 457-558 m. The roadway has a rectangular section, with a construction net width of 5.0 m, a net height of 3.5 m, a section area of 17.5 m^2, and a coal seam height of 3.8 m. It is excavated along the 2# coal roof, which is used for two purposes at the same time: to control the gas in the upper corner of the working face 22301, used as the belt trough of the working face 22303. The two working faces are basically at the same level. During the service period, it is subject to multiple dynamic pressure disturbances such as high ground stress, double roadway excavation, and mining of working face 22301 and 22303. The mine location, mining space, and occurrence state of roadway roof are shown in Figure 5.

Based on the numerical simulation on the failure of coal roadway with different composite roof structures, combined with the gas control roadway of 22301 working face in Tunlan Mine, we carried out the numerical simulation research on the stability of the roadway surrounding rock under three mining movements in the whole service cycle. There are two kinds of soft rock strata within 4 m of roadway roof: the thin carbonaceous mudstone and coal line. The soft rock stratum structure is a lower type, whose thickness is between 0.5 and 0.7 m, and the lateral pressure coefficient is 1. Based on the mining subsidence theory and the key layer theory, we established a numerical simulation model with a size of $200 * 160 * 66$ m, which adopts the Mohr-Coulomb yield criterion for model calculation. The top of the model adopts stress constraint with the top stress of 12.5 MPa (buried depth of 500 m); and both sides adopt displacement constraint. The parameters of rock stratum are shown in Table 1.

After excavating the two roadways, the original stress balance is broken, and a new equilibrium is formed after the stress redistribution. The vertical stress distribution of the working face and that around the roadway after three mining influences are shown in Figure 6. In FLAC3D numerical simulation, when the working face is excavated to stress balance, a large amount of stress concentration occurs in the lower left corner of the goaf due to boundary effect. This is inconsistent with the actual situation on site. Therefore, combined with the actual analysis, the vertical stress distribution above the goaf in Figure 6 is postprocessed to a certain extent. There is a pressure relief zone around the roadway, and stress concentration at a certain distance from the roadway surrounding the rock surface and the vertical stress is symmetrically distributed in the axial direction of the model, showing a reversed V-shaped curve. The “M”-type stress field is formed in the

Table 1: Rock stratum distribution and mechanical parameters.

Lithology	Thickness (m)	Density (kg·m^{-3})	Bulk modulus (GPa)	Shear modulus (GPa)	Internal friction angle (°)	Tensile strength (MPa)	Cohesion (MPa)
Siltstone	17.4	2600	9.4	3.4	36	5.78	1.7
Sandy mudstone	10.0	2300	6.0	2.2	32	2.36	1.2
Fine sandstone	4.0	2600	9.4	3.4	36	5.78	1.7
Coal line	0.7	1600	0.2	0.5	32	0.3	0.2
Sandy mudstone	3.0	2300	6.0	2.2	32	2.36	1.2
Carbonaceous mudstone	0.7	1700	0.2	0.5	32	0.3	0.2
2# coal	3.8	1300	2.0	1.2	30	1.08	0.8
Siltstone	5.4	2600	9.4	3.4	36	5.78	1.7
Sandy mudstone	21.0	2300	6.0	2.2	32	2.36	1.2



(a) Numerical model

(b) Roadway excavation

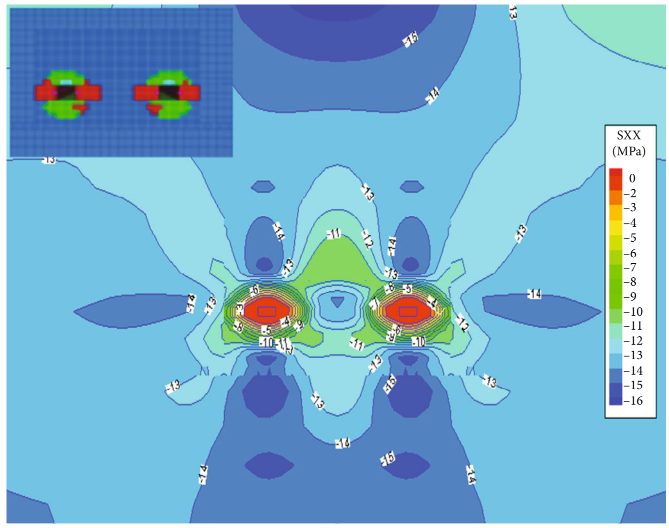
(c) 22301 working face mining

(d) 22303 working face mining

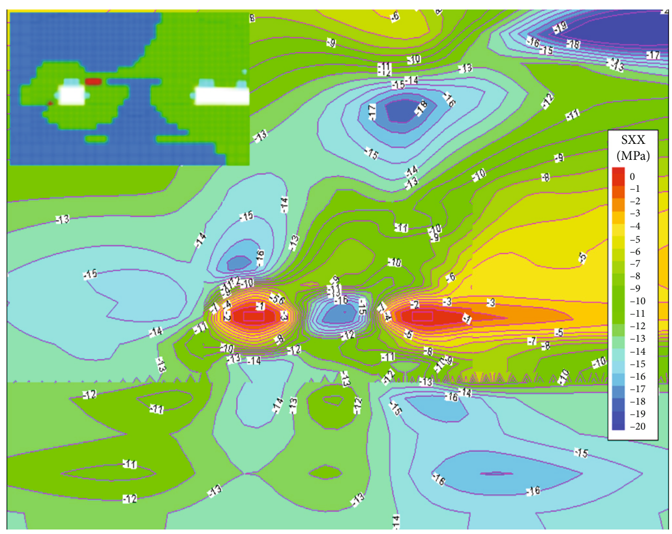
Figure 6: Numerical model of multiple dynamic pressure disturbances and vertical stress distribution.

middle of the two roadways (the coal pillar side wall). The stress concentration area formed by double roadway excavation is superimposed on the coal pillar side; thus, the vertical stress of the coal pillar side wall is slightly higher than that of the solid coal side wall. The maximum vertical stress in the coal pillar side wall is 23 MPa, and the stress concentration factor reaches 1.70. The overall strike length of the "M"-type stress field formed 20 m above the coal pillar is short, with a shape of an acute angle, which is because the two roadway stress curves intersect, with the high-stress curve covering the low-stress curve.

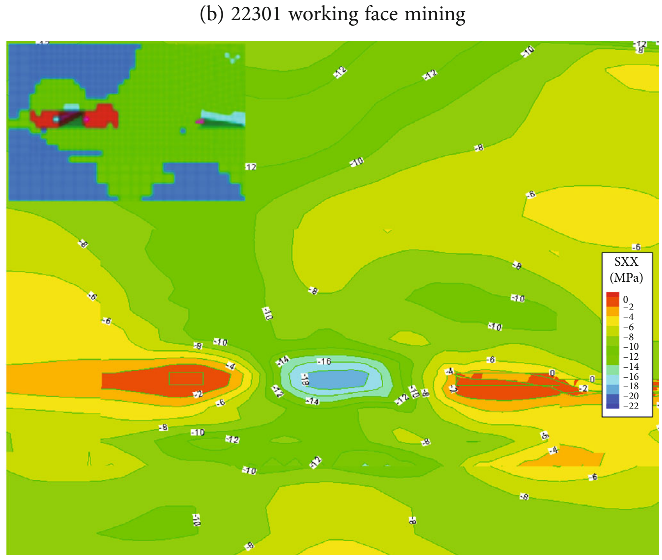
During the mining process of the 22301 working face, the gas control roadway roof and the solid coal side wall are mainly affected by the lateral abutment pressure caused by mining. In this stage, an "O-X" fracture will first occur in the goaf roof, forming an advanced abutment pressure zone, a lateral abutment pressure zone, and a lagging abutment pressure zone around the goaf. The stress will transfer during the sinking process of the goaf roof, and the stress concentration will occur in the advanced and the lateral abutment pressure zones. After the goaf roof sinks to full compaction, the stress in the bearing area of the lagging



(a) Double roadway excavation



(b) 22301 working face mining



(c) 22303 working face mining

Figure 7: Distribution of horizontal stress and plastic zone.

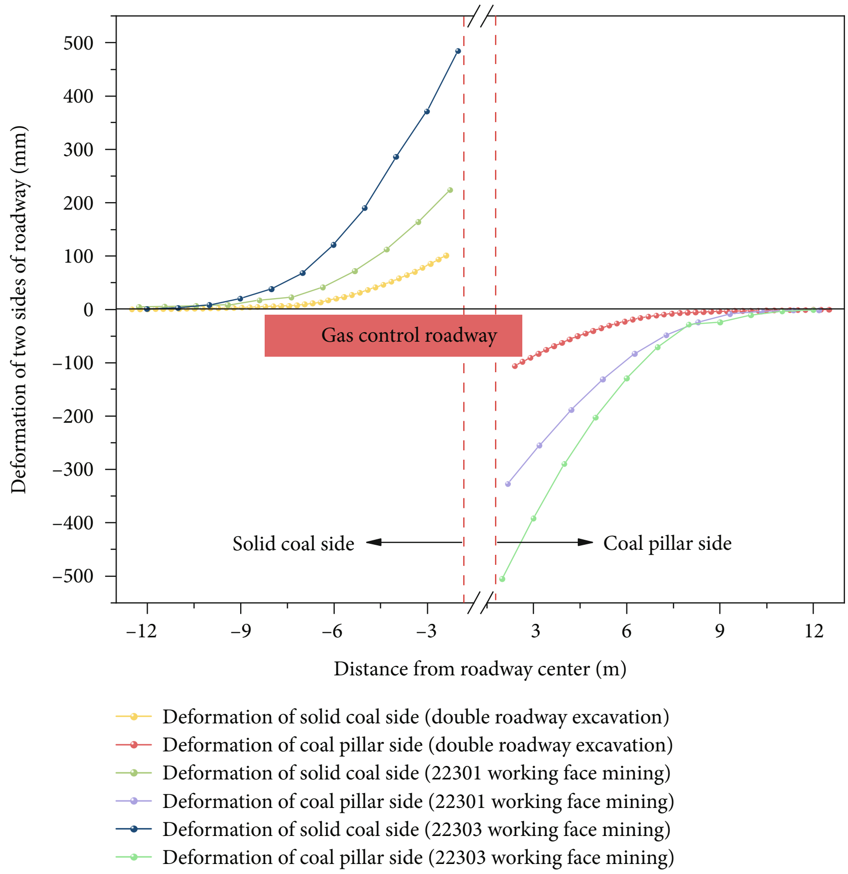
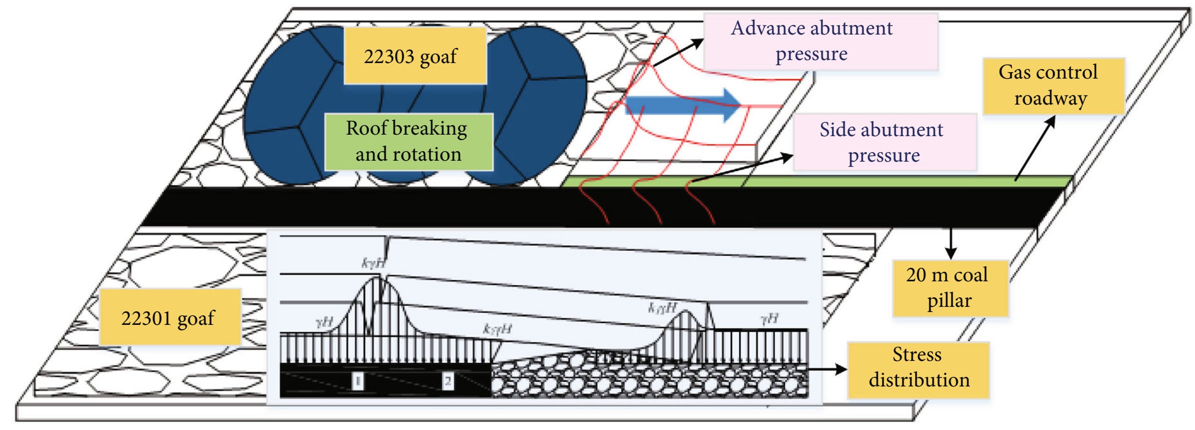


Figure 8: Deformation of two sides of coal roadway with composite roof under multiple dynamic pressure disturbances.

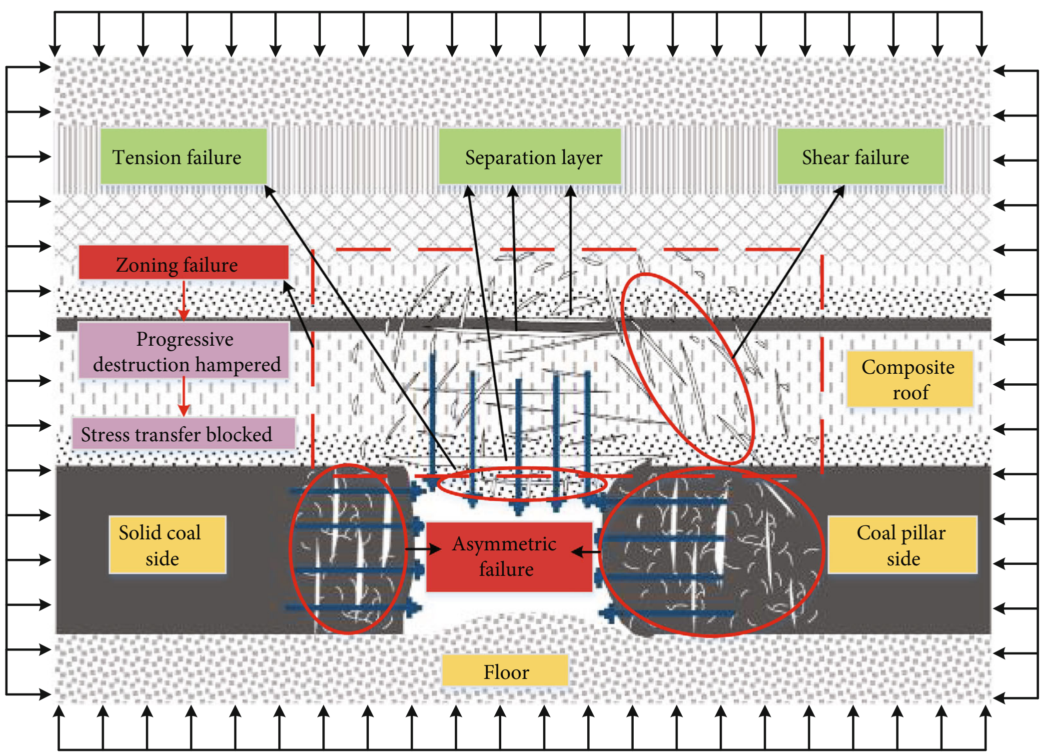
abutment pressure becomes smaller than the original rock stress. The vertical stress distribution of the coal seam will be affected by working face mining, and the vertical stress field shows the hump shape on "coal pillar-roadway-solid coal." The stress in the coal pillar side wall remains slightly higher than that in the solid coal side wall, and the stress near the working face is higher than that behind the working face. After the stress is rebalanced and stabilized, the vertical stress can be divided into low-stress area, high-stress area, and original rock stress area along the coal seam strike. The peak value of vertical stress is about 34 MPa and the stress concentration factor is 2.52.

After the mining of 22303 working face, the advance abutment pressure is formed in front of the working face, which will be superimposed to the coal pillar side with the lateral abutment pressure generated by the mining of 22301 working face. As shown in the figure, affected by the advanced abutment pressure of working face 22303, the stress of the coal pillar in front of the working face increases, while the stress behind the working face increases first and then decreases. On the one hand, because both sides of the coal pillar have been mined out, while the roof of the working face has not fully collapsed, so the coal pillar bears all the roof pressure, resulting in stress concentration; on the other hand, the continuous deformation of the coal pillar releases the stress behind the working face, until it becomes stable and close to the original rock stress. Compared with primary mining, the vertical stress continues to increase, with the peak of 38.3 MPa, and the stress concentration factor is 2.84.

The distributions of horizontal stress and plastic zone are shown in Figure 7. In the process of double roadway excavation, the stress distribution around the roadway is basically identical, and the surrounding rock is damaged under the action of stress. The brittleness of coal body makes it prone to slip and dislocation under disturbance stress, so there is mainly shear failure in the two sides. The plastic areas on both sides are within 4-5 m. The roof and floor of the roadway are subject to shear and tensile failure under disturbance stress. The weak rock stratum is first subject to tensile failure and then extends upward; the roof is subject to flexural deformation, and the bending moment in the middle is the largest. The horizontal stress of the roadway surrounding rock after primary mining is larger than that during excavation, and the value of horizontal stress is less than that of vertical stress, while the stress concentration is still high in the side slope of coal pillar, indicating that larger horizontal stress occurs in the stress concentration part in the process of working face mining, resulting in more serious deformation and damage of side slope of coal pillar compared with that of solid coal. To a certain extent, the horizontal stress is released due to the existence of goaf, and the maximum horizontal stress is about 17.8 MPa. The increase of horizontal stress leads to more serious bending and subsidence of composite roof. Affected by mining, the cracks further expand and the plastic



(a) Stress distribution of roadway surrounding rock



(b) Failure mode of roadway surrounding rock

Figure 9: Instability form of multiple dynamic pressure disturbance of surrounding rock in compound roof roadway.

zone further increases. In particular, the plastic zone of the coal pillar side wall has extended to 6 m inside the surrounding rock, resulting in a wedge-shaped failure zone. Compared with the side slope of coal pillar, the influence of mining on the side slope of solid coal is relatively mild. The roof is affected by horizontal shear stress, shear dislocation occurs in rock stratum, and the damage area extends to about 5 m above the roof. With the mining of the 22303 working face, the dynamic pressure disturbance has a more severe impact than previous mining, and the value and concentration of horizontal stress are also increasing, with the maximum horizontal stress of 22 MPa and the stress concentration factor of 1.55. The roadway roof and shoulder angle are more prone to shear failure under higher horizontal stress in composite roof. Under secondary mining influence, the failure range of surrounding rock plastic zone is wider; due to stress superposition, the plastic zone passes through the coal pillar, and the failure range is wider under shear stress; thus, the range of floor plastic zone is significantly expanded compared with the primary mining.

Under the disturbance of excavation stress, obvious displacement occurs in the roadway surrounding rock. As shown in Figure 8, the maximum subsidence of the roof is about 42 mm, and the maximum floor heave is about 38 mm. According to the distance of two sides of roadway from roadway center, the deformation of two sides is not much different under the disturbance of excavation stress. The maximum displacement of the coal pillar side is 82 mm and that of solid coal side is 80 mm.

After the mining of 22301 working face, the displacement of surrounding rock of gas control roadway increases significantly affected by the lateral abutment pressure, the total displacement of roof and floor reaches 567 mm, and there is obvious asymmetric deformation on both sides and roof. Under the influence of secondary mining, the square roof, floor, and two sides in front of the working face have undergone large deformation, which is large within 25 m in front of the working face. In specific, the maximum subsidence of the roof is 471 mm; the maximum displacement of the coal pillar side is 505 mm; and the maximum displacement of the solid coal side is 484 mm. The deformation of surrounding rock within 25-55 m in front of the working face is large and severe, and the deformation degree 55 m away is gradually alleviated.

3. Discussion

After roadway excavation, the stress state of roadway roof rock mass changes from three-dimensional compression equilibrium to two-way compression. The tensile stress first appears in the normal direction of roof, resulting in a tensile stress area; during the stress adjustment, the roadway surrounding rock deflects towards the free surface; when the rock tensile strength is less than the tensile stress, tensile cracks will occur in the roof rock, which will reduce the bearing capacity of surrounding rock; and when the surrounding rock stress exceeds the ultimate strength, the surrounding rock damage will occur and the stress will transfer to the deep. Accordingly, it can be seen that the surrounding rock is gradually damaged from shallow to deep, as described in the theory of primary and secondary bearing structure of the roadway surrounding rock [28, 29]; at this time, the strength of rock beam bearing structure formed by roadway roof itself is the basis of roadway stability. On the one hand, the upper part of composite roof coal roadway is composed of alternating soft-hard strata with poor integrity, making it difficult to form a stable roof rock beam; additionally, the interlayer bonding force is further weakened under the action of stress; in the process of progressive surrounding rock failure from surface to inside, the fracture of surrounding rock is hindered by the rock strata interface, which leads to inconsistent deformation in the deflection area between rock strata, resulting in delamination and zonal failure; similarly, in the process of three-dimensional stress recovery, the delamination will further block the stress transfer, resulting in the intermittent roof, and the stress continues to expand to the tip of the two sides of the delamination, which leads to the stress partition distribution. On the other hand, during the service period, due to the multiple dynamic pressures and the superimposition of the advance and lateral abutment pressure, the stress peak and concentration coefficient of the roadway surrounding rock increase significantly, deviatoric stress appears inside the surrounding rock due to the uneven stress distribution inside the roadway, the tensile cracks expands further, and the shear failure intensifies; at the same time, the shear failure of the left and right shoulder sockets of the roadway roof and the bottom corners of the floor is more serious due to stress concentration; with the aggravation of mining stress, obvious dislocation and frequent separation of layers occur in composite roof under the deviatoric stress and horizontal stress, and the support effect of two sides and the roof subsidence are more obvious, causing greater vertical shear effect on the coal and rock mass area of the roadway. As a result, the induced fracture range is further developed and expanded, the shoulder angles on both sides of the roof in the coal pillar and solid coal side wall show significant asymmetric failure characteristics, and there is even the structural failure under the overall extrusion of surrounding rock and anchor cable, as shown in Figure 9. Therefore, the key to realize the stability control of the roadway surrounding rock is to effectively suppress the zonal failure roof subsidence and asymmetric failure bulging of the surrounding rock of composite roof coal roadway under multiple dynamic pressure disturbances.

Accordingly, timely support is required after roadway excavation to provide and strengthen certain tensile resistance for the roadway surrounding rock, so as to prevent the expansion of tensile cracks [30]. However, the stress environment is poor in the coal roadway with multiple dynamic pressure disturbances, it is difficult to form a stable rock beam structure in the roof, and the bearing capacity of the shallow part of the roadway surrounding rock is poor. Under this condition, if the support resistance provided by the composite bearing small structure composed of shallow surrounding rock, bolt, anchor cable, etc. is large enough and multiple layers of composite roof can be connected in series into a stable high-strength bearing rock beam, then the shear and sliding strength of surrounding rock in the bearing area under this support condition is greater than the driving force of surrounding rock failure, stabilizing the roadway. While, after multiple dynamic pressure disturbances, the loosening range of composite roof coal roadway gradually increases and the partition failure of roof and asymmetric failure of slope worsened, the foundation anchorage length and anchorage strength of its anchor bolt are very important. Once the thickness of foundation anchorage constructed by support parameters is unreasonable after roadway excavation, it cannot effectively limit the long-term deformation of the surrounding rock of roadway with composite roof even if a high preload is applied. As a result, the crack development is prone to break through the anchoring circle constructed by the traditional short anchor, resulting in the overall deformation of the anchor cable and surrounding rock and the structural sliding of the shallow anchor due to the expansion and driving force.

On the one hand, the anchorage length should be able to pass through the partition failure surface of the roadway composite roof and be anchored in areas with low damage and small deformation of the rock mass, so as to eliminate the separation between partitions and realize stress homogenization and continuous stress transmission. The key factor to control the stability of the roadway surrounding rock is to inhibit the detachment and instability of lower anchorage structure. On the other hand, differential support should be adopted for the upper parts on both sides to balance and restrain asymmetric damage and achieve equivalent

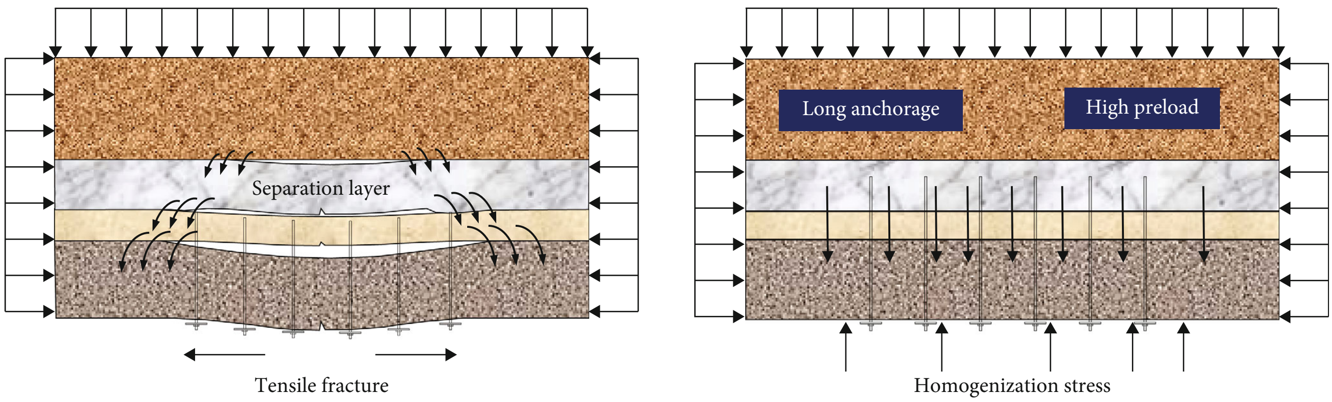


FIGURE 10: Control principle of high pretightening and long anchorage in coal roadway with composite roof.

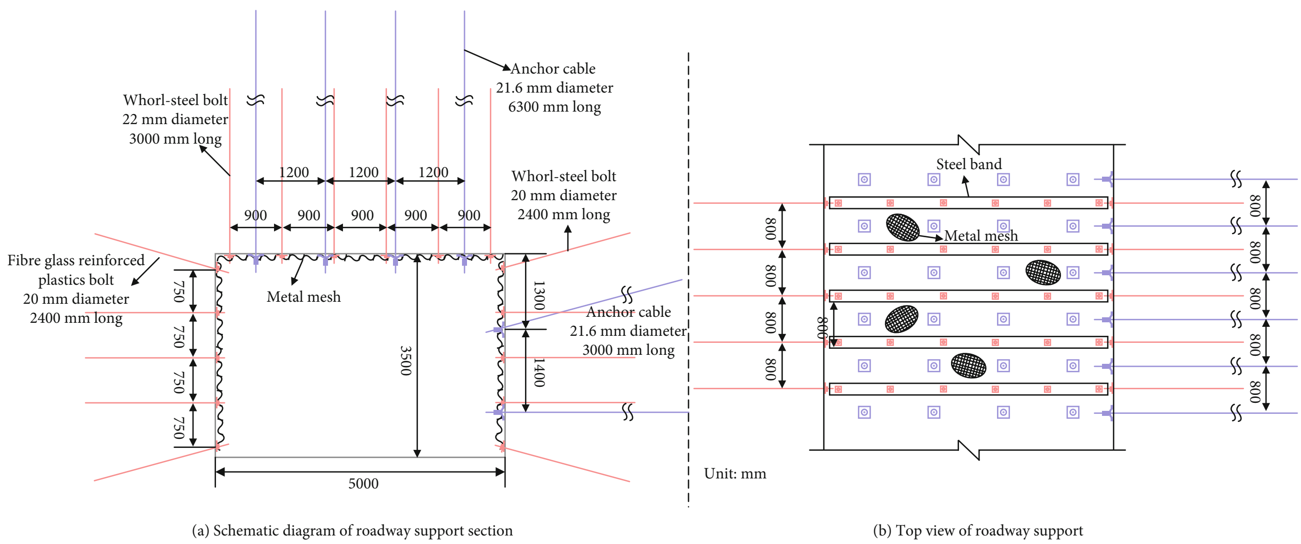


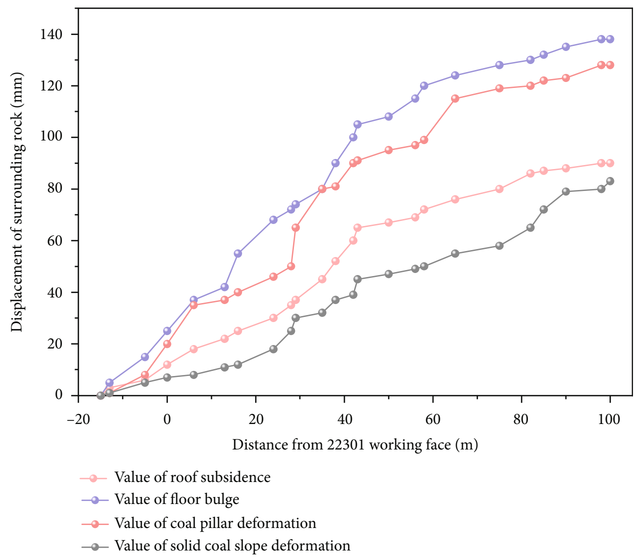
FIGURE 11: Roadway support design drawing.

support. With the increase of anchorage depth of high prestressed support into the surrounding rock, the bearing sensitivity of anchor rod to the relative displacement of surrounding rock is also strengthening. By effectively increasing the anchor thickness and preload of the foundation, a wider range of shallow rock mass can be effectively mobilized to construct the bearing structure, and the high preload enables the composite roof roadway to form a stable thick reinforced anchor rock beam in time. The large deformation of roadway surface is restrained by relatively small internal displacement, which realizes the coordinated deformation of composite roof and two sides, the linkage of internal and external rock mass and the rapid increase of resistance under the deformation trend of surrounding rock, and elimination of the discontinuous zonal failure and asymmetric failure of composite roof, so as to realize the stability control of the roadway surrounding rock and improve its ability to resist dynamic pressure disturbances. The specific structure is shown in Figure 10.

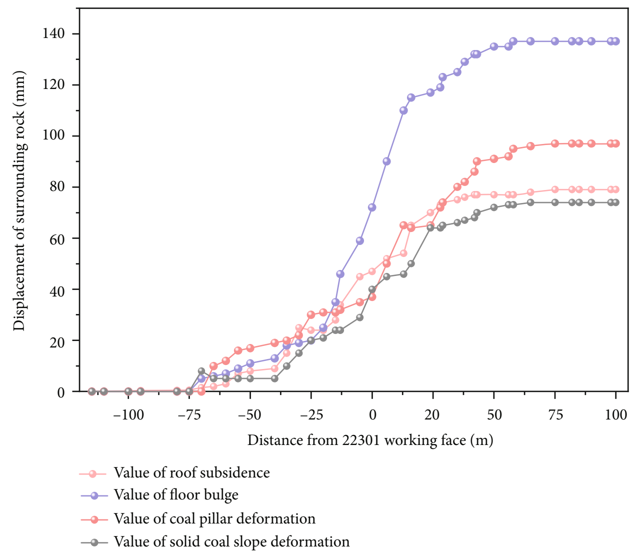
Based on the above analysis, in view of the serious zonal failure and asymmetric failure of composite roof under multiple dynamic pressure disturbances, the support design should follow the following principles: (1) the control principle of high-strength thick anchorage support is to take the first level anchorage of high-strength long anchor with high pretightening force as the core and complete the secondary reinforcement with the anchor rope; (2) asymmetric support principle: since the stress distribution and deformation of the upper part on both sides are asymmetric, differential support control shall be adopted to achieve equivalent support; (3) overall control principle and principle of improving residual strength of surrounding rock: under dynamic pressure disturbances, severe damage occurs in the roadway shoulder angle and some sections of surrounding rock, resulting in weak residual strength. Therefore, local strengthening support and grouting should be properly considered to improve the residual strength of rock mass and further realize the overall control of surrounding rock.

4. Engineering Application

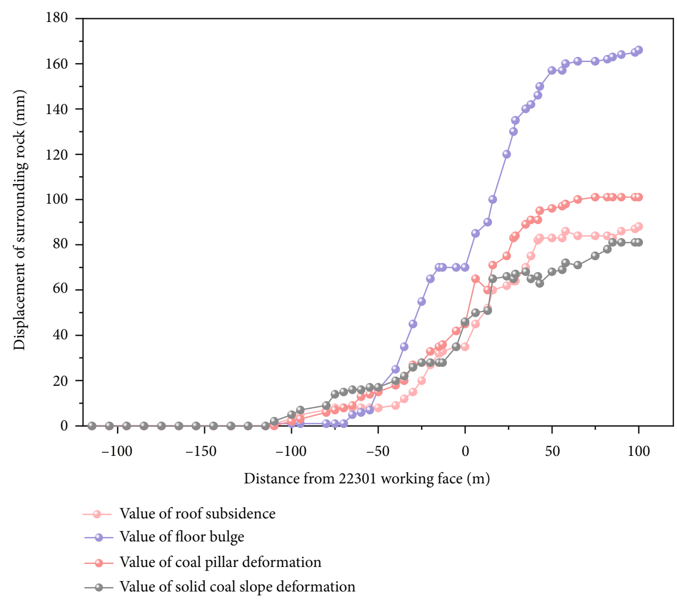
4.1. Support Scheme Design. Based on the analyses of stability, stress distribution, and deformation mode under



(a) 1# measuring point



(b) 2# measuring point



(c) 3# measuring point

Figure 12: Deformation of roadway surrounding rock.

dynamic pressure disturbances of gas control roadway with composite roof in the 22301 working face, combined with the engineering geological conditions of composite roof roadway, this paper designed the support scheme of gas control roadway in 22301 working face under multiple dynamic pressure disturbances and determined the coupling support system of "high-strength long bolt (high preload) + metal mesh + strong anchor cable + steel belt," to ensure the roadway stability control of the gas control roadway in the mining process. The specific support parameters are shown in Figure 11.

The roof adopts the combined support of anchor cable; there are 6 high-strength deformed steel bolts in each row, $\Phi 22 \times 3000$ mm with a spacing of 900 mm, row spacing 800 mm; there are five anchor cables in each row, with a row spacing of 800 mm, $\Phi 21.6 \times 6300$ mm. It adopts bolt (cable) pressing M5 steel belt and reinforcement mesh combined support. Each bolt is equipped with one $150 \times 150 \times 10$ mm outer square inner arch steel tray and one CK2380 resin roll; each anchor cable is equipped with three resin rolls: one CK2380 type and two Z2380 type, one 300 mm $\times$ 300 mm $\times$ 16 mm outer square inner arch steel tray and one K22 type lock. One Φ28 mm drill is equipped, anchor cable exposed 300 mm (tail to rock surface). What is more, the preload of bolt shall not be less than 150 kN, and the pretension of anchor cable shall not be less than 200 kN.

The anchor bolt of solid coal side wall adopts the $\Phi 20 \times 2400$ mm FRP anchor bolt, which is easy to be mined and cut in the working face; each row of 5 bolts is pressed with an M5 steel belt and 10# mining hexagonal woven metal mesh, with the spacing of 750 mm and row spacing of 800 mm; the anchor bolt of coal pillar side wall adopts

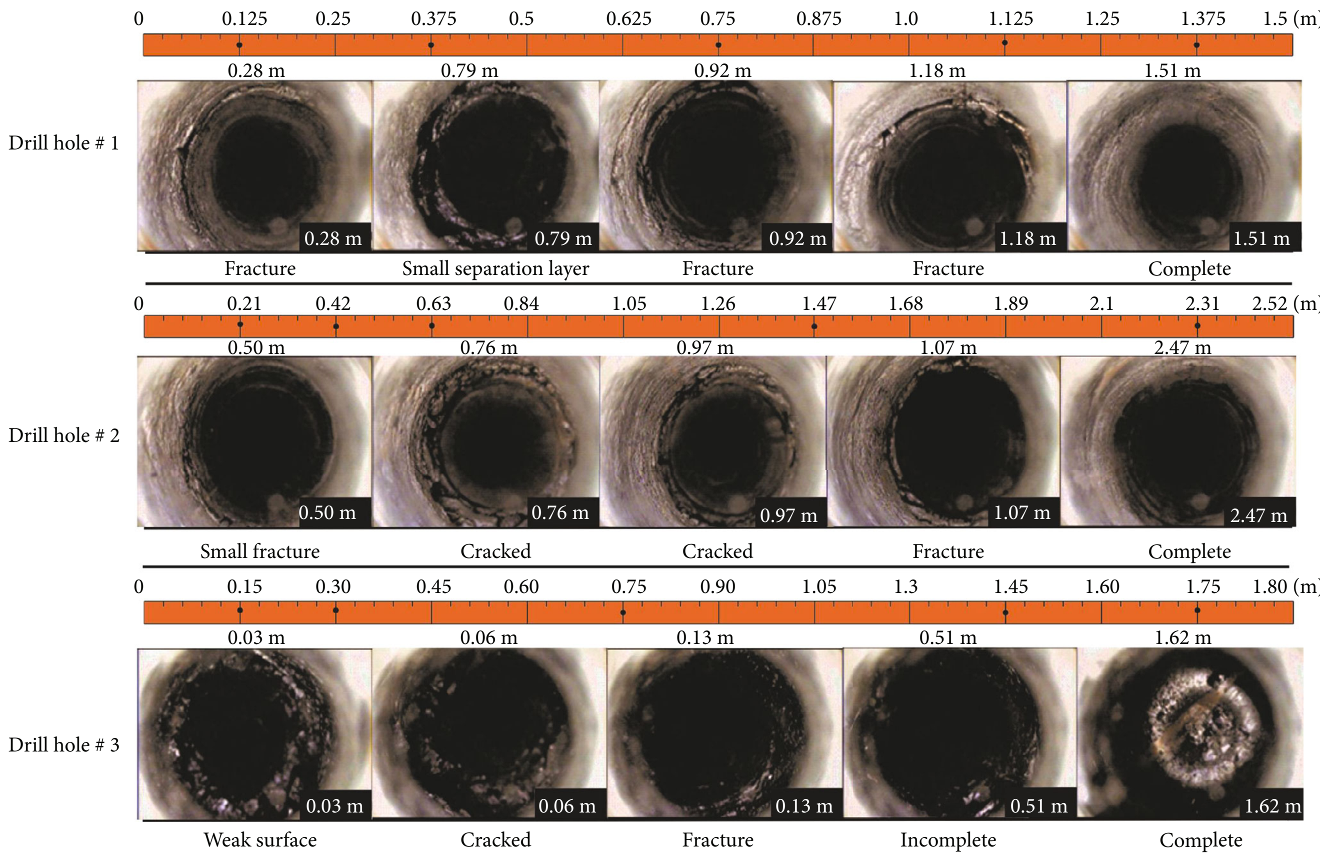


Figure 13: Fracture development of roadway surrounding rock.

the $\Phi 20 \times 2400$ mm threaded steel anchor bolt; each row of 5 anchors with steel mesh, the spacing is 750 mm, and the row spacing is 800 mm; each anchor is equipped with a 15 mm $\times$ 150 mm $\times$ 10 mm outer arch steel tray and a CK 2380 resin roll.

As for the supporting method of anchor cable, two anchor cables ($\Phi 21.6 \times 4300$ mm) are needed in each row of the coal pillar side wall, which are located at 1300 mm and 2700 mm below the roof in the middle of each two rows of anchor bolts along the excavation direction, with an angle of 30° above the slope, and the row spacing is 800 mm. Each anchor cable is equipped with a 300 mm $\times$ 300 mm $\times$ 16 mm outer arch steel tray, a CK2380 section and a Z2380 resin roll, and a KM22 lock. The pretension of anchor cable shall not be less than 200 kN.

4.2. Mine Strata Pressure Measured Results. We selected appropriate sections to arrange measuring stations to monitor the surface deformation of the roadway surrounding rock, among which 1#, 2#, and 3# measuring points are located at 15 m, 115 m, and 215 m in front of the 22301 working face, respectively. The deformation of the roadway surrounding rock from measuring point arrangement to the completion of monitoring is shown in Figure 12. The surrounding rock deformation of each measuring point shows obvious phased and continuous deformation characteristics. When the roadway is 50 m ahead of the working face, it is less affected by the mining disturbance stress of the working face, and the surrounding rock deformation is small; within the range from 50 m ahead to about 10 m beyond the working face, the deformation of the roadway surrounding rock increases gradually under the advance support stress, while the overburden and masonry beam above the working face have not started to rotate, and the overall deformation of the roadway has not increased significantly; when it is pushed over the range of 10-55 m, the rock mass above the working face rotates and the roof begins to collapse; the roadway surrounding rock at the measuring point begins to be affected by the large supporting pressure, with the deformation speed increasing suddenly; until the working face is pushed over the range of 55 m, the roof behind the goaf basically collapses and compacts, and the deformation speed of the roadway roof and two sides gradually decreases, but the continuous deformation still shows in the roadway floor within a certain distance.

According to the overall analysis, in the range from about 50 m ahead to about 55 m beyond the working face, the deformation of the roadway surrounding rock increases and presents a certain periodicity due to the abutment pressure generated by the working face mining. Although affected by the working face mining, under the reinforcement of the new support scheme, the total displacement of the roadway roof and floor is basically controlled at about 220 mm, and that of the two sides is controlled at about 190 mm, indicating that the optimized new support scheme has a good control effect on the surrounding rock of the composite roof coal roadway under the influence of mining, and the surrounding rock can basically remain stable.

The roof crack development is completed by peeping through boreholes, as shown in Figure 13. A total of 3 boreholes are peeped, numbered 1#, 2#, and 3#. Drilling peep of 1# is located 12 m away from the working face; drilling peep of 2# is located 30 m in front of the working face; drilling peep of 3# is located 40 m in front of the working face. There are a few cracks in the roadway roof where drill hole 1# is located, and the strata damage inside is shown in the figure. All separation and crushing areas are within 1.5 m, with positions of 0.28 m, 0.79 m, 0.92 m, and 1.18 m, respectively; the roadway roof is relatively flat where drill hole 2# is located, and the strata damage is shown in the figure. All separation layers and crushing areas appear within 1.1 m, with positions of 0.5 m, 0.76 m, 0.97 m, and 1.07 m, respectively. In drill hole 3#, the surface at the position of the sidewall drill is split and developed into small pieces or flakes. The strata damage in drill hole 3# is shown in the figure. The sidewall fracture occurs on the shallow surface within 0.13 m, with the positions of 0.03 m and 0.06 m, respectively. Due to the lack of construction conditions in the mine, only a single hydraulic drilling rig is used for construction, so the peeping effect is general. The following conclusions can be drawn from the analysis: (1) the separation and fracture of the roof only occur in the shallow part without developing to the deep part, which proves that the optimized support has a good effect on the control of roof subsidence and damage. (2) In the process of advancing the working face, the advance abutment pressure increases gradually, and the crack develops to the depth of 1.1-1.5 m, which can meet the support requirements.

5. Conclusion

(1) Based on the deflection formula of the roof rock beam, this paper analyzed the instability conditions of composite roof coal roadway after excavation, and the instability characteristics of composite roof coal roadway under different influencing factors were obtained using FLAC3D numerical simulation software. It is concluded that the initial stability of roadway is determined by the strength of the roof rock beam and the stress distribution of surrounding rock, and the occurrence characteristics of weak interbedding of composite roof determine that it is difficult to form a stable bearing rock beam. When the composite roof is stressed, the strata damage first occurs in the weak stratum, which is more prone to large deformation and separation than the ordinary one. The deflection and plastic zone of roadway roof are negatively correlated with the distance between the weak rock stratum and the roadway, while positively correlated with the thickness of the weak rock stratum. With the increase of the lateral pressure coefficient, the plastic zone of the roadway surrounding rock decreases first and then increases

(2) This paper expounded the stress distribution law, progressive crack development characteristics, and surrounding rock instability failure mode of composite roof coal roadway under multiple dynamic pressure disturbances. Taking the gas control roadway of 22301 working face of Tunlan coal mine in Shanxi Coking Coal Co. Ltd. as the research background, the analysis shows that its geological condition is a typical composite roof coal roadway, which is disturbed by multiple dynamic pressures including the roadway excavation and primary and secondary working face mining. With the increasing intensity of dynamic pressure disturbances, the high ground stress, excavation stress, lateral abutment pressure, and advance abutment pressure are superimposed, intensifying the damage degree of composite roof coal roadway. Thus, the cracks gradually develop from the tension splitting of the roof to deep shear sliding of the surrounding rock. Besides, there are frequently roof separation, discontinuous stress transmission, uncoordinated deformation, and obvious zonal failure. Moreover, under the induction of partial stress, the roadway surrounding rock is characterized by significant asymmetric failure

(3) This paper proposed the control concept of high-strength thick anchorage of the roof + differential support of the side. Firstly, the support scheme takes the primary foundation anchorage with high-strength long anchor bolt and high preload as the core; and complete the secondary reinforcement with the anchor rope, cooperate with the metal mesh and steel belt passed through the zonal failure surface of composite roof and anchored in the area with low damage and small deformation of the rock mass, so as to form the stable thick-layer reinforced anchored rock beams in time to eliminate the separation between zones, realizing stress homogenization and continuous stress transmission. Secondly, the coal pillar side slope is reinforced with anchor cables to restrain the asymmetric failure of surrounding rock. In addition, the corresponding support parameters are designed and applied to the on-site industrial test, which shows that under the influence of mining, the total displacement of roadway roof and floor is about 220 mm, and that of two sides is about 190 mm; the fracture development of surrounding rock only occurs in the shallow part, and the deformation of surrounding rock has been effectively controlled

Data Availability

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

Conflicts of Interest

The authors declare that there is no conflict of interest regarding the publication of this paper.

Acknowledgments

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