Roof deformation characteristics and experimental verification of advanced coupling support system supporting roadway

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Abstract
The advanced hydraulic support group combined with bolts (cables) constitutes the advanced coupling support system, which can ensure the stability of coal mine roadway safely and effectively. To explore the influence of the initial bearing capacity of the advanced hydraulic support on roadway roof deformation under anchor bar (cable) action. According to the actual geological conditions and roadway support of the 15106 working faces in Pingshuo Coal Mine, Yangquan, and Shanxi. Based on the change in the stress and displacement of the half-space body under normal force in elastic theory and combined with the linear superposition principle, the contact mechanics model of the advanced hydraulic support group and roadway roof under anchorage is established. Selecting the measured anchor bar (cable) preload in the coal mine as the input load of the model, the stress and displacement distribution of the roadway roof under different initial bearing capacities are calculated. The calculation results show that the stress and displacement of the roadway roof along the axis exhibit a “symmetrical distribution” and the whole roadway exhibits a “saddle surface” gradient distribution during the active support of advanced hydraulic support under anchorage. The increase in the initial bearing capacity of the advanced hydraulic support has a great influence on the support effect in the middle area of the roadway and has a relatively small influence on the support effect in the surrounding area of the roof. With an increase in an initial support capacity, the stress gradient changes violently but substantially less than it would under the crushing strength of the surrounding rock. Using the similar simulation experiment theory, a similar simulation experiment roadway was established at 1:40, and the simulation experiment of an advanced hydraulic support coupling active support under anchor protection was carried out. The maximum error percentage of the theoretical calculation and experimental results of roof
deformation under different initial bearing capacities is 19.51\%, and the maximum error percentage of stress is 13.77\%. The accuracy of the distribution law of roof stress and displacement under the initial bearing capacity in the process of active support obtained by theoretical calculation is verified. The research results provide a theoretical basis for advanced coupling adaptive support and the control of fully mechanized mining faces.

KEYWORDS
advanced hydraulic support, coupling support, graded distribution, initial bearing capacity, simulation experiment

1 | INTRODUCTION

With the continuous development of coal resources, the reserves of shallow coal resources have sharply decreased, and the shortage of shallow reserves has led to the development of deep coal mining.\(^1\) The geological conditions of the working face roadway in a thousand-meter-deep mine are complex, and the ground stress is high. Therefore, the influence of mining on the stability of roadway surrounding rock is intensified, the duration of influence increases, and the damage severity increases.\(^2\) The traditional advanced support methods include wooden supports, wood pallets, steel belts, and single hydraulic props with hinged beams. However, with an increase in the mining intensity, power, and volume of fully mechanized mining equipment and the roadway end face size, traditional advanced support methods cannot meet the needs of safe and efficient mining.\(^3\),\(^4\) In the past 10 years, the application of advanced hydraulic supports in the fully mechanized mining face has achieved good results and provides a guarantee for the safe and efficient mining of the fully mechanized face.

With the application of advanced hydraulic support and anchor bars (cables) in roadway support in coal mines, many domestic and international experts and scholars have investigated the characteristics of anchor bar (cable) support and the coupling support mechanism of an advanced hydraulic support for roadway surrounding rock. From the perspective of a pure anchor bar (cable) that supports roadway surrounding rock, Dudin et al.\(^5\) examined the technology and experience of using AB series anchor supports (modifications AB03) in a highly cracked coal seam roof mass in the zone of reference pressure from the face. Tan et al.\(^6\) designed a support system composed of foamed concrete and U-shaped steel pipes for large deformation tunnels in coal mines. Wu et al.\(^7\) conducted an anchoring experiment in steeply inclined geological formations, which provided a basis for the rational design of bolt support methods. Yu et al.\(^8\) used a combination of numerical simulation and orthogonal experiments to optimize the combined support parameters of soft-rock roadways. Wang et al.\(^9\) proposed a partitioned, anchor–grouting reinforcement technology for the deformation and failure mechanism of crossed roadways and conducted on-site monitoring of its support effect. Wang et al.\(^10\) discovered that the combined support system of additional energy-absorbing elements and bolt supports can improve the support performance of roadways in the case of rock bursts. Yang et al.\(^11\) established a bolt support mechanics model and determined that the length and pretightening force of the anchor bar have a greater influence on the efficiency of the anchor bar support.

On the other hand, the advanced hydraulic support structure optimization design and theoretical analysis are considered. Wang et al.\(^12\) designed a stack-type, advanced hydraulic support, combined with the geological conditions of the Bayangaole Coal Mine’s fully mechanized mining face, and determined the main stack-type, advanced hydraulic support using theoretical analysis and measured experimental data, technical parameters, and mechanism characteristics. Li et al.\(^13\) determined the type of multicolumn and multisection, self-moving, advanced support frame suitable for the geological conditions and mining efficiency of a fully mechanized mining face with a height of 8.2 m in the Jinjitan Coal Mine and developed anchor avoidance, roof protection, and new walking structures, such as control and slope connecting beams. Tang et al.\(^14\) designed a self-moving, energy-absorbing, and scour-proof roadway support for impact ground pressure mines and conducted systematic, theoretical, numerical, and experimental studies on the energy-absorbing and scour-proof mechanism of impact ground pressure roadways and supports. Li et al.\(^15\) simplified the advanced hydraulic support column to an elastic support, established a rock beam model of advanced support group elastic support, and verified the
effectiveness of advanced hydraulic support and noneq-
ual strength support to enhance the adaptability of the
advanced support system. Wang and Niu\textsuperscript{16} analyzed the
influence range and distribution rules of mining dynamic
loads using a combination method of numerical simula-
tion and field monitoring and proposed the concept of
nonequal strength coupling support with “low initial
support, high resistance.” Xie et al.\textsuperscript{17} established a
similar experimental advanced hydraulic support simu-
lation prototype according to a 1:8 scaling ratio based on
a similar simulation theory, conducted mechanical
experiments on the coupling system of the simulated
advanced hydraulic support and the roof, and obtained
different supports and roof deformation and force under
loading conditions.

The above research results mostly from the anchor
bar (cable) support mechanism and advanced hydraulic
support structure design point of view did not consider
the anchor bar (cable) and advanced hydraulic support
coupling support system, the influence of the initial
bearing capacity on the roof characteristics, and that the
key to advanced hydraulic support adaptive control is the
initial adaptive bearing capacity. In our research, elastic
mechanics theory\textsuperscript{18} combined with the linear superposi-
tion principle\textsuperscript{19,20} is applied to establish the contact
mechanics model of a single group of advanced hydraulic
supports and the surrounding rock of a roadway under
the influence of anchor support. According to the
geological conditions of the 15106 working face in
Pingshuo Coal Mine, Yangquan, Shanxi, the two-repeat
Gauss–Legendre integral\textsuperscript{21} solution method is selected to
solve and analyze the contact mechanics model under
different initial bearing capacities and to obtain the
roof displacement and stress distribution law of advanced
hydraulic support under different initial bearing
capacities. According to similarity simulation theory, a
simulation experiment model is established to verify the
displacement and stress distribution law of the active
support roof of the coupling system. The research results
can provide not only a theoretical basis for the optimal
support of a mining roadway in a fully mechanized
mining face but also a reference for the adaptive control
of advanced hydraulic support.

2  MECHANICAL MODEL OF
THE CONTACT BETWEEN THE
ADVANCED HYDRAULIC SUPPORT
AND THE ROOF

2.1  Basic theory of the contact model
between the advanced hydraulic support
and the roof

The advanced section of the underground mining roadway
uses the advanced hydraulic support group and anchor bar (cable) arranged in the surrounding rock of the roadway to support the roof of the roadway. The combined support method is shown in Figure 1. It can be seen from the type of roadway roof supported by the advance hydraulic support group and anchor bar (cable) support group that the advance hydraulic support group only relies on the top beam of the support to contact the roadway roof surface and that the anchor bar (cable) mainly relies on the rectangular tray to contact the roadway roof surface. The contact form is shown in Figure 2.

As seen from Figure 2, the leading hydraulic support roof beam, anchor bar, and anchor cable tray can be regarded as a rigid body, and the roof coal and rock can be regarded as an infinite elastic half-plane. When the advanced hydraulic support supports the...
roof coal and rock mass, the roof will be compressed along the z-direction by a small amount of thickness, that is, the roadway roof coal and rock mass will be elastically deformed, and during the supporting process, the anchor bar (cable) tray support will be attributed to the change in the stress of the roof coal and rock mass. Therefore, the process of supporting coal and rock masses with advanced hydraulic support and anchor bars (cables) can be regarded as the contact problem of multiple rigid rectangular surfaces and elastic half-planes.

According to the theory of elasticity, the half-plane contact problem can be solved by solving the half-space body subjected to the normal concentrated force on the boundary. As shown in Figure 3, there is an elastic half-space body whose gravity is not considered. If a normal concentrated force \( F_N \) is applied to its boundary, the force on the half-space plane can be solved according to the axisymmetric problem with the normal concentrated force as the axis of symmetry. According to existing knowledge, the axisymmetric problem can be described in the form of polar coordinates, the Boussinesq potential function\(^{22,23} \) is applied to solve the problem, and the displacement and stress of any point in the half-space plane under the action of the normal concentrated force can be obtained, as shown in Formulas (1) and (2) as follows:

\[
\begin{align*}
\sigma_\rho &= \frac{F_N}{2\pi R^2} \left[ \frac{(1 - 2\mu)R}{R + z} - \frac{3\rho^2 z}{R^3} \right], \\
\sigma_\phi &= \frac{(1 - 2\mu)F_N}{2\pi R^2} \left( \frac{z}{R} - \frac{R}{R + z} \right), \\
\sigma_z &= -\frac{3F_N \rho^2}{2\pi R^3}, \\
\tau_\phi &= -\frac{3F_N \rho z^2}{2\pi R^5},
\end{align*}
\]  

(2)

where \( E \) is the elastic modulus of the half-space elastomer, \( \mu \) is the Poisson's ratio of the half-space elastomer, and \( R = \sqrt{\rho^2 + z^2} \).

2.2 | Pallet–roof contact mechanics model

The anchor bar (cable) mainly relies on the contact between the pallet and the coal and rock mass on the roof.
surface when supporting the roof of the roadway. The existing anchor bar (cable) pallet consists of mostly square structures. Therefore, when solving the displacement of a point outside the contact range of the half-space body, a singular value point should appear at the position of the rectangular angle. The anchor bar (cable) pallet is regarded as a circular area, that is, a square inscribed in a circle. The method of solving displacement and stress under the force of the circular area at a point outside the contact range of the half-space body is adopted, combined with the principle of linear superposition, to establish the mechanical model of the contact between the anchor bar (cable) and the roadway roof, as shown in Figure 4.

Assume that the distance between the center of the left anchor bar and the boundary of the roadway is \( c \), the radius of the anchor bar is \( r_{\text{bolt}} \), the uniform load acting on each anchor bar pallet is \( p_{\text{bolt}} \), the radius of the anchor cable is \( r_{\text{cable}} \), the uniform load acting on the anchor cable is \( p_{\text{cable}} \), the row distance between the anchor bar and the anchor cable is \( M \) and \( B \), respectively, the circular area of the left anchor bar is \( D_{t1} \) and that of the right is \( D_{t2} \), and the circular area of the anchor cable is \( T_{vg} \). There are a total of \( n_2 \) anchor bars on the left and right sides of the half-space and a total of \( n_4 \) anchor cables in the circular area. The expression of the distance from the point \( E(x, y, z) \) outside the contact range of the roadway roof are summed by the anchor bar (cable) to obtain the relationship between the stress and the displacement of the roof of the anchored and supported roadway.

\[
R_{(\text{bolt})_{ij}} = \{(x - c)^2 + [(t_i - 1)M + r_{\text{bolt}} - y]^2\}^{1/2} \\
(t_i = 1, 2, 3, ..., n_2), \tag{3}
\]

\[
R_{\text{cable}(vg)} = \{(x - c - vE)^2 + [(g - 1)M + r_{\text{cable}} - y]^2\}^{1/2} \tag{5}
(g = 1, 2, 3, ..., n_2), \quad (v = 1, 2, 3, 4).
\]

According to linear superposition theory, the stress and displacement of point \( E(x, y, z) \) outside the contact range of the roadway roof are summed by the anchor bar (cable) to obtain the relationship between the stress and the displacement of the roof of the anchored and supported roadway.

### 2.3 Roof-top beam contact mechanics model of advanced hydraulic support

The advanced hydraulic support is composed of two single-row supports on the left and on the right. The top beam support range is symmetrically distributed according to the centerline of the support. Therefore, when investigating the mechanical characteristics of the roadway roof supported by the advanced hydraulic support, they can be simplified to examine the top beam support of the single-row, advanced hydraulic support mechanical properties of coal and rock mass in roadway protection. According to linear superposition theory, the mechanical properties of the support top beams on the left and right sides are superimposed to obtain the mechanical properties of the coal rock mass supported by the advanced hydraulic support group.

According to the contact form of the roadway roof supported by the advanced hydraulic support, the advanced hydraulic support mainly depends on the contact between the upper-end surface of the roof beam and the roadway roof in the supporting process. The top beam of the advanced hydraulic support is a rectangular surface, the length of the top beam of the advanced hydraulic support is \( a \) and the width is \( b \). Therefore, when the support supports the roof coal and rock mass, the contact surface is a rectangular contact surface, as shown in Figure 5. However, because the vertices of the rectangular area cannot be integrated or because the results are singular when calculating the stress and deformation of the half-space body, the rectangular contact area is divided into \( n_1 \) columns along the length direction of the top beam, into \( m \) rows along the width direction of the top beam, and into circular regions with a radius of \( r_{ij} \) \((i = 1, 2, 3, ..., m, j = 1, 2, 3, ..., n_1)\), and it is assumed that the load...
acting on each circular area by the leading hydraulic support column is a uniform load. When calculating the stress and displacement of point $E(x, y, z)$ outside the contact range of the roadway roof under the contact support of the leading hydraulic support top beam, they can be converted to the stress and displacement change of point $E(x, y, z)$ outside the contact range under the action of the circular area. Therefore, the changes in the stress and displacement outside the contact range of the circular area at different positions can be calculated, and the action results can be linearly superimposed and summed to obtain the coal and rock mass of the roadway roof outside the contact range during the contact support process of the single-row, advanced hydraulic support changes in stress and displacement.

The contact model of the advanced hydraulic support and roadway roof is established, as shown in Figure 6. The $x$-direction is the width of the roadway, the $y$-direction is the direction of the roadway, the contact circle on the left is represented by $C_{ij}$, and the contact circle on the right is represented by $A_{ij}$. Assume that the width of the roadway is $j$, that the length is $e$, that the distance between the top beam of the advanced hydraulic support and the two sides of the roadway is $L$, that the distance between the end of the support and the coordinate axis is $Q$, that the distance between the top beams of the two sets of supports is $N$, and that the radius of the circular area is $r$. The distance from any point in the half-space plane of the roadway roof to the top beams on both sides of the advanced hydraulic support group is shown in Equations (3) and (4) as follows:

$$R_{\text{right}(j)}^2 = [(L + 2b + N) - 2mr + r - x]^2 + [(Q + a) - y - 2n_r r + r]^2.$$  

(6)

In the process of supporting the surrounding rock of the roadway, the advanced hydraulic support mainly provides pressure from the emulsion pump station to support the support column. Therefore, the support process of the advanced hydraulic support is mainly under the action of the initial bearing capacity, and the uniformly distributed load of each circular region in the model can be calculated according to Formula (5) as follows:

$$p_{i} = \frac{p_r}{m n_i}.$$  

(8)

Since the considered points fall outside the contact area, that is, $x^2 + y^2 > r$, Formulas (6)–(8) can be integrated into Formulas (1) and (2) and then summed.
to obtain the displacement and stress changes of any point outside the contact area of the roof coal and rock under the action of the roof beam. The displacement and stress relationship of the roadway roof can be obtained by linear superposition of any point outside the contact range between the roof coal and rock mass and the top beam of the advanced hydraulic support under the action of the top beam support on both sides.

2.4 Mechanical model of advanced hydraulic support–roof contact under the influence of anchor support

The top contact form of the advanced hydraulic support group and anchor bar (cable) coupled support roadway is shown in Figure 7. The top contact mechanical model of roof contact for the roadway supported by advanced hydraulic support and the roof contact mechanics model of the roadway supported by anchor equipment are linearly superimposed. The displacement and stress changes of the roadway roof under the combined action can be obtained. The roof displacement and stress changes of a single group of advanced hydraulic support anchoring combined support roadways are shown in Formulas (9)–(14).

Displacement relation:

\[ u_{\text{whole}}(z) = u_{\text{support}}(z) + u_{\text{anchorage}}(z) \]

\[ = \frac{m}{\pi E} \sum_{i=1}^{m} \sum_{j=1}^{n_i} (1 + u) \frac{1}{\pi E} \int_{x}^{y} \frac{1}{a^2 + z^2} \]
Stress relation:

\[
\sigma_{\text{whole}(\zeta)} = \sigma_{\text{support}(\zeta)} + \sigma_{\text{anchorage}(\zeta)}
\]

\[
= \sum_{i=1}^{m} \sum_{j=1}^{n_i} \frac{1}{\pi} \left\{ \int_{0}^{\arctan\left(\frac{R_{(i)}}{R_{(j)}}\right)} \psi_i \left( \frac{1}{\sqrt{a^2 + z^2}} \right) \right. \nonumber \\
+ \int_{\arctan\left(\frac{R_{(i)}}{R_{(j)}}\right)}^{\arctan\left(\frac{R_{(i)}}{R_{(j)}}\right)} \psi_i \left( \frac{1}{\sqrt{a^2 + z^2}} \right) \nonumber \\
\left. \left( 1 - 2\mu \right) \sqrt{a^2 + z^2} + \frac{3\alpha^2 \zeta}{(a^2 + z^2)^2} \right\} \text{adap} \psi \nonumber \\
+ \sum_{k=1}^{n_2} \frac{1}{\pi} \int_{0}^{\arctan\left(\frac{R_{(k)}}{R_{(j)}}\right)} \psi_k \left( \frac{1}{\sqrt{a^2 + z^2}} \right) \nonumber \\
\left( 1 - 2\mu \right) \sqrt{a^2 + z^2} + \frac{3\alpha^2 \zeta}{(a^2 + z^2)^2} \right\} \text{adap} \psi \nonumber \\
+ \sum_{k=1}^{n_2} \sum_{l=1}^{n_2} \frac{1}{\pi} \int_{0}^{\arctan\left(\frac{R_{(k)}}{R_{(l)}}\right)} \psi_k \left( \frac{1}{\sqrt{a^2 + z^2}} \right) \nonumber \\
\left( 1 - 2\mu \right) \sqrt{a^2 + z^2} + \frac{3\alpha^2 \zeta}{(a^2 + z^2)^2} \right\} \text{adap} \psi \nonumber \\
= \sum_{i=1}^{m} \sum_{j=1}^{n_i} (1 - 2\mu) \left\{ \int_{0}^{\arctan\left(\frac{R_{(i)}}{R_{(j)}}\right)} \frac{1}{\sqrt{a^2 + z^2}} \right. \nonumber \\
+ \int_{\arctan\left(\frac{R_{(i)}}{R_{(j)}}\right)}^{\arctan\left(\frac{R_{(i)}}{R_{(j)}}\right)} \frac{1}{\sqrt{a^2 + z^2}} \nonumber \\
\left( 1 - 2\mu \right) \sqrt{a^2 + z^2} + \frac{3\alpha^2 \zeta}{(a^2 + z^2)^2} \right\} \text{adap} \psi \nonumber \\
+ \sum_{k=1}^{n_2} (1 - 2\mu) \left\{ \int_{0}^{\arctan\left(\frac{R_{(k)}}{R_{(j)}}\right)} \frac{1}{\sqrt{a^2 + z^2}} \right. \nonumber \\
+ \int_{\arctan\left(\frac{R_{(k)}}{R_{(j)}}\right)}^{\arctan\left(\frac{R_{(k)}}{R_{(j)}}\right)} \frac{1}{\sqrt{a^2 + z^2}} \nonumber \\
\left( 1 - 2\mu \right) \sqrt{a^2 + z^2} + \frac{3\alpha^2 \zeta}{(a^2 + z^2)^2} \right\} \text{adap} \psi \nonumber \\
+ \sum_{k=1}^{n_2} \sum_{l=1}^{n_2} (1 - 2\mu) \left\{ \int_{0}^{\arctan\left(\frac{R_{(k)}}{R_{(l)}}\right)} \frac{1}{\sqrt{a^2 + z^2}} \right. \nonumber \\
+ \int_{\arctan\left(\frac{R_{(k)}}{R_{(l)}}\right)}^{\arctan\left(\frac{R_{(k)}}{R_{(l)}}\right)} \frac{1}{\sqrt{a^2 + z^2}} \nonumber \\
\left( 1 - 2\mu \right) \sqrt{a^2 + z^2} + \frac{3\alpha^2 \zeta}{(a^2 + z^2)^2} \right\} \text{adap} \psi \nonumber \\
\]
\( g = 1, 2, 3, \ldots, n_2 \); \( E \) is the elastic modulus of the roof coal and rock mass; and \( \mu \) is the Poisson’s ratio of the roof coal rock mass.

### 3 | MODEL SOLVING AND ANALYSIS

#### 3.1 | Engineering geological background parameters

This paper selects the 15106 working face mining roadway in Wenjiazhuang Coal Mine of Pingshuo Company of the Yangquan Group in Shanxi as the research background. The overall shape of the working face is a monoclinic structure with a high north and low south. The elevation is +695.3 to +763.2 m, the depth of the coal seam is 488.3–498.1 m, and the strike length is 1948.3 m. The layout of the roadways on both sides of the working face is shown in Figure 8.

The coal seam of the working face belongs to the pressure mining area of the Taiyuan Formation limestone. The coal quality of the mine is soft, the gas content is high, and the permeability is poor. The hydrological type is medium. The average dip angle of the coal seam is 6°, and the coal is massive and terminal, mainly mirror coal, followed by bright coal and dark coal, belonging to bright coal. The recoverable index of the coal seam is 1, the coefficient of variation is 9%, and the coal seam is generally stable. The immediate roof of the No. 15 coal seam is K2 limestone with a thickness of 2.32 m, and the old roof is sandy mudstone with a thickness of 9.05 m, containing white mica. The direct bottom is sandy mudstone with a thickness of 3.3 m and a large amount of sandstone bands. The old bottom is fine-grained sandstone with a thickness of 4.34 m, a large amount of black minerals, and a small amount of carbon chips. The separation is poor, the grinding circle is poor, argillaceous cementation occurs, and the layer has plant leaf prints. The specific stratification is shown in Figure 9.

According to the size of the end face of the return air roadway, the roadway parameters in the contact mechanics model can be determined, as shown in Table 1. The return air roadway uses the ZQL2×4000/23/50-type, advanced hydraulic support to support the...
roadway roof, as shown in Figure 10. According to the parameters of the ZQL2×4000/23/50-type advanced hydraulic support, the parameters in the contact mechanics model can be determined, as shown in Table 2.

### Table 1 Parameters of the roadway

| Parameters | $\varepsilon$ | $j$ | $L$ | $Q$ | $C$ | $D$ | $E$ | $u$ |
|------------|---------------|-----|-----|-----|-----|-----|-----|-----|
| Parameter value | 6.5 m | 5 m | 0.8 m | 0.24 m | 0.25 m | 0.44 m | 3.62 GPa | 0.21 |

**Figure 9** Synthesis column map of the 15106 working face.

### Field test of anchor bar (cable) preload

Combining the actual situation in the mine, we know that at the initial moment when the anchor bar (cable) is
erected on the roof of the roadway, the pretightening force of each anchor bar (cable) is the same. As a result, the force of each anchor bar (cable) on the roof of the roadway changed, resulting in differences in the force of the anchor bar (cable) at different positions. Therefore, to obtain the true supporting force of the anchor bar (cable) within the support range of a single group of supports, the MCZ-300 anchor bar (cable) dynamometer produced by the Shandong Dapu Machinery Manufacturing Co., Ltd. is used to monitor the force of the anchor bars (cables) within the range of a single set of supports. The monitoring equipment is shown in Figure 11.

The 15106 working face started mining on November 24, 2018; it was pressed for the first time on December 10, 2018, and the working face advanced 53 m. On December 11, 2018, the end close to the working face was used to support the first group of the roof of the roadway within the support range of the advanced hydraulic support to monitor the force of the anchor bars (cables). Along the direction of the roadway, the force of 12 rows of anchor bars (cables), for a total of 24 anchor bars and 48 anchor cables, is monitored. During the mining process, the return air roadway is condensed by gas and cannot be monitored. Therefore, at the beginning and end of the mining, the force of the anchor bars (cables) is monitored, and the average value is taken. Using Gaussian cubic and cubic sine sum functions to fit the force of the roof anchor bars (cables), the forces of the anchor bars (cables) at different positions can be obtained, as shown in Formula (15) as follows:

\[
\begin{align*}
P_{\text{bar}(1)} &= 1.79 e^0 \times \exp(-(t_1 + 241.9)/58.54)^2) \\
&+ 3.67 \times \exp(-(t_1 + 0.17)/0.78)^2) \\
&+ 14.26 \times \exp(-(t_1 - 1.626)/1.28)^2), \\
P_{\text{cable}(1g)} &= -1146 \times \exp(-(g + 9.22)/15.12)^2) \\
&+ 1293 \times \exp(-(g + 10.24)/16.88)^2) \\
&+ 60.45 \times \exp(-(g - 20.84)/8.88)^2), \\
P_{\text{cable}(2g)} &= 423.6 \times \sin(0.17g + 0.55) \\
&+ 321.8 \times \sin(0.2g + 3.49) \\
&+ 0.42 \times \sin(1.01g + 1.84), \\
P_{\text{cable}(3g)} &= 175.4 \times \exp(-(g + 10.49)/21.14)^2) \\
&- 1.35 \times \exp(-(g - 0.07)/0.03)^2) \\
&+ 0.6 \times \exp(-(g + 2.87)/1.76) \\
P_{\text{cable}(4g)} &= 2.07e^4 \times \sin(0.84g + 1.68) \\
&+ 2.05e^4 \times \sin(0.84g - 1.46) \\
&+ 4.17 \times \sin(2.87g - 1.76), \\
P_{\text{bar}(2l_2)} &= 457.1 \times \sin(0.15l_2 + 0.5) \\
&+ 278 \times \sin(0.22l_2 + 3.1) \\
&+ 18.11 \times \sin(0.39l_2 + 4.8),
\end{align*}
\]

where \(t_1 = 1, 2, 3, ..., n_2\), \(l_2 = 1, 2, 3, ..., n_2\), and \(g = 1, 2, 3, ..., n_2\).
FIGURE 12  Roof displacement changes in the coupling support roadway under different initial bearing capacities: (A) 6000 kN, (B) 7000 kN, (C) 8000 kN, and (D) 9000 kN.

FIGURE 13  Variation in the roof displacement gradient of the coupling support roadway with different initial bearing capacities.
According to the actual use of the anchor bars (cables) in the coal mine, the parameters of the anchor bars (cables) in the model are determined, as shown in Table 3.

3.3 Analysis of theoretical calculation results

Considering that the established mechanical model of roof contact of the advanced hydraulic support anchorage coupling support roadway is to solve the displacement and stress state outside the contact area between the roadway roof and the support member by an analytical method, it is difficult to obtain the analytical expression of the original function of the integral function. Therefore, in this paper, the above model is solved by using the numerical solution method of the double-repetition, Gauss–Legendre integral.

The variation in the z-direction displacement of the roof in the process of the supporting roadway roof with different initial bearing capacities of the advanced hydraulic support under the action of anchor bar(cable) is obtained by solving, as shown in Figure 12. The figure shows that the overall roof displacement within the support range of the advanced hydraulic support increases with an increase in the initial bearing capacity of the advanced hydraulic support and that the variation in the roof displacement of the roadway presents a saddle surface gradient distribution. Since the influence of mining and rock movement on the variation in the roof displacement of the coupled support roadway is not considered, the roof displacement is symmetrically distributed along the central axis of the roadway. The

![Figure 14](image-url) Stress distribution of roadway roof coupling support with different initial bearing capacity coupling supports: (A) 6000 kN, (B) 7000 kN, (C) 8000 kN, and (D) 9000 kN.
closer to the top beam of the advanced hydraulic support, the greater the displacement is, and the maximum displacement is generated within the middle position of the top beam of the advanced hydraulic support.

Since the roof displacement changes are symmetrically distributed, selecting the roadway central axis as the boundary, the coupling support range of one side from the periphery to the central axis is analyzed, as shown in Figure 13. When the initial bearing capacity is 6000 kN, the displacement changes of the six gradients are distributed from the roadway boundary to the roadway axis, and the maximum gradient change is 7 mm. The initial bearing capacity is 7000 kN, the displacement changes of seven gradients are distributed from the roadway boundary to the roadway axis, and the maximum gradient change is 8 mm. The initial bearing capacity is 8000 kN, the displacement changes of six gradients are distributed from the roadway boundary to the roadway axis, and the maximum gradient change is 7 mm. The initial bearing capacity is 9000 kN, the displacement changes of seven gradients are distributed from the roadway boundary to the roadway axis, and the maximum gradient change is 6 mm. It can be seen that with an increase in the initial bearing capacity of the advanced hydraulic support, the gradient change of the displacement of the roof is more intense, which easily causes roof damage.

The stress distribution of the roadway roof under different initial bearing capacity coupling supports is shown in Figure 14. The figure shows that the overall roof stress within the support range of the advanced hydraulic support group increases with an increase in the initial bearing capacity of the advanced hydraulic support and that the roof stress of the roadway presents a saddle surface gradient distribution. Since the influence of mining and rock movement on the roof stress distribution of the coupled support roadway is not considered, the roof stress is symmetrically distributed along the width direction of the roadway. The closer to the top beam of the advanced hydraulic support, the greater the stress is, and the maximum stress is generated within the middle position of the top beam of the advanced hydraulic support.

As shown in Figure 15, when the initial bearing capacity is 6000 kN, the stress changes of seven gradients are distributed from the roadway boundary to the roadway central axis, and the maximum gradient change is 0.3 MPa. The initial bearing capacity is 7000 kN, the stress changes of eight gradients are distributed from the roadway boundary to the roadway central axis, and the maximum gradient change is 0.25 MPa. The initial bearing capacity is 8000 kN, the stress changes of eight gradients are distributed from the roadway boundary to the roadway central axis, and the maximum gradient change is 0.2 MPa. The initial bearing capacity is 9000 kN, the stress changes of nine gradients are distributed from the roadway boundary to the roadway central axis, and the maximum gradient change is 0.25 MPa. It can be seen that with an increase in the initial bearing capacity of the advanced hydraulic support, the stress gradient of the roof changes more violently, which easily causes roof damage.

The displacement and stress changes of the surrounding rock roof of the roadway supported by different initial bearing capacities of the advanced hydraulic support and anchor bar (cable) coupling support were compared. The results of the comparison indicate that with an increase in the initial bearing capacity of the advanced hydraulic support, the superposition effect of the supporting stress area of the top beam on both sides is obvious, especially the stress value and the area of the middle area of the roadway. The results show that the increase in the initial bearing capacity has a great influence on the supporting effect of the middle area of the roadway and has a relatively small influence on the supporting effect of the surrounding area of the top beam. By improving
the initial bearing capacity of the advanced hydraulic support, the supporting effect of the roadway can be significantly improved. The research methods and results can provide a theoretical basis for the mechanism of roadway surrounding rock bodies supported by advanced hydraulic support and the adaptive control of advanced hydraulic support in fully mechanized mining faces.

4 | EXPERIMENTAL VERIFICATION

4.1 | Parameter determination of a similar simulation experiment

The experiment uses the three-dimensional similar simulation experiment platform designed by the "Key Laboratory of Mine Subsidence Disaster Prevention and Control of the Department of Education of Liaoning Province" for experimental research. The experimental platform is shown in Figure 16. The test bench can meet the compensation loading of overlying rock formation pressure and surrounding rock horizontal pressure.

According to the engineering geological conditions and supporting methods of the 15106 working face in Wenjiazhuang Coal Mine, Pingshuo, Yangquan, Shanxi Province, combined with the actual size of the similar simulation experiment platform, the geometric similarity constant is determined to be 40:1. Through the extraction and measurement of the rock core in the background working face combined with Formula (16), the similarity constant of bulk density is 1.6:1, the time similarity constant is $\sqrt[4]{40}:1$, and the strength similarity constant is $64:1$.

\[
\begin{align*}
\alpha_L &= \frac{L_a}{L_m}, \quad \alpha_t = \frac{\sqrt{L_a}}{\sqrt{L_m}}, \\
\alpha_y &= \frac{\gamma_a}{\gamma_m}, \quad \alpha_p = \frac{\rho_a}{\rho_m},
\end{align*}
\]  

(16)

where $\alpha_L$ is the geometrical similarity constant, $L_a$ is the prototype length, $L_m$ is the simulated length, $\alpha_t$ is the bulk density similarity constant, $\gamma_a$ is the prototype bulk density, $\gamma_m$ is the simulated bulk density, $\alpha_y$ is the time similarity constant, and $\alpha_p$ is the intensity similarity constant.

After a similar simulation experiment plan is determined, the relevant regulations of the comprehensive mining experiment were referenced to determine that the aggregate material is sand and that the cementing material is lime and gypsum and to calculate the ratio of aggregate to cementing material in each layer according to Formula (17). The amount of similar materials for each layer is obtained as follows:

\[
G = l_m \times d_m \times h_m \times \gamma_m,
\]  

(17)

where $G$ is the total weight of the model layered material (kg), $l_m$ is the model length (m), $d_m$ is the model width (m), $h_m$ is the thickness of the simulated layer (m), and $\gamma_m$ is the bulk density of the simulated rock layer (kg/m$^3$).

Figure 10 shows the layered comprehensive histogram of the background roadway top rock layer combined with similar proportions to determine similar model materials, as shown in Table 4.

According to the above proportioning method and amount of aggregate and cementing material, a pouring simulation experiment model is employed. The electro-hydraulic servo universal test machine is selected to test the samples processed according to the experimental standard. The experimental test is shown in Figure 17.

The physical and mechanical parameters of similar simulation samples of coal and rock strata in the 15106 working face of Wenjiazhuang Mine of Pingshu Company, Yangquan Group in Shanxi Province were tested by laboratory experimental research methods. According to the strength similarity constant, bulk density similarity constant, and geometric similarity constant, the experimental results were converted to obtain the physical and mechanical parameters of the coal and rock mass corresponding to the simulation samples, as shown in Table 5.
### Table 4: Table of similar simulation experimental material ratio

| Serial number | Lithology          | Buried depth (m) | Proportion number | Layer number | Stratification thickness (mm) | Gross mass (kg) | sand (kg) | Lime (kg) | Gypsum (kg) | Water (kg) |
|---------------|--------------------|------------------|-------------------|--------------|-------------------------------|-----------------|-----------|-----------|-------------|------------|
| 1             | Medium-grained sandstone | 457.77           | 355               | 4            | 15.7                          | 227.3           | 151.6     | 25.3      | 25.3        | 25.3       |
| 2             | Coarse-grained sandstone | 460.28           | 437               | 3            | 14.4                          | 149.2           | 106.1     | 8.0       | 18.6        | 16.6       |
| 3             | Siltstone          | 462.01           | 437               | 3            | 14.0                          | 151.7           | 107.9     | 8.1       | 18.9        | 16.9       |
| 4             | Medium-grained sandstone | 463.69           | 355               | 5            | 15.1                          | 273.4           | 182.3     | 30.4      | 30.4        | 30.4       |
| 5             | Coarse-grained sandstone | 466.71           | 437               | 4            | 15.9                          | 219.8           | 156.3     | 11.7      | 27.3        | 24.4       |
| 6             | Sandy mudstone    | 469.26           | 537               | 1            | 14.9                          | 52.1            | 38.6      | 2.3       | 5.4         | 5.8        |
| 7             | Fine-grained sandstone | 469.85           | 455               | 2            | 12.4                          | 84.6            | 60.1      | 7.5       | 7.5         | 9.4        |
| 8             | Sandy mudstone    | 470.84           | 537               | 1            | 16.8                          | 58.8            | 43.6      | 2.6       | 6.1         | 6.5        |
| 9             | Fine-grained sandstone | 471.52           | 455               | 2            | 10.7                          | 72.8            | 51.7      | 6.5       | 6.5         | 8.1        |
| 10            | Medium-grained sandstone | 472.37           | 355               | 5            | 16.9                          | 308.9           | 206.0     | 34.3      | 34.3        | 34.3       |
| 11            | Fine-grained sandstone | 475.75           | 455               | 2            | 13.9                          | 94.3            | 67.1      | 8.4       | 8.4         | 10.5       |
| 12            | Sandy mudstone    | 476.86           | 537               | 3            | 13.9                          | 146.6           | 108.6     | 6.5       | 15.2        | 16.3       |
| 13            | Limestone         | 478.53           | 337               | 2            | 10.7                          | 84.2            | 56.1      | 5.6       | 13.1        | 9.4        |
| 14            | Coal seam         | 479.39           | 573               | 1            | 2.3                           | 4.3             | 3.2       | 0.5       | 0.2         | 0.5        |
| 15            | Sandy mudstone    | 479.48           | 537               | 3            | 17.8                          | 187.1           | 138.6     | 8.3       | 19.4        | 20.8       |
| 16            | Coal seam         | 481.62           | 573               | 1            | 11.8                          | 21.9            | 16.3      | 2.3       | 1.0         | 2.4        |
| 17            | Mudstone          | 482.09           | 555               | 1            | 7.3                           | 23.1            | 17.1      | 1.7       | 1.7         | 2.6        |
| 18            | Limestone         | 482.38           | 337               | 1            | 16.4                          | 64.3            | 42.9      | 4.3       | 10.0        | 7.1        |
| 19            | Mudstone          | 483.04           | 555               | 1            | 9.6                           | 30.5            | 22.6      | 2.3       | 2.3         | 3.4        |
| 20            | Coal seam         | 483.42           | 573               | 1            | 2.5                           | 4.7             | 3.5       | 0.5       | 0.2         | 0.5        |
| 21            | Sandy mudstone    | 483.52           | 537               | 2            | 15.1                          | 105.5           | 78.1      | 4.7       | 10.9        | 11.7       |
| Serial number | Lithology               | Buried depth (m) | Proportion number | Layer number | Stratification thickness (mm) | Gross mass (kg) | sand (kg) | Lime (kg) | Gypsum (kg) | Water (kg) |
|---------------|-------------------------|------------------|-------------------|--------------|------------------------------|----------------|-----------|-----------|-------------|------------|
| 22            | Limestone               | 484.73           | 337               | 2            | 19.0                         | 148.9          | 99.3      | 9.9       | 23.2        | 16.6       |
| 23            | Sandy mudstone         | 486.25           | 537               | 1            | 10.4                         | 36.4           | 27.0      | 1.6       | 3.8         | 4.1        |
| 24            | Fine-grained sandstone | 486.66           | 455               | 2            | 10.0                         | 68.0           | 48.4      | 6.1       | 6.1         | 7.6        |
| 25            | Sandy mudstone         | 487.46           | 537               | 4            | 15.1                         | 211.1          | 156.4     | 9.4       | 21.9        | 23.5       |
| 26            | Limestone               | 489.88           | 337               | 1            | 15.5                         | 60.6           | 40.4      | 4.1       | 9.4         | 6.7        |
| 27            | Coal seam               | 490.5            | 573               | 2            | 10.5                         | 39.2           | 29.1      | 4.1       | 1.7         | 4.4        |
| 28            | Sandy mudstone         | 491.38           | 537               | 2            | 11.0                         | 77.0           | 57.1      | 3.4       | 8.0         | 8.6        |
| 29            | Fine-grained sandstone | 492.54           | 455               | 2            | 14.5                         | 98.4           | 70.0      | 8.8       | 8.8         | 10.9       |
| 30            | Sandy mudstone         | 493.08           | 537               | 1            | 13.5                         | 47.1           | 34.9      | 2.1       | 4.9         | 5.2        |
| 31            | Mudstone                | 493.29           | 555               | 1            | 5.3                          | 17.0           | 12.6      | 1.3       | 1.3         | 1.9        |
| 32            | Siltstone               | 494.91           | 437               | 3            | 13.5                         | 146.2          | 103.9     | 7.8       | 18.2        | 16.2       |
| 33            | Fine-grained sandstone | 495.38           | 455               | 1            | 11.8                         | 40.1           | 28.5      | 3.6       | 3.6         | 4.5        |
| 34            | Sandy mudstone         | 495.97           | 537               | 1            | 14.7                         | 51.3           | 38.0      | 2.3       | 5.3         | 5.7        |
| 35            | limestone               | 496.66           | 337               | 1            | 17.3                         | 67.8           | 45.2      | 4.2       | 10.6        | 7.5        |
| 36            | Carbonaceous mudstone  | 496.84           | 537               | 1            | 4.5                          | 14.3           | 10.6      | 0.6       | 1.5         | 1.6        |
| 37            | Sandy mudstone         | 497.18           | 537               | 1            | 8.3                          | 29.2           | 21.6      | 1.0       | 3.1         | 3.2        |
Physically and mechanically test site of the samples

Shear strength test of the specimens

Tensile strength test of the samples

Compressive strength test of the samples

FIGURE 17  Test of physical and mechanical parameters of the samples. (A) Physical and mechanical test site of the sample, (B) shear strength test of the specimens, (C) tensile strength test of the samples, and (D) compressive strength test of the samples.

| Physical and mechanical parameters | The old roof | Immediate roof | Coal samples | Direct bottom | Old bottom |
|------------------------------------|-------------|----------------|--------------|---------------|-----------|
| Natural visual density (g/cm³)     | 2.62        | 2.95           | 1.4          | 2.62          | 2.55      |
| Compressive strength (MPa)         | 34.2        | 93.8           | 15.76        | 34.2          | 48.4      |
| Tensile strength (MPa)             | 2.83        | 2.97           | 0.68         | 4.61          | 6.23      |
| Elastic modulus (GPa)              | 35.63       | 31.23          | 3.62         | 36.84         | 33.65     |
| Poisson ratio                      | 0.36        | 0.3            | 0.21         | 0.20          | 0.22      |
4.2 Similar simulation experiment design and model pouring

After the similarity constant is determined, a similar experimental model is designed, as shown in Figure 18. Three experimental roadways are arranged in the experiment, including an advanced hydraulic support anchorage combined support experimental roadway, an anchorage support experimental roadway, and a standby experimental roadway. The roadway layout is shown in the figure.

In the experiment, a BX120-10AA (10×5) foil-type, acetal strain gauge was used to test the stress change in
the experimental roadway roof, and a miniature, round-tube, self-resetting, KSP-25 mm displacement sensor was utilized to test the deformation of the roadway roof. The sensor layout is shown in Figure 19. To ensure the accuracy of the experimental results, a 100 mm (actually 4 m) coal pillar is installed at the front of the working face to simulate the boundary conditions. A total of 6 rows of 24 anchor cables and 12 bolts are arranged on the roof of the roadway, and a total of 6 rows of 36 anchors are arranged on the left and right sides of the roadway. The advanced hydraulic supports were erected near the boundary conditions. A set of advanced hydraulic supports were arranged in the experimental roadway. A special adhesive was used to paste four pieces of 10 × 5 mm² strain gauges in the middle of the top beam of the supports. From the left to the front and rear of the supports, four strain gauges were installed in sequence on the right. A total of 12 strain gauge sensors are arranged in the experimental roadway, and 4 rows are arranged in the reverse direction along the roadway, with 3 displacement sensors in each row. A total of 12 displacement sensors are arranged.

Model pouring was carried out according to the determined similar simulation ratio plan. During pouring, an appropriate amount of 45 mesh mica powder was added between each layer as natural bedding, and 10 mesh mica powder was used as an interlayer for cracks and joints. After the model was naturally dried for 5 days, the surrounding baffles were removed for further drying treatment. During the drying process, proper watering and maintenance treatment were performed every 12 h to prevent the model from cracking due to external factors such as temperature changes. After 10 days of curing, the experimental test can be carried out. The model after pouring similar materials is shown in Figure 20.

In the experiment, a pneumatic thin cylinder is used to simulate the advanced hydraulic support group, as shown in Figure 21. The main technical parameters of the cylinder are shown in Table 6.

To meet the requirements of similar simulation experiments and the actual working requirements of the advanced support group, two pneumatic pumps with a TYW-1 maximum supply pressure of 1.5 MPa were utilized for continuous pressure supply. The PU pipe is selected to connect the reversing valve, pressure reducing valve, cylinder and other components to form a simulated advanced hydraulic support group operation and control system. In the simulation system of the advanced hydraulic support group, the output end of the directional valve is connected with the pressure reducing valve to control the initial bearing capacity of the advanced hydraulic support group.

Experimental data were collected using the TST3822 static strain gauge, as shown in Figure 22A. The roof stress value is collected once a second, and the test and acquisition system of the surrounding rock displacement of the experimental roadway adopts ADAM-4117-AE (solid, eight-way, differential A/D input module) to compile the acquisition program to collect the displacement signal, as shown in Figure 22B. The acquisition system can meet real-time, online testing to ensure the accuracy of test data transmission.

### Table 6 Main technical parameters of the simulated advanced hydraulic support group

| Parameter                               | Parameter value |
|-----------------------------------------|-----------------|
| Effective support height of cylinder    | 57.5–125 mm     |
| Support length                          | 142.5 mm        |
| Working temperature                     | −20 to 80°C     |
| Speed range                             | 30–500 mm/s     |
| Range of travel tolerances              | +1.0            |
| Actuating medium                        | Air (filtered by a filter of 40 μm or more) |
| Mode of operation                       | Pneumatic operation |

#### 4.3 Comparative analysis of experimental results

In the experiment, the simulated advanced hydraulic support group was used to support the roof of the anchor bar roadway, and the initial bearing capacity of the simulated support was adjusted by controlling the pressure-reducing valve for coupling support. Each initial bearing capacity of the advanced hydraulic support group is adjusted from small to large. Each initial bearing support time is 10 min, and the experiment is 40 min.
Considering the influence of the simulated roadway boundary constraint on the experimental results, after the experiment, the roof deformation and stress change along the width direction of roadway are taken at the middle position along the direction of roadway, that is, 175 mm from the end (the first 100 mm is the boundary position). The experimental results are transformed according to the geometric similarity constant and strength similarity constant, as shown in Figures 23 and 24.

Figure 23 shows that with an increase in the initial bearing capacity of the advanced hydraulic support, the deformation of the roadway roof increases, and the maximum deformation occurs at the boundary of the top beam support of the advanced hydraulic support. The deformation of the roof along the width direction of the roadway is distributed in a symmetrical gradient according to the central axis of the roof.

The stress curve of the roadway roof under different initial bearing capacity supports is shown in Figure 24. It can be seen that with an increase in the initial bearing capacity of the overdue hydraulic support, the stress value of the roof gradually increases, and the maximum stress value occurs at the boundary position of the roof beam of the advanced hydraulic support. The stress along the width direction of the roadway is distributed in a gradient symmetry according to the central axis of the roof.

The comparative analysis of the stability coupling support results at the same position as the theoretical analysis and simulation experiment is shown in Figure 25. The maximum difference in roof deformation with different advanced hydraulic support initial bearing capacities is 9.3 mm. The maximum difference accounts
for 19.51% of the experimental results. Considering the complexity of the influencing factors in similar simulation experiments and the simplification of the boundary conditions of the theoretical model, an error less than 20% can be the receiving error.

As shown in Figure 26, the maximum stress difference between the theoretical calculation and the simulation experiment for different initial bearing capacities of advanced hydraulic support is 0.34 MPa. The maximum difference is 13.77% of the experimental results, which is an acceptable error range, so the accuracy of the theoretical calculation results is verified.

5 | CONCLUSION

(1) The elastic mechanics theory is used to establish the contact mechanics model of the advanced hydraulic support anchoring combined support roadway roof under the condition of anchor bar(cable) support, and the analytical solutions of the roof stress and strain are obtained.

(2) Analyzed the changes in the stress of the roadway roof under the initial bearing capacity of the advanced hydraulic support. The results show that the increase of initial bearing capacity has a greater impact on the support effect of the middle area of the roadway, and less impact on the support effect of the area near the top beam; Although the increase of initial bearing capacity will cause the gradient of adjacent stress areas to increase, the maximum gradient is much smaller than the breaking strength of the surrounding rock; within the scope of initial bearing capacity studied in this paper, the increase of initial bearing capacity will significantly increase the support of the roadway effect.

(3) Using similarity theory 1:40 to establish a simulation experiment model for similar experiment research, comparing theoretical calculations and experimental results, it is found that the theoretical calculation has a maximum error of 19.51% relative to the experimental results, which verifies that the simulation experiment has many influencing factors and the error does not exceed 20% is the acceptance error, so similar simulation experiments verify the accuracy of the theoretical calculation results and the rationality of the mechanical properties of the surrounding rock of the roadway supported by the advanced hydraulic support under the anchor support conditions, which is the advanced mining roadway for the fully
mechanized mining face. The adaptive control of support and advanced hydraulic supports provides a theoretical basis.

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