Overburden Deformation Rule in Super High Seam Fully Mechanized Caving Mining of Thick Unconsolidated Stratum

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It is urgent for westward coal mine development to control surrounding rock of thick unconsolidated stratum, thin basement rock, and super high seam under geological conditions. Studying surrounding rock deformation control mechanism under special geological conditions in western regions is of practical guiding significance for the implementation of China’s energy strategy in western regions, with special mining geological conditions of the experimental mine as the background, deformation rule, failure characteristic, and stress distribution characteristic of surrounding rock in fully mechanized caving mining of thick unconsolidated stratum. By establishing the structural mechanics model of “sidestepped rock beam,” the stability criterion is determined. The horizontal displacement of surrounding rock in the middle of the working face is small, and the displacement at both ends is large. After the mining of the working face, an "O" ring appears on the main roof. The ring extends outward with the excavation, reaches the maximum at the initial weighting, and its displacement decreases from inside to outside. Finally, through theoretical analysis, experimental research, numerical calculation, and field observation, the thin bedrock and ultra-high coal seam are systematically analyzed, and its deformation and failure mechanism is revealed. Relevant research findings are successfully implemented on the scene and effectively ensured safety, high yield, and high efficiency of the mine.

1. Introduction

Energy is the material basis for human survival and a topic of common concern all over the world. Energy issues affect a country’s politics, economy, security, and many other aspects. It is closely related to the national economy and people’s livelihood and national development strategy [1]. With the continuous advancement of China’s industrialization and modernization, it has been difficult to meet the needs of national sustainable development in the future using domestic traditional fossil energy [2–5]. Energy is the foundation for national economic development, and fossil energy accounts for above 90% of total energy consumption worldwide [6–8]. Although a clean, safe, and reliable sustainable energy system is extensively promoted worldwide, it will take at least decades and even a hundred years to make it a main part of energy consumption. In a very long period, fossil energy will still play a dominant position in energy exploitation and utilization. In China’s energy structure, there are abundant coal reserves but scarce reserves of petroleum and natural gas. The coal reserves of thick coal seams in China are very rich and widely distributed. The production mines with thick coal seams in the country account for 40.6% of the total production mines, and the recoverable reserves account for 45% of the total production mines. With the gradual westward shift of the strategic layout of China’s coal development, the shallow and extra thick coal seams in Inner Mongolia, Shaanxi, and Xinjiang have a large area and vast territory. The new complex geological conditions have posed higher challenges for coal mining. There are more and more new major scientific and technological issues, such as how to achieve high-yield and efficient mining and ensure production safety, among which coal caving mining technology has developed rapidly and been used.
The research on surrounding rock deformation mechanism and control of roadway in fully mechanized top coal caving mining in thick unconsolidated and extra thick seam has become one of the hot spots and difficulties in the current research. Under complex conditions such as thick loose beds, extra thick coal seams, large mining height, and large top coal caving rate, the existing generally fully mechanized top coal caving theory has difficulty in ensuring the safe and efficient mining of coal resources. Therefore, it is an urgent task to study the deformation mechanism and control of roadway surrounding rock under the condition of full space and multiple factors. Therefore, it is an urgent task to carry out the research on the deformation mechanism and control of roadway surrounding rock under the condition of full space and multifactors.

Fully mechanized caving mining results in serious deformation of overlying rock and earth surface, and long-term theoretical researches on mining subsidence and production practice prove it an interdisciplinary integrated subject concerning knowledge of surveying, mining, mechanics, mathematics, geology, and computer, so mining subsidence has been studied in different perspectives. In the early 1960s, British scholars Berry and Sales [9, 10] regarded rock mass as a homogeneous elastomer classified into plane isotropy, transverse isotropy, and space problem and put forward calculation methods for rock mass subsidence specific to unclosed, partly closed, and fully closed boundary conditions of gob. Conroy and Gyarmaty observed vertical and horizontal movement in overlying strata using borehole extensometer and borehole clinometers to not only figure out the horizontal movement rule in overlying strata but also observe the slippage and separation layer of overlying strata along bedding [11]. Since the 1980s, subsidence deformation computation has turned automatic, intelligent, and visualized with the development of computer technology and continuous improvement of mining subsidence theory [12–14]. Wang et al. [15] presented the surface movement rule in coal mining under the super thick unconsolidated stratum and its prediction method. Guo et al. [16] established the prediction method for surface movement under extremely incomplete mining conditions. Guo et al. [17] studied the neural network method selected as per rock stratum movement angle. Hu et al. [18] studied a method to determine mining subsidence predicting parameters using the probability integral method. Zhu et al. [19] discussed the change rule of the rock stratum movement angle with mining thickness, mining depth, and coal bed pitch and established a computational formula for the rock stratum movement angle under deep mining conditions. In recent years, Liu et al. [20] studied the evolution rule of overlying stratum sidestepped diastrophism during large cutting height mining and water inrush prevention and control through numerical simulation and field measurement specific to special geological conditions for western coal mining with the westward shift of China’s energy strategy. Huang et al. [21] studied periodic weighting roof structure and support load in the massive mine of shallow seam through scene investigation and physical simulation. Liu et al. [22] and Gong et al. [23] built an analog simulation test model to deeply study deformation of end surrounding rock and stability of the coal pillar under special geological conditions in western regions. In recent years, the study of surrounding rock stability and its control in fully mechanized top coal caving mining has been paid attention by many scholars at home and abroad, and a large number of achievements have been made. However, the research work on the ground pressure behavior law and surrounding rock control of the fully mechanized top coal caving roadway, especially the thick loose bed and extra thick coal seam under the thick sand and thin bedrock, is not sufficient. A large number of articles and technologies still use the traditional support theory and support method in the past [24–26].

Shallow working seam of the experimental mine, thin roof bed rock, and large working thickness resulted in serious surface deformation, obvious surface subsidence, and large and deep cracks. Sharp surface subsidence and large cracks arising from mining caused heavy environmental destruction and severe hidden danger, which not only ruined the environment and generated conflicts between workers and peasants but also enabled downhole air leak and water burst and caused spontaneous combustion or water inrush accident of the coal seam. Therefore, thoroughly analyzing and studying overlying stratum movement in fully mechanized caving mining of super high seam of thick unconsolidated stratum can not only extend the service life of the mine, increase job opportunities, and improve the living condition and ecological environment of people in the mine as quickly as possible, but also have incontrovertible great practical significance for ensuring social stability and prompting economic development [27–30].

2. Engineering Geological Conditions

The experimental mine is located in Inner Mongolia Autonomous Region of China (Figure 1). The experimental mine is located in Dalu Town in the north of the Junger Coal Field, about 95 km away from Hohhot City in the north and 150 km away from Erdos City in the west; covers an area of 33 km²; and has coal reserves of 1.145 billion tons. The surface is covered by vast thick loess and aeolian sand and features complex terrain, crossed ravines and gullies, and developed arborization gully. The research area is an intermediate depression high in the west and low in the east covered by a 20.0–34.1 m thick loess layer and featuring scarce vegetation, serious water and soil loss, and developed gully. Ejiagou passes the middle part, there is no running water in the brook, and the maximum flow rate in rainy season is 20 m³/h. The main seam is carboniferous Upper Permian Taiyuan group 6# seam with developed crack, average coal thickness of 16 m, coal bed pitch of 0°–8° and 4° on average, and average mining depth of 295 m and adopts integrated mechanized coal caving mining technology. Taking the subtest mine as the research object, studying the surrounding rock deformation control mechanism under special geological conditions in Western China has practical guiding significance for the implementation of the energy strategy in Western China.
According to analysis of the SEM microstructure [31, 32] (Figure 2) of coal sample of the experimental mine, coarse minerals of coal (rock) sample are inlaid, form a binding structure, and have large developed cracks and holes. The amount of tiny holes and cracks keeps increasing with increase in magnification times.

3. Structural Mechanics Module of Roof “Short Masonry Beam”

3.1. Rock Load Analysis. Based on observing internal movement of plenty of mining rocks and summarizing hypotheses of rock blocks with hinge structure and preformed cracks, Qian and Shi put forward the “masonry beam” structural model of rock mass structure in the 1970s and in the early 1980s [33]. According to “masonry beam” theory, the rock stratum broke under extreme span, broken rock-formed horizontal force due to rotation and mutual extrusion to generate frictional force among rocks and achieve equilibrium of three-hinged arch fracture beam under suitable horizontal extrusion force. The “short masonry beam” model of the overlying stratum structure was built for mechanical analysis of stability of the overlying stratum structure to reveal the stability principle of the overlying stratum structure (Figure 3).

According to the references [34],

\[ T = \frac{4i \sin \theta_1 + 2 \cos \theta_1}{2i + \sin \theta_1(\cos \theta_1 - 2)} p_1, \]

(1)

\[ Q_A = \frac{4i - 3 \sin \theta_1}{4i + 2 \sin \theta_1(\cos \theta_1 - 2)} p_1, \]

(2)

where \( T \) is the horizontal thrust; \( Q_A, Q_B \) are the shear force on contact hinges A and B, respectively; and \( \theta_1, \theta_2 \) are the rotation angles of blocks B and C, respectively.

3.2. Sliding Instability Analysis. Conditions for preventing sliding instability of the roof structure are

\[ T \tan \phi \geq Q_A, \]

(3)

where \( \tan \phi \) is the frictional coefficient among rocks, which is identified as 0.5 by experiment.
Equations (1) and (2) are substituted into (3):

\[ i \leq \frac{2 \cos \theta_1 + 3 \sin \theta_1}{4(1 - \sin \theta_1)}. \]  

(4)

Generally, sliding instability will not occur when the \( i \) value is within 0.9. The measured lumpiness of the experimental mine \( i = 1.15 \) and the roof structure will have sliding instability [35].

3.3. “Sidestepped Rock Beam” Structural Model and Stability Analysis. The “sidestepped rock beam” structure is formed after the sliding instability of the “short masonry beam” structure of the roof. According to the structural characteristics of roof sliding, the structural mechanics model of the “sidestepped rock beam” is established. The purpose of numerical simulation is to verify the relationship between surface settlements.

According to field measurement and analog simulation test, lumpiness (1.0-1.4) of broken rock on the shallow seam roof was high and the roof structure was a “short masonry beam” structure [36-38], which was hard to remain stable and would have sliding instability. When roof lumpiness was less than 1 or strength was weak or rotation angle was greater than 10°, sliding instability may occur easily, as shown in Figure 4.

According to field measurement and model test of the experimental mine, roof rock dropped (sliding instability) from the support beam during workface mining. The shape was formed after sliding instability, as shown in Figure 5, which was vividly called the “sidestepped rock beam” structure. Rock C was completely on the caving rock and rock B was supported by rock C at point C as the workface advanced and rotated. Then, rock C was basically compacted and \( R_s = P_2 \). The units of T and PI are N.

Deflection of rock C was

\[ \omega = m - (K_p - 1) \sum h, \]  

(5)

where \( \sum h \) is the immediate roof thickness, \( m \); \( m \) is the height mining; \( m \); and \( K_p \) is the coefficient of rock expansion which can be taken as 1.3.

As shown in Figure 5, \( b = 0 \) when rock B reached the maximum rotation angle [39], \( \sin \theta_{1 \max} = \omega/l \).

\[ T = \frac{P_1}{i - 2 \sin \theta_{1 \max} + \sin \theta_1}. \]  

(6)

According to general conditions for fully mechanized caving faces of the shallow seam, \( \theta_{1 \max} \) of the roof rock was often 8°-12° [40]. Substitute equations (6) and \( \tan \phi = 0.5 \) into equation (4) and conditions for preventing sliding instability of the “sidestepped rock beam” are

\[ i \leq 0.5 + 2 \sin \theta_{1 \max} - \sin \theta_1. \]  

(7)

The result showed that sliding instability did not occur only when \( i \) was less than 0.9. The measured lumpiness of the experimental mine was \( i = 1.15 \) and the possibility that the “sidestepped rock beam” had sliding instability was high.

4. Three-Dimensional Discrete Element Simulation

4.1. 3D Distinct Element. 3DEC (3 Dimension Distinct Element Code) is a large discrete element software developed by America ITASCA and an individual element method of the discrete element method, which can accurately simulate coal and rock mass destruction at natural true triaxial stress state. 3DEC software extends 2D planar model to 3D space based on basic theory of discrete element method to describe mechanical behavior of dispersed medium. It covers all application fields of FLAC, FLAC3D, and UDEC programs and has obvious advantages in simulating surrounding rock failure of deep underground works, large high slope stability deformation mechanism, and mining caving.

4.2. Modeling and Parameter Selection. Specific to thin bed rock and thick coal seam of experimental mine, a 3D mining rock deformation calculation model was built to analyze the influence of working face depth and working thickness on stability of surrounding rock.

According to engineering geological conditions, the 3D mining rock deformation calculation model (length × width × height = 300 m × 200 m × 50 m) of fully mechanized caving
faces was built, whose working face arrangement and mining way were consistent with the site condition. The size of the working face is length × width × height = 240 m × 100 m × 16 m; the size of the two crosshead section is width × height = 5.5 m × 3.7 m. Boundary conditions of the model are as follows: horizontal displacement of the side boundary and vertical displacement of the bottom boundary should be limited. According to the computational formula of equivalent load, \( q = \gamma \times H \), suppose \( \gamma = 2.5 \text{kN/m}^3 \) was the average unit weight of rock stratum and \( H \) was the increased thickness of bed rock stratum and apply horizontal stress to the model according to side pressure coefficient. 3D model and joint meshing are shown in Figure 6.

The physical mechanics test was conducted on coal (rock) blocks taken, as shown in Table 1, and the test result showed elastoplasticity of deformation and destruction of surrounding rock during loaded compression, which was consistent with the stress-strain relationship of the elastic-plastic model. Thus, simulation analysis was carried out on relevant block structures using the Mohr-Coulomb yield criterion model, and relevant joint surfaces were analyzed using the Coulomb slippage constitutive model.

4.3. Numerical Simulation Scheme Calculation. Several modeling schemes were designed using the single factor analysis method to analyze the rule of influence of the working face depth and working thickness on stability of surrounding rock. The numerical calculation scheme is shown in Tables 2 and 3.

5. Numerical Result Analysis

5.1. Working Face Depth Impact Analysis

5.1.1. Study of Surrounding Rock Deformation Rule. It can be seen from Table 2 that the research depths are 200 m, 400 m, 600 m, 800 m, and 1000 m. The displacement diagram of surrounding rock in the fully mechanized top coal caving face with large mining height at different depths has similar characteristics, as shown in Figure 7, which is the vertical displacement diagram of surrounding rock at 800 m depth. Zone B is located in the left half of the working face. The horizontal displacement of the surrounding rock in the middle of the working face is small, and the horizontal displacement of both ends is large. After the mining of the working face, the surrounding rock at both ends forms a masonry beam structure, and the horizontal displacement of the upper left corner and lower right corner of zone B is the largest. Vertical displacement characteristics are as follows: surrounding rock in the middle of the working face had large vertical displacement and that on both ends had small vertical displacement, and vertical displacement was in the shape of an "inverted circular truncated cone" from the middle to both ends.

5.1.2. Study of Surrounding Rock Stress Distribution Characteristics. The depth of the working face determines the vertical stress of surrounding rock, which is the main factor affecting the stability of surrounding rock. By discussing the influence of envelope stress, analyzing its stress characteristics, and combining with displacement image, the mechanism of surrounding rock instability can be better revealed.

The surrounding rock stress diagram of large cutting height fully mechanized caving faces of different depths had similar characteristics (Figure 8). Horizontal stress characteristics of surrounding rock are as follows: surrounding rock in the middle of the working face and coal pillar had large horizontal stress and that on both ends had small horizontal stress. Vertical stress characteristics are as follows: surrounding rock in the middle of the working face and coal pillar had large vertical stress and that on both ends had small stress. The tensile stress of surrounding rock on both ends was greater than the tensile strength, and surrounding rock on both ends was at tensile failure state.
5.2. Mining Height Impact Analysis of Working Face

5.2.1. Primary Rupture Regularity of Main Roof. Under geological conditions of coal seam of the experimental mine and according to drill column, the main roof was a single key layer structure and destruction of the overlying rock to surface was determined by rupture of the key layer [41–43]. In the mining process, the main roof was destroyed mainly because of stress redistribution and damage accumulation arising from step-by-step excavation. In fact, it was a process of continuous excavation, unloading, and damaging.

Figure 9 shows a large "O" ring of displacement change in the main roof displacement diagram after excavation, the "O" ring gradually grew outwards, and its displacement decreased from inside to outside. The "O" ring was the largest, and first weighting was caused by rupture of the main roof. First weighting pitch was 65 m when mining height was 8 m, 55 m when mining height was 12 m, and 50 m when mining height was 16 m, 20 m, and 24 m, indicating that first weighting pitch decreased with increase in mining height with a range and tended to be stable when mining height exceeded a certain value. When mining height was 20 m and 24 m, rock around the rupture line of the "O" ring of the main roof “turned over.” The working face was basically

<table>
<thead>
<tr>
<th>Lithology</th>
<th>Density (kg * m$^{-3}$)</th>
<th>Bulk modulus (GPa)</th>
<th>Shear modulus (GPa)</th>
<th>Cohesion (MPa)</th>
<th>Tensile strength (MPa)</th>
<th>Angle of internal friction (°)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coarse sand mudstone</td>
<td>2700</td>
<td>6.2</td>
<td>4.8</td>
<td>10.8</td>
<td>11.79</td>
<td>39</td>
</tr>
<tr>
<td>Sandy mudstone</td>
<td>2691</td>
<td>9.22</td>
<td>6.91</td>
<td>10.80</td>
<td>9.45</td>
<td>38</td>
</tr>
<tr>
<td>Mudstone</td>
<td>2450</td>
<td>5.57</td>
<td>3.18</td>
<td>3.25</td>
<td>1.09</td>
<td>30</td>
</tr>
<tr>
<td>Coal</td>
<td>1334</td>
<td>2.66</td>
<td>1.16</td>
<td>1.52</td>
<td>0.65</td>
<td>27</td>
</tr>
<tr>
<td>Mudstone</td>
<td>2100</td>
<td>4.17</td>
<td>2.38</td>
<td>2.50</td>
<td>0.80</td>
<td>30</td>
</tr>
<tr>
<td>Gritstone</td>
<td>2691.6</td>
<td>9.22</td>
<td>6.91</td>
<td>10.80</td>
<td>9.45</td>
<td>38</td>
</tr>
</tbody>
</table>

Table 1: Physical and mechanical parameters of surrounding rock properties.

<table>
<thead>
<tr>
<th>Scheme</th>
<th>Depth of working face (H/m)</th>
<th>Mining thickness (h/m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Scheme 1</td>
<td>200</td>
<td></td>
</tr>
<tr>
<td>Scheme 2</td>
<td>400</td>
<td></td>
</tr>
<tr>
<td>Scheme 3</td>
<td>600</td>
<td>16</td>
</tr>
<tr>
<td>Scheme 4</td>
<td>800</td>
<td></td>
</tr>
<tr>
<td>Scheme 5</td>
<td>1000</td>
<td></td>
</tr>
</tbody>
</table>

Table 2: Working face depth calculation scheme.

<table>
<thead>
<tr>
<th>Scheme</th>
<th>Mining thickness (h/m)</th>
<th>Depth of working face (H/m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Scheme 6</td>
<td>8</td>
<td>200</td>
</tr>
<tr>
<td>Scheme 7</td>
<td>12</td>
<td>200</td>
</tr>
<tr>
<td>Scheme 8</td>
<td>16</td>
<td>200</td>
</tr>
<tr>
<td>Scheme 9</td>
<td>20</td>
<td>200</td>
</tr>
<tr>
<td>Scheme 10</td>
<td>24</td>
<td>200</td>
</tr>
</tbody>
</table>

Table 3: Influence of mining thickness calculation scheme.
at the tensile stress state after excavation, and there was a stress concentration area about 10 m before the working face.

First weighting was caused by "O-X" fracture to the main roof when the working face advanced at a certain distance [44–46]. The "O" ring was divided by the tensile rupture line.
and pressed rupture line into four plates, namely, master plates A and B and slave plates C and D. After first weighting, a visible circular arc transition area appeared on the main roof above the head and tail of fully mechanized caving faces, which formed an arc-shaped triangular block.

5.2.2. Periodic Rupture Rule of Main Roof. After first weighting, a half "O" ring appeared in the main roof displacement diagram with excavation, the half "O" ring gradually grew outwards, and its displacement decreased from inside to outside [47–49]. When the main roof ruptured, the half "O" ring was the largest, and periodic weighting was caused. The periodic rupture rule of the main roof in the excavation process can be briefly described according to the vertical displacement diagram and vertical stress diagram of the primary rupture of the main roof at different mining heights, as shown in Figure 10.

As shown in Figure 10, periodic weighting was caused by "O-X" fracture to the main roof when the working face advanced at a certain distance. After periodic weighting, a visible circular arc transition area appeared on the main roof above the head and tail of fully mechanized caving faces, which formed an arc-shaped triangular block.
5.3. Stress Distribution Characteristics of Surrounding Rock on Ends Affected by Mining. The roadway is often next to coal pillar, so stability of the coal pillar is closely related to stability of the roadway [50]. Stress of the coal pillar has a great impact on roadway deformation, and stress of the roadway declines significantly with unloading after roadway deformation. Analyzing stress of the coal pillar and solid coal on the right of the roadway can effectively study stability of the roadway.

Figure 12: Change of vertical stress of solid coal on the right of coal pillar and roadway with length of end empty section (15 m coal pillar).

Figure 13: Change of vertical stress of solid coal on the right of coal pillar and roadway with length of end empty section (20 m coal pillar).

Around the roadway when the coal pillar widths were 15 m or 20 m, with the left two peaks at the coal pillar, which indicated stable stress of the coal pillar, and the right peak at the solid coal on the right of the roadway.

(1) When the coal pillar width remained unchanged, the stress peak of the solid coal in the roadway tended to decrease with increase in end length, while the stress peak in the coal pillar tended to increase.

(2) When the coal pillar width remained unchanged and the end length was small, the vertical stress peak of the roadway was greater than that of the coal pillar.
When the end length was large, the vertical stress peak of the roadway was smaller than that of the coal pillar. When the end length was small, the solid coal of the roadway mainly suffered from the load. When the end length increased, the stress moved to the coal pillar direction and the load gradually shifted to the coal pillar (3) As the end length increased, the stress peak in the coal pillar was far away from the roadway and tended to increase. However, the position of the stress peak of the solid coal on the right changed slightly.

6. Field Measurement of Overburden Deformation in Fully Mechanized Caving Mining

6.1. Underground Pressure Rule of Fully Mechanized Caving Faces

6.1.1. KJ216-A-Type Working Resistance Monitoring System. 140 hydraulic supports were adopted for the fully mechanized caving faces of the experimental mine, including 128 supports (ZF15000/27/43 type) for the working face, 4 transition supports (ZFG15000/27/43A type), 1 head and tail support (ZFP13800/26/40 type), and 1 group of end supports (ZFT27600/23/40 type).

Working resistance of hydraulic supports on fully mechanized caving faces of the experimental mine was collected using the KJ216-A- (Figure 14) type working resistance monitoring system. A monitoring line was set every 7 of 140 hydraulic supports, namely, 7# support-1# monitoring line, 14# support-2# monitoring line, and so on (Figure 15), and support (ZFP13800/26/40 type), and 1 group of end supports (ZFT27600/23/40 type).

Table 4: Related data of working resistance of hydraulic supports (in the upper part of working face).

<table>
<thead>
<tr>
<th>Month</th>
<th>Number of hydraulic support</th>
<th>Pressure step (m)</th>
<th>Face advanced distance (m/d)</th>
<th>Dynamic load factor</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>28#, 42#</td>
<td>16.15</td>
<td>2.9</td>
<td>1.16</td>
</tr>
<tr>
<td>2</td>
<td>28#, 42#</td>
<td>16.30</td>
<td>3.1</td>
<td>1.26</td>
</tr>
<tr>
<td>3</td>
<td>7#, 28#</td>
<td>10.00</td>
<td>4.5</td>
<td>1.16</td>
</tr>
<tr>
<td>4</td>
<td>42#, 49#</td>
<td>14.78</td>
<td>6.3</td>
<td>1.16</td>
</tr>
<tr>
<td>5</td>
<td>42#, 49#</td>
<td>14.78</td>
<td>6.3</td>
<td>1.16</td>
</tr>
<tr>
<td>6</td>
<td>7#, 35#</td>
<td>10.00</td>
<td>4.5</td>
<td>1.21</td>
</tr>
<tr>
<td>7</td>
<td>7#, 35#</td>
<td>10.26</td>
<td>4.7</td>
<td>1.23</td>
</tr>
<tr>
<td>8</td>
<td>42#, 49#</td>
<td>10.80</td>
<td>6.3</td>
<td>1.18</td>
</tr>
</tbody>
</table>
A total of 19 monitoring lines were set. A monitoring substation was installed on every monitoring line, and pressure data of pillars before and after supports was collected automatically every 5 min when the working face advanced and transmitted to the ground dispatching room in real time for analysis.

6.1.2. Periodic Weighting of Working Face. According to field observation from January to August and analysis of working resistance of supports of the mine pressure monitoring substation in the upper part of the working face, related data of working resistance of hydraulic supports in the upper, middle, and lower parts of the working face from January to August is shown in Tables 4–6.

### Table 5: Related data of working resistance of hydraulic supports (in the middle part of working face).

<table>
<thead>
<tr>
<th>Month</th>
<th>Number of hydraulic support</th>
<th>Pressure step (m)</th>
<th>Face advanced distance (m/d)</th>
<th>Dynamic load factor</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>70#, 77#</td>
<td>15.50</td>
<td>6.4</td>
<td>1.50</td>
</tr>
<tr>
<td>2</td>
<td>70#, 77#</td>
<td>14.50</td>
<td>6.4</td>
<td>1.46</td>
</tr>
<tr>
<td>3</td>
<td>70#, 77#</td>
<td>5.50</td>
<td>6.2</td>
<td>1.38</td>
</tr>
<tr>
<td>4</td>
<td>70#, 77#</td>
<td>8.60</td>
<td>5.6</td>
<td>1.42</td>
</tr>
<tr>
<td>5</td>
<td>63#, 70#</td>
<td>14.37</td>
<td>8.6</td>
<td>1.50</td>
</tr>
<tr>
<td>6</td>
<td>63#, 77#</td>
<td>5.50</td>
<td>8.5</td>
<td>1.45</td>
</tr>
<tr>
<td>7</td>
<td>63#, 70#</td>
<td>6.80</td>
<td>8.6</td>
<td>1.50</td>
</tr>
<tr>
<td>8</td>
<td>63#, 70#</td>
<td>6.90</td>
<td>8.5</td>
<td>1.52</td>
</tr>
</tbody>
</table>

### Table 6: Related data of working resistance of hydraulic supports (in the lower part of working face).

<table>
<thead>
<tr>
<th>Month</th>
<th>Number of hydraulic support</th>
<th>Pressure step (m)</th>
<th>Face advanced distance (m/d)</th>
<th>Dynamic load factor</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>105#, 112#</td>
<td>16.50</td>
<td>6.7</td>
<td>1.35</td>
</tr>
<tr>
<td>2</td>
<td>105#, 112#</td>
<td>14.62</td>
<td>6.3</td>
<td>1.34</td>
</tr>
<tr>
<td>3</td>
<td>105#, 112#</td>
<td>15.20</td>
<td>5.8</td>
<td>1.38</td>
</tr>
<tr>
<td>4</td>
<td>105#, 112#</td>
<td>16.48</td>
<td>6.3</td>
<td>1.43</td>
</tr>
<tr>
<td>5</td>
<td>91#, 98#</td>
<td>14.37</td>
<td>6.3</td>
<td>1.41</td>
</tr>
<tr>
<td>6</td>
<td>91#, 105#</td>
<td>17.40</td>
<td>5.5</td>
<td>1.32</td>
</tr>
<tr>
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<td>91#, 98#</td>
<td>8.20</td>
<td>6.3</td>
<td>1.37</td>
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<tr>
<td>8</td>
<td>91#, 98#</td>
<td>8.50</td>
<td>5.5</td>
<td>1.34</td>
</tr>
</tbody>
</table>

Figure 16: Destruction of surface subsidence map work.

6.2. Surface Movement Failure Characteristics of Fully Mechanized Caving Faces. Due to large coal thickness of fully mechanized caving faces of the thick unconsolidated stratum of the experimental mine, working face mining...
had a marked impact on surface movement deformation and overburden movement and destruction spread to earth surface (Figure 16). Upon first weighting, there were major fractures on the surface vertical to the working face, which were accompanied by some tiny fractures. Upon periodic weighting, there were small and shallow periodic main fractures on the surface vertical to the working face, which became deeper and wider as the working face advanced. After a period, “roof step” appeared behind the working face and further evolved into the “funnel-type” settlement.

7. Conclusion

Through theoretical analysis, experimental research, numerical calculation, and field observation, the ultra-high coal seam with thin bedrock is systematically analyzed, and its deformation and failure mechanism is revealed. The relevant research results were successfully implemented on site, effectively ensuring the safety, high yield, and high efficiency of the mine. The specific conclusions are as follows:

1) The “sidestepped rock beam” structure was formed after sliding instability of the roof “short masonry beam” structure. The “sidestepped rock beam” structural mechanics model was built specific to the architectural feature of the roof after sliding, and its stability criterion was worked out through mechanical calculation and analysis.

2) Surrounding rock in the middle of the working face had small horizontal displacement and that on both ends had large horizontal displacement. After working face mining, surrounding rock on both ends formed a masonry beam structure, horizontal displacement at the top left corner and bottom right corner of block B reached the peak, and the peak horizontal displacement of surrounding rock on both ends tended to increase with increase in buried depth. Surrounding rock in the middle of the working face had large vertical displacement and that on both ends had small vertical displacement, and vertical displacement was in the shape of an “inverted circular truncated cone” from the middle of the working face to both ends.

3) After working face mining, an “O” ring appeared on the main roof, which grew outwards with excavation and reached the largest upon first weighting and whose displacement decreased from inside to outside. With a certain range, first weighting pitch decreased with increase in mining height and tended to be stable when mining height exceeded a certain value. At large mining height, rock around the rupture line of the largest “O” ring of the main roof “turned over” in first weighting. After first weighting, a half “O” ring appeared in the main roof displacement diagram, which gradually grew outwards with excavation and whose displacement decreased from inside to outside. The half “O” ring was the largest when the main roof ruptured, and consequently, periodic weighting was caused.

4) Upon periodic weighting, there were small and shallow periodic main fractures on the surface vertical to the working face, which became deeper and wider as the working face advanced. After a period, “roof step” appeared behind the working face and further evolved into “funnel-type” settlement.

5) The deficiency can be analyzed by the discrete element numerical calculation method under different combination conditions such as coal pillar width, length of noncaving section, and coal seam thickness, and the variation law of the overburden subsidence coefficient with coal seam thickness is obtained.

6) When the width of the coal pillar is constant, the peak stress of the solid coal in the roadway decreases with the increase of the end length. The peak stress of the coal pillar tends to increase. When the width of the coal pillar is constant and the end length is small, the peak value of the roadway vertical stress is greater than that of the coal pillar vertical stress. When the end length is large, the peak value of the vertical stress of the roadway is smaller than that of the coal pillar. When the end length is small, the roadway solid coal is mainly loaded. With the increase of the end length, the stress moves towards the coal pillar, and the load gradually moves towards the coal pillar. With the increase of the end length, the peak stress in the coal pillar is far away from the roadway and tends to increase. However, the position of the stress peak of the right solid coal has little change.

Data Availability

The data used to support the findings of this study are available from the corresponding author upon request.

Conflicts of Interest

The authors declare no conflicts of interest.

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