Hindawi
Advances in Civil Engineering
Volume 2018, Article ID 6871820, 15 pages
https://doi.org/10.1155/2018/6871820

Research Article

Effect of Mining Thickness on Overburden Movement and Underground Pressure Characteristics for Extrathick Coal Seam by Sublevel Caving with High Bottom Cutting Height

Yongkang Yang,[1,2,3,4] **Yanrong Ma**,[1] **Chunxu Ji**,[1] **Tianhe Kang**,[1] **and Xingyun Guo**[1]

[1]*Key Laboratory of In-situ Property-improving Mining of Ministry of Education, Taiyuan University of Technology, Taiyuan, Shanxi 030024, China*
[2]*State Key Laboratory of Coal Resources and Safe Mining, China University of Mining & Technology (Beijing), 100083 Beijing, China*
[3]*State Key Laboratory for Geomechanics & Deep Underground Engineering, China University of Mining & Technology, Xuzhou 221116, Jiangsu, China*
[4]*Key Laboratory of Safety and High-efficiency Coal Mining, Ministry of Education, Anhui University of Science and Technology, Huainan 232001, China*

Correspondence should be addressed to Tianhe Kang; kangtianhe@163.com

Received 25 July 2018; Accepted 15 October 2018; Published 22 November 2018

Guest Editor: Jian Sun

Because the coal seam is particularly thick and the mining intensity is large, the mining of extremely thick coal seams often causes a wide range of disturbed fractures, which in turn induces the phenomenon of strong underground pressure such as induced support crushing and water inrush. Through theoretical analysis, laboratory similarity simulation test, and other methods, this paper studies the effect of mining thickness on overburden movement and underground pressure characteristics for extremely thick coal seams by sublevel caving with high bottom cutting height. Some conclusions can be drawn as follows: (i) under the "beam-hinged cantilever beam rocks" structure theory, the rock pillar thickness which needs to be controlled increases linearly as a function of mining thickness is achieved, and the reason of increased of support resistance in full-mechanized caving mining in extremely thick seams is explained in the theory; (ii) based on the results of the theoretical analysis and the lab simulation tests, the law of the abutment pressure peak is inverse to the full-seam mining thickness, and the distance between abutment peak and working face is proportional to the full-seam mining thickness, that is to say that the damage range of overlying strata increased; (iii) there are three working states of loading support in extrathick coal seams, such as normal circumstance, lower main roof pressure, and higher main roof pressure, meanwhile these states keep changing; (iv) under the guarantee of stope safety conditions, due to lower support strength, it will benefit the special thick seam top-coal caving under normal circumstance; (v) increasing the supporting strength can balance the impact loading under the lower main roof pressure, guaranteeing valid support for roof strata; (vi) by releasing high pressure, due to lower production, lower recovery rate of coal and other measures guarantee the stability of the stope support in the case of the higher main roof pressure.

1. Introduction

The extrathick coal seam by sublevel caving with more than 3.5 m bottom cutting height has obvious technical advantages, such as increased cutting height, optimized caving ratio, increased caving by mining pressure, increased ventilation section of working face, and reduced ventilation resistance. This technique has been used in large scale in Tashan coal mine, Huating coal mine, Xiagou coal mine, etc.

Because the coal seam is particularly thick and the mining intensity is large, the mining of extremely thick coal seams often causes a wide range of disturbed fractures,

which in turn induces the phenomenon of strong underground pressure such as induced water bursting and support crushing. This indicates that it is not appropriate to use traditional mining pressure theory to directly guide the fully mechanized mining of ultrathick coal seams.

Many scholars have studied the fully mechanized mining of thick coal seams from different ways of thinking, such as the mechanical model of support [1], the external load theory of support [2], quantitative relationship analysis of support surrounding and rock [3], the relationship between different roof structures and support resistance [1, 3–6], factors influencing the rate of the coal caving ratio in fully mechanized top-coal caving in large mining height [7, 8], principle of the working resistance of the support jumps, and the support is not unique [9]. However, there are few literatures in theory and practice on the movement rule of overburden in ultrathick coal seams with a thickness of more than 14 m and surrounding rock control in mining field [10]. The different thickness of coal seams leads to the movement of overburden and the control of surrounding rock, which is different from the comprehensive sublevel mining and general comprehensive mining of large mining height. The effects of mining thickness on the surrounding rock stress, failure, displacement, and support load should be studied, which is helpful to understand the force source of mine pressure and provide reference for surrounding rock control technology.

Through theoretical analysis, laboratory similarity simulation test, and other methods, this paper studies the effect of mining thickness on overburden movement and underground pressure characteristics for extremely thick coal seams by sublevel caving with high bottom cutting height, by taking Xiegou No.13 coal mine as the background, as the hard coal seam in the soft roof.

2. The Theoretical Analysis of Mining Thickness Effect

2.1. Overburden Structure and Support Pressure. The main roof rock is a transition layer, which belongs to a special "semibearing structure" in the mechanical structure. After the main roof fracture, it is the main load acting on the support and an important bearing structure [10–17]. The immediate roof and the main roof are the main objects of roof control in the working face, and the load on the support of the working face acted from the roof rock strata. Its essence is to study the interaction between the supporting and surrounding rock in the transition from the original rock equilibrium state to another equilibrium state. The key of the study is to determine the main roof.

The Kuznetsov hinged block criterion $\Delta_j > \Delta$ is an effective method [18], while Δ_j is the rock limit sink [17], $\Delta_j = \sum h - (ql^2/4kh[\sigma_c])$, where $[\sigma_c] = (0.3 - 0.35)R_C$, R_C is the uniaxial compressive strength, Σh is the thickness of immediate roof collapse, q is the linear load, and l is the roof weighting step, and Δ is the unfilled height between main roof and immediate roof. If $\Delta_j > \Delta$, when the roof is broken, the hinged structure of self-bearing will be formed, and if $\Delta_j < \Delta$, the main roof strata are transformed into the immediate roof strata.

As shown in Figure 1, the immediate roof is a "cantilever beam" structure, which consists of the irregular falling zone and regular caving zone rock mass. The main roof is a balanced structure of "articulated rock beam," which is composed of rock mass in the fractured zone.

If M is the mining thickness and K_p is the hulking coefficient, there will be a relation for the unfilled height Δ between the main roof strata:

$$\Delta = \Sigma h + M - K_P \cdot \Sigma h = M - \Sigma h\left(K_P - 1\right). \quad (1)$$

The unloaded immediate roof should be satisfied with $\Delta_j < \Delta$, as

$$\Delta_j = \sum h - \frac{ql^2}{4kh\left[\sigma_c\right]} < \Delta = M - \Sigma h\left(K_P - 1\right). \quad (2)$$

As the mining thickness increases, the probability of confirmation this formula increases, the range of immediate roof increases significantly, while the roof activity space increases in the goaf area, and the horizontal and vertical surrounding rock movement of roof increases significantly, so it is necessary to control the range of rock layers.

The 13# coal mine is 15 m in the Xiegou coal mine, which is medium hard coal seam; the overburden synthesis is shown in Table 1, and the physical and mechanical parameters are shown in the literature [19]. If the coal extraction rate is 90%, the average fracturing expansion coefficient of the immediate roof and top coal in the goaf is $K_p = 1.25$.

If $M = 3$ m, the unfilled height $\Delta_{3m} = 3 \times 90\% - (1.25 - 1)(3 \times 10\% + 1.5 + 2.07) = 1.73\,\text{m} < 2.02\,\text{m}$, while the No. 50 fine grained sandstone is the main roof, and the immediate roof that needs to be controlled is 3.57 m.

If $M = 6$ m, $\Delta_{6m} = 6 \times 90\% - (1.25 - 1)(6 \times 10\% + 1.5 + 2.07 + 2.02 + 2.5 + 1 + 0.82 + 1.49) = 1.88\,\text{m} < 2.08\,\text{m}$, while the No. 45 sandy mudstone is the main roof, and the immediate roof that needs to be controlled is 13.48 m, including the No. 46 to No. 52 rock strata.

In a similar way, $\Delta_{9m} = 2.89\,\text{m} < 6.5\,\text{m}$, while the No. 40 marlstone is the main roof, and the immediate roof that needs to be controlled is 19.94 m, including the No. 41 to No. 52 rock strata.

$\Delta_{15m} = 5.22\,\text{m} < 5.39\,\text{m}$, while the No. 36 coarse sandstone is the main roof, and the immediate roof that needs to be controlled is 31.62 m, including the No. 37 to No. 52 rock strata.

$\Delta_{20m} = 6.12\,\text{m} < 8.31\,\text{m}$, while the No. 30 medium sandstone is the main roof, and the immediate roof that needs to be controlled is 45.53 m, including the No. 31 to No. 52 rock strata.

$\Delta_{25m} = 5.28\,\text{m} < 9.65\,\text{m}$, while the No. 25 coarse sandstone is the main roof, and the immediate roof that needs to be controlled is 66.37 m, including the No. 26 to No. 52 rock strata.

The relation between the unfilled height, thickness of immediate roof which needs to be controlled, and the

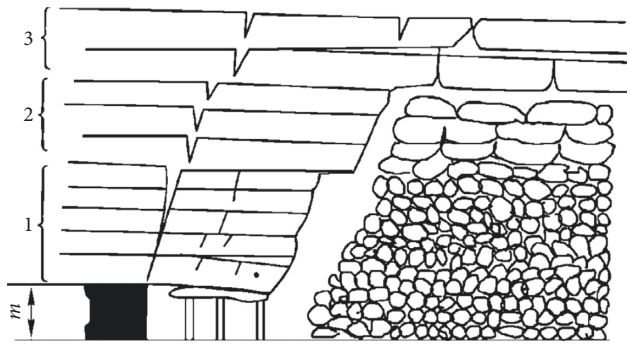


Figure 1: Structure of the roof. 1: irregular fall zone; 2: regular caving zone; 3: fractured zone.

Table 1: Rock formation columnar section of the 13# coal seam in the Xiegou coal mine.

No.	Name	Thickness (m)
1	Loess	26.84
2	Packsand	7.95
3	Sandy mudstone	12.18
4	Packsand	8.50
5	Sandy mudstone	20.30
6	Medium sandstone	12.06
7	Sandy mudstone	31.49
8	Packsand	4.60
9	Sandy mudstone	21.85
10	Mudstone	25.22
11	Sandy mudstone	3.60
12	Medium sandstone	17.30
13	Sandy mudstone	6.50
14	Packsand	13.76
15	Mudstone	6.56
16	Packsand	3.82
17	Mudstone	4.52
18	Medium sandstone	7.84
19	Sandy mudstone	4.12
20	Packsand	7.02
21	Sandy mudstone	9.51
22	Medium sandstone	11.92
23	Mudstone	15.02
24	6# coal mine	1.50
25	Gritstone	9.65
26	Siltstone	3.71
27	Sandy mudstone	1.00
28	8# coal mine	5.69
29	Mudstone	2.13
30	Medium sandstone	8.31
31	Marlstone	2.12
32	Mudstone	2.01
33	9# coal mine	0.51
34	Sandy mudstone	2.43
35	Mudstone	1.45
36	Gritstone	5.39
37	Mudstone	2.32
38	10# coal mine	0.60
39	Mudstone	2.26
40	Marlstone	6.50
41	Sandy mudstone	1.53
42	11# coal mine	0.61
43	Mudstone	1.76
44	Limestone	2.56

Table 1: Continued.

No.	Name	Thickness (m)
45	Sandy mudstone	2.08
46	Mudstone	1.49
47	12# coal mine	0.82
48	Mudstone	1.00
49	Medium sandstone	2.50
50	Micropsammite	2.02
51	Mudstone	2.07
52	Sandy mudstone	1.50
53	13# coal mine	5.25
54	Dirt band	0.25
55	13# coal mine	3.25
56	Dirt band	0.25
57	13# coal mine	4.15
58	Dirt band	0.30
59	13# coal mine	1.55
60	Mudstone	2.00
61	14# coal mine	0.33
62	Mudstone	3.05
63	Sandy mudstone	2.00
64	Limestone	1.29
65	Mudstone	1.00
66	15# coal mine	0.44
67	Sandy mudstone	3.00
68	Mudstone	2.05
69	Siltstone	2.56
70	Packsand	2.41

mining thickness is be shown in Figure 2. Obviously, the unfilled height in goaf increases exponentially with the increase of mining thickness, and the thickness of the immediate roof which needs to be controlled increases linearly with the mining thickness.

With the advance of the working face, the overlying rock strata of "cantilever girder-hinged rock beam" in the mining field have subsidence and pressure on the top coal. Along with the increase of the subsidence, the compression on the vertical and displacement in the horizontal of the top coal are increasing, the crack is also gradually reduced, and then the stiffness of the top coal gradually becomes stronger, which can transfer the deformation pressure from the immediate roof and the main roof to the hydraulic support. As the external load of the hydraulic support combines with the

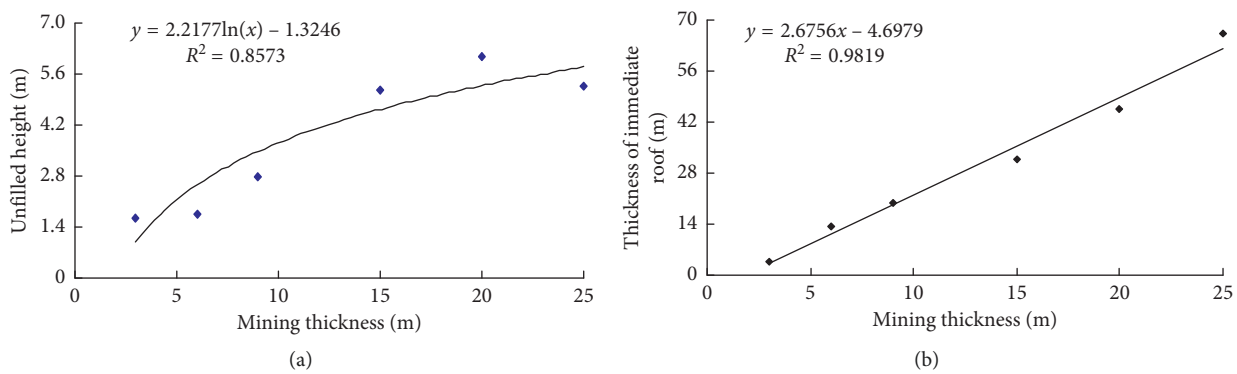


Figure 2: The relation between the unfilled height, thickness of immediate roof which needs to be controlled, and the mining thickness. (a) Unfilled height and mining thickness. (b) Thickness of the immediate roof which needs to be controlled.

weight of the top coal, the deformation stress from the main roof and the immediate roof increases. With the expansion of the immediate roof range, the deformation pressure of the main roof and the immediate roof will inevitably increase, and the working resistance of the support will increase.

2.2. Change of Abutment Pressure. The law of abutment pressure distribution in the plastic zone in front of the working face is given below:

$$\sigma_y = \tau_0 ctg\varphi \cdot \frac{1+\sin\varphi}{1-\sin\varphi} e^{(2fx/M)(1-\sin\varphi/1+\sin\varphi)}, \quad (3)$$

where f is the friction coefficient between layers, M is the thickness of the coal seam, φ is the internal friction angle of coal, x is the distance from any point in the plastic zone to the coal wall, and $\tau_0 ctg\varphi$ is the self-bearing capacity of the coal.

If $\sigma_y = K\gamma H$, the distance between the peak point of abutment pressure and coal wall can be obtained:

$$x_0 = \frac{M}{2f} \cdot \frac{1+\sin\varphi}{1-\sin\varphi} In \frac{K\gamma H(1-\sin\varphi)}{\tau_0 ctg\varphi(1+\sin\varphi)}. \quad (4)$$

Therefore, with the increase of the mining thickness, the peak point of the abutment pressure is transferred to the deep part of the coal wall and the influence range of the abutment pressure is increased.

3. Research on the Effect of Mining Thickness by the Similarity Simulation Test

3.1. Test Apparatus. A large planar strain test device with length × height × thickness = 4370 mm × 4600 mm × 200 mm was used, and the surrounding area was restrained by No. 20 channel steel and 25 mm thick organic glass plate constraints.

According to the prototype conditions and test equipment, the specific similar conditions that the model should meet are determined:

(i) Geometric proportion: $a_l = 1/200$

(ii) Volume to weight ratio: $a_\gamma = (1.5/2.5) = 0.6$ (coal); $a_\gamma = (1.7/2.5) = 0.68$ (rock)

(iii) Similarity ratio of stress and elastic modulus: $a_{\sigma E} = a_l \cdot a_\gamma = 0.005 \times 0.68 = 0.0034$(rock); $a_{\sigma E} = a_l \cdot a_\gamma = 0.005 \times 0.6 = 0.003$(coal)

(iv) Load similarity ratio: $a_F = a_l^3 \times a_\gamma = 8.5 \times 10^{-8}$

(vi) Similarity ratio of strain and Poisson stress: $a_\varepsilon = 1$

(vii) Time similarity ratio: $a_t = \sqrt{a_l} = \sqrt{1/200} = 0.0707$

3.2. Experiment Scheme and Design. River sand was selected as aggregate, with lime, calcium carbonate, and gypsum as the cementing materials and borax as the coagulant. Considering the discontinuity caused by the influence of joints and cracks on rock materials, the crack coefficients of 0.7 and 0.6 are, respectively, considered for the mechanical properties of rock and coal measured in the laboratory. Then, according to the similarity theory [19], the mechanical parameters of the layered materials in the model are obtained, and the material proportion is selected to calculate the amount of materials [19]. The model is laid in layers along the horizontal direction, and the mica powder is sprinkled between layers to simulate the bedding face of the coal seam. Vertical to the surface of the laid coal bed, the bedding is irregularly cut on the rock layer and mica powder is sprinkled in the simulation joints. During the laying process, the operation is carried out strictly in accordance with the actual simulated size of each coal layer. The self-made mould is used for leveling and compacting. The upper part of the model is free end, which can move along with the rock strata.

The simulated support consists of a resistance strain gauge, a compressible strong toughness polyurethane gasket, and a rigid upper and lower organic plate. The working resistance of the simulated bracket was calibrated before the test. According to the dynamic strain gauge analysis, the corresponding relation between strain and stress is load $Y = a \times$ strain gauge reading $+ b$, and load Y is equivalent to the weight of the rock pillar whose bulk weight is r and height is h, so $Y = rh$. According to the geometric relationship, the actual weight of the rock pillar can be obtained, and the working resistance P can be further obtained, $P = \gamma H \times$ effective support area S.

Nikon DTM-531E total station was used to measure the overlying strata movement. The YJ-5 static dynamic strain

(a) (b)

Figure 3: Test system. (a) The diagram of displacement measuring points and displacement measured by the total station. (b) The connection of strain brick and testing instrument.

gauge and automatic data acquisition system were used to collect the dependent variables of sensors embedded in the model, as shown in Figure 3.

The mining height of the three models was 2 cm (actual 4 m), the top coal was 2.5 cm (actual 5 m), 5.5 cm (actual 11 m), and 10.5 cm (actual 21 m), respectively, and the caving ratios were 1.25, 2.72, and 5.25, respectively.

The open-off cut is width × height = 4 cm × 2 cm, and then the simulated support is imbedded. The time of model 1 d is $T_m = 24 \times \sqrt{a_l} = 24 \times \sqrt{1/200} = 1.69\,\text{h} = 101.4\,\text{min}$, while 8 h with the corresponding model time is $T'_m = (101.4/3) = 33.8\,\text{min}$. The working face propulsion speed is determined with 40 mm/d (8 m/d), as excavation was performed for 2 cm every 33.8 min.

3.3. Test Results and Analysis

3.3.1. Evolution Processes and Characteristics of Roof and Top Coal. The evolution process of mining roof caving in coal with a thickness of 9 m is shown in Figure 4. When the working face advances 32 m, a horizontal separation layer appears at 16 m (8 cm) above the top coal, and a horizontal bedding layer is also seen at 6 m (3 cm) above the top coal. When the working face advances 40 m, the top coal separates from the top plate and falls for the first time. When the working face advances 52 m, the separation layer appeared at 19 m (9.5 cm) above the top coal, and a vertical crack of 15 m (7.5 cm) appeared at the same time, approaching the bottom of No. 36 subcritical layer. Advancing to 60 m, the 19 m lower roof overall slowly bends down and touches the gangue, caving angle 60°. Advancing to 80 m, the full thickness of the top-coal caves, the immediate roof of 5 m (2.5 cm) range above the top coal will be collapsed. Advancing to 90 m, a separation occurs in the middle key stratum layer of No. 30, which was 37 m (18.5 cm) from the top coal, and an inclined crack appears that has 44° with the horizontal direction, which is along the separating location and the coal wall of the working face. Advancing to 112 m, the key stratum layer of No. 36 caves along the delamination, while a horizontal bedding derives from the diagonal split that extended to 8 m (4 cm) and begins to expand to the working face of the coal wall position, extending to 52°. Advancing to 120 m, the No. 36 subcritical layer collapses as a whole along the aforementioned crack. Advancing to 172 m, the roof caving falls to the bottom of the No. 30 subcritical layer, with a maximum caving thickness of 45.53 m (22.77 cm). Advancing to 184 m, the lower roof including the No. 36 subcritical layer collapses again, which was under the protection of the key stratum layer of No. 30, and the front caving angle is 36°. Advancing to 220 m, the compound key stratum of the No. 30 and No. 36 layers breaks off, and it creates the front caving angle of 57°. Advancing to 256 m, the rock below the critical layer of No. 25 dissociates and slowly sinks, while the No. 25 key stratum significantly bends down, which drives the separation layer below the No. 22 subcritical layer and obvious displacement above the No. 22 subcritical layer; the front caving angle is 60° and the back caving angle is 57°.

Some typical roof collapse of mining caving in coal with a thickness of 15 m and 25 m is, respectively, shown in Figures 5 and 6. The comprehensive analysis and test results show that the movement rule of the overburden strata is different while in different thick coal seams.

(1) With the increase of the mining thickness, the height of the overburden movement damage increases, whereas the maximum collapse zone height decreases when the mining thickness is greater than 15 m, as shown in Table 2 and Figure 7.

(2) The movement deformation and failure of overlying strata with different mining thicknesses all lead to the collapse arch. With the increase of mining thickness, the collapse angles of both front and back tend to increase, and the increase of the back caving angle is more obvious, indicating that the flattening rate of the collapse arch tends to decrease with the increase of mining thickness.

(3) With the increase of mining thickness, the thickness of the roof entering the caving zone increases. The roof strata in the collapse zone and the height of the fault zone are mainly composed of key strata, which have obvious composite deformation, subsidence, and failure. The key layer of the hinged structure can be formed in the thinner coal seam, and the stable hinged structure cannot be formed in the thicker coal seam due to the large amount of rotary deformation, which is manifested as the overall collapse of the "cantilever beam" structure, while the stable hinged

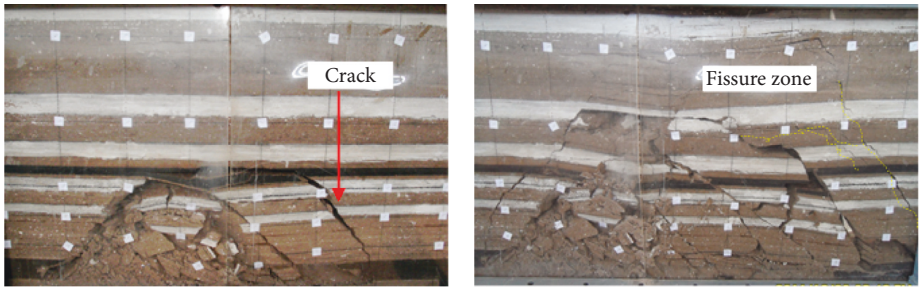
Figure 4: Evolution process of mining roof caving in coal with a thickness of 9 m. (a) 32 m, (b) 40 m, (c) 52 m, (d) 60 m, (e) 80 m, (f) 92 m, (g) 112 m, (h) 120 m, (i) 172 m, (j) 184 m, (k) 220 m, and (l) 256 m.

rock beam structure can be formed at the higher key layer.

(4) The migration and collapse of top coal and roof is a dynamic process, in which the thickness of the rock pillar that should be controlled in the immediate roof is changing, while the thickness and configuration of the lower main roof are also dynamically changing.

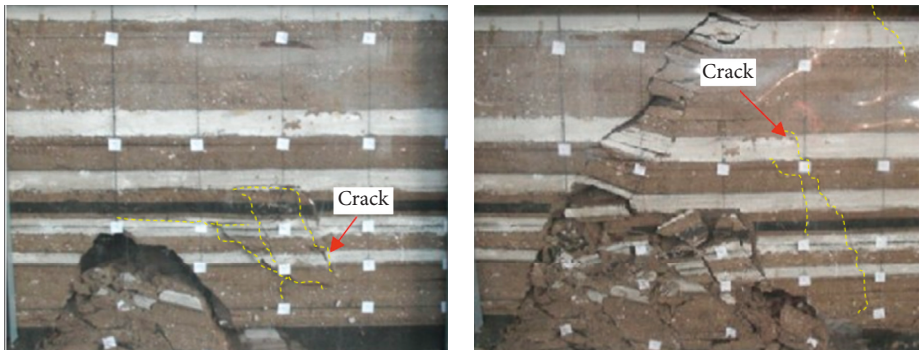
3.3.2. Law of Stress Distribution in Overlying Strata. Figure 8 shows the stress variation curves with different layers of overlying strata in different coal thicknesses with the advancing distance of the working face.

Take the example of 9 m coal seam for illustration, as shown in Figure 8(a). With the advance of the working face, the stress at each layer of the leading working face increases slowly. When the working face advances to 110 m, the stress of each measuring point increases further. When the working face advances to 70 m, the stress of each measuring point increases further. There is no obvious peak value at the measuring points 250 m and 150 m from the coal seam roof, and other measuring points peak at 12~24 m in front of the coal wall. Subsequently, the abutment pressure at the measured point of 10 m from the top plate of the coal seam decreases rapidly, while the stress at other measured points decreases relatively gently. When the working face is pushed forward, the measured stress at 250 m from the roof of the coal seam does not change much, and the stress at other measured points decreases, and the rock body is in the stress reduction zone.



(a) (b)

Figure 5: Some typical roof collapse of mining caving in coal with a thickness of 15 m. (a) 188 m; (b) 296 m.



(a) (b)

Figure 6: Some typical roof collapse of mining caving in coal with a thickness of 25 m. (a) 144 m; (b) 200 m.

Table 2: The movement of overlying strata with different mining thicknesses.

Thickness (m)	Maximum height of the fault zone (m)	Front caving angle (°)	Back caving angle (°)
9	50.84	59	57
15	85.32	60	62
25	115.85	62	68

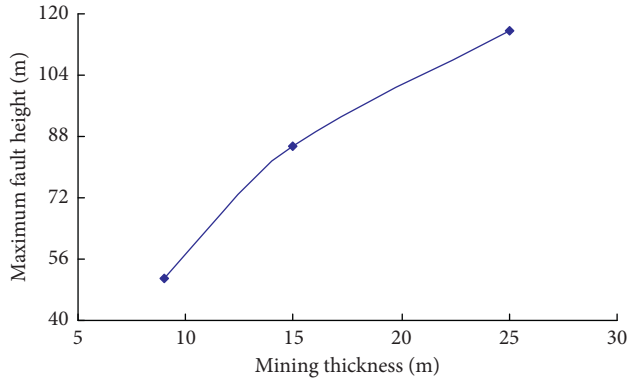


Figure 7: Relationship between the height of the maximum caving zones and the mining thickness.

By the comparison of Figures 8(b) and 8(c), the law of abutment pressure about the mining thickness effect can be obtained:

(1) The stress of different layers in the roof rock of the coal seam changes constantly. As the working face advances, the rock body stress presents the elastic stability zone, slow rising zone, and sharp rising zone and decreasing zone.

(2) The greater the mining thickness, the greater the mining influence on the roof of the coal seam, the

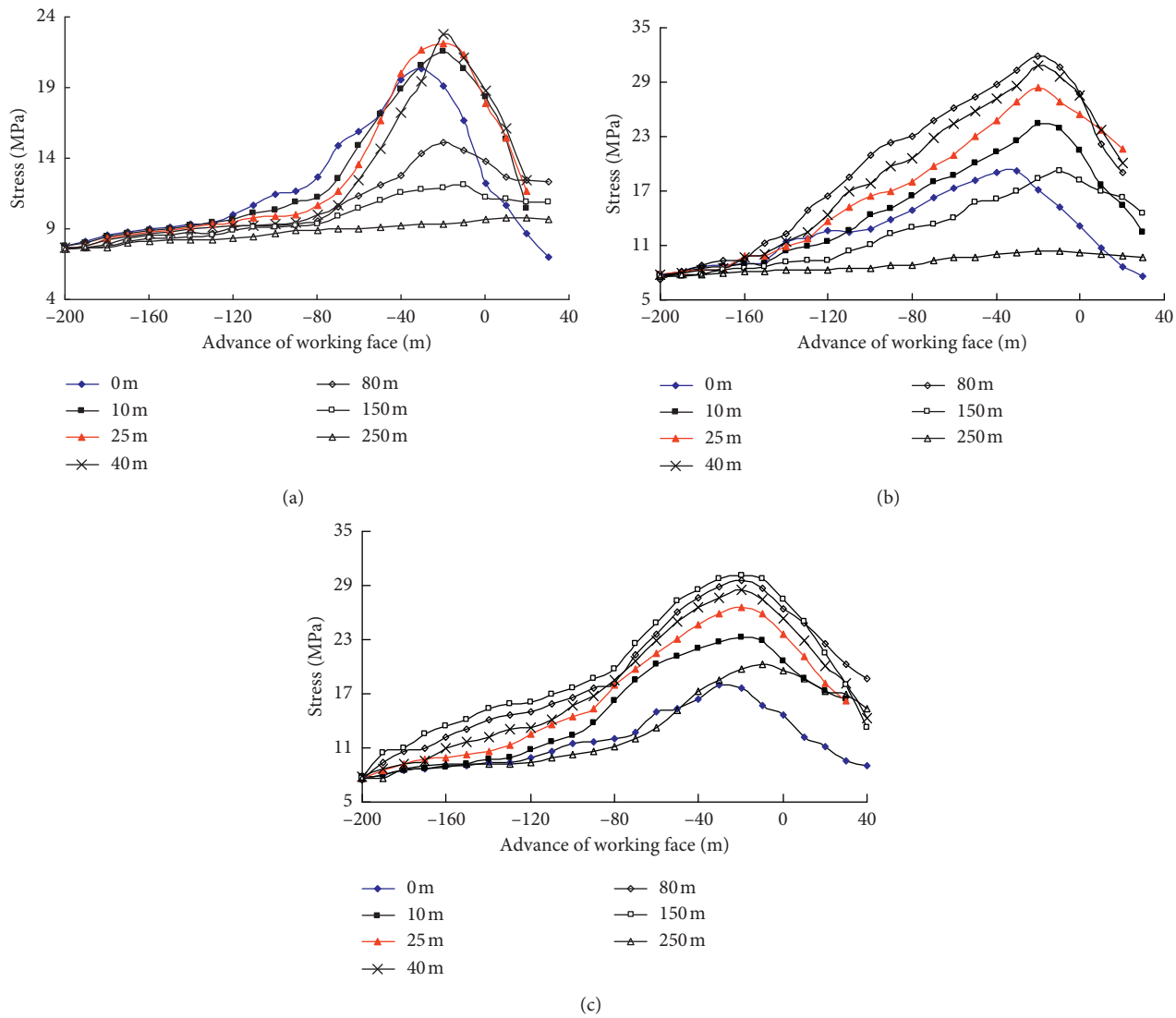


Figure 8: Stress variation curves with different layers of overlying strata in different coal thicknesses with advance of working faces. (a) 9.0 m; (b) 15.0 m; (c) 25 m.

higher the stress peak of the rock body at the higher level, and the greater the influence range of the front supporting pressure (Table 3).

(3) There is a dynamic stress arch in the upper part of the full-seam mining for the extrathick coal seam by sublevel caving mining with high bottom cutting height, which is the main bearing carrier, and the masonry beam or hinged rock beam structure composed of key layers in the arch has certain bearing capacity.

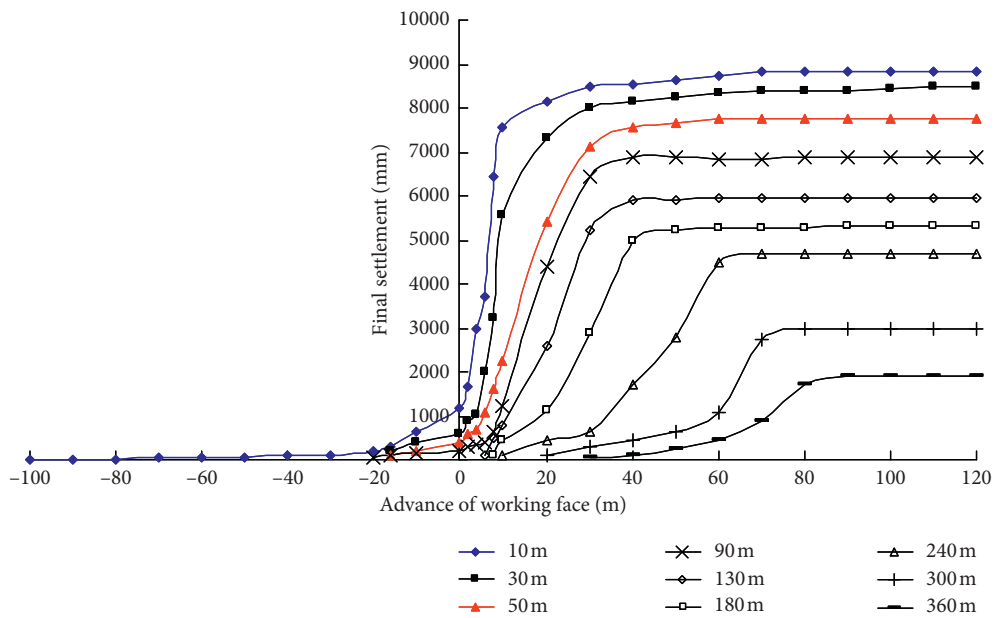
3.3.3. Analysis of Overburden Movement Characteristics. Variety of overlying strata convergence with the advance of working face in different thick coal seams is shown in Figure 9, and the curves of overlying strata movement with the advancing distance of 200 m in different thick coal seams are shown in Figure 10. The effect of mining thickness on overburden movement characteristics can be drawn as follows:

(1) In the mining process of the working face, the rock layer of the immediate roof gradually appears in the abscission layer, which undergoes movement deformation and collapses from the surface to the inside. Because of the different physical and mechanical properties of the overburden strata, there exists the formation of movement characteristic marked by the key strata. With the increase of mining thickness, the chances of the formation-coordinated movement of the overburden rock are greater.

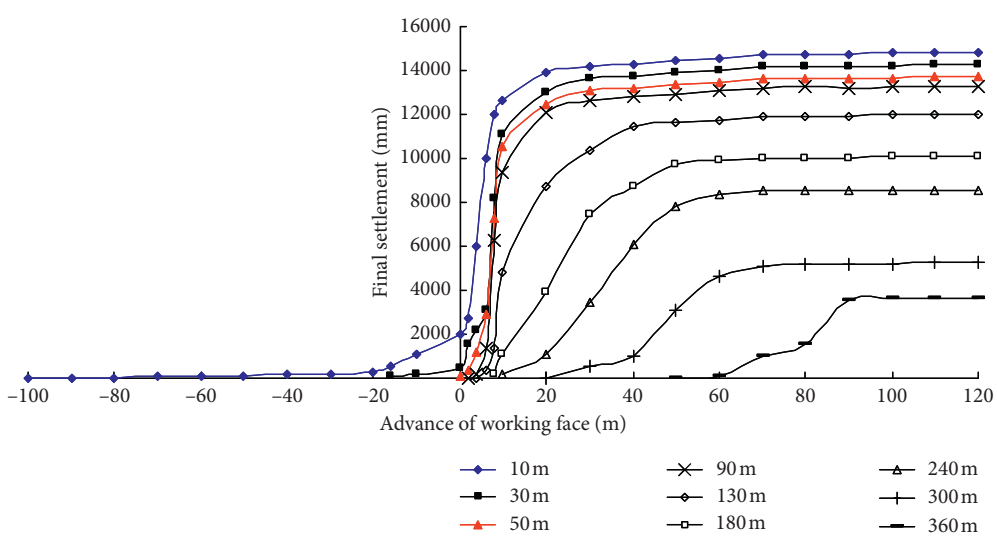
(2) The horizontal and vertical deformations occur in the advancing coal wall of the lower overlying strata near the coal seam, while the upper rock mass far from the coal seam lags behind the coal wall of the working face.

Table 3: Factor of stress concentration and the distance between peak stress and coal wall in different mining thicknesses.

Distance from coal seam roof (m)	9 m		15 m		25 m	
	Factor (k)	Distance (m)	Factor (k)	Distance (m)	Factor (k)	Distance (m)
0	2.26	24	2.14	26	1.99	29
10	2.39	22	2.71	24	2.59	27
25	2.46	21	3.15	23	2.96	25
40	2.53	18	3.42	20	3.17	23
80	1.68	12	3.55	18	3.28	21
150	1.34	4	2.13	13	3.35	19
250			1.15	4	2.26	14



(a)



(b)

Figure 9: Continued.

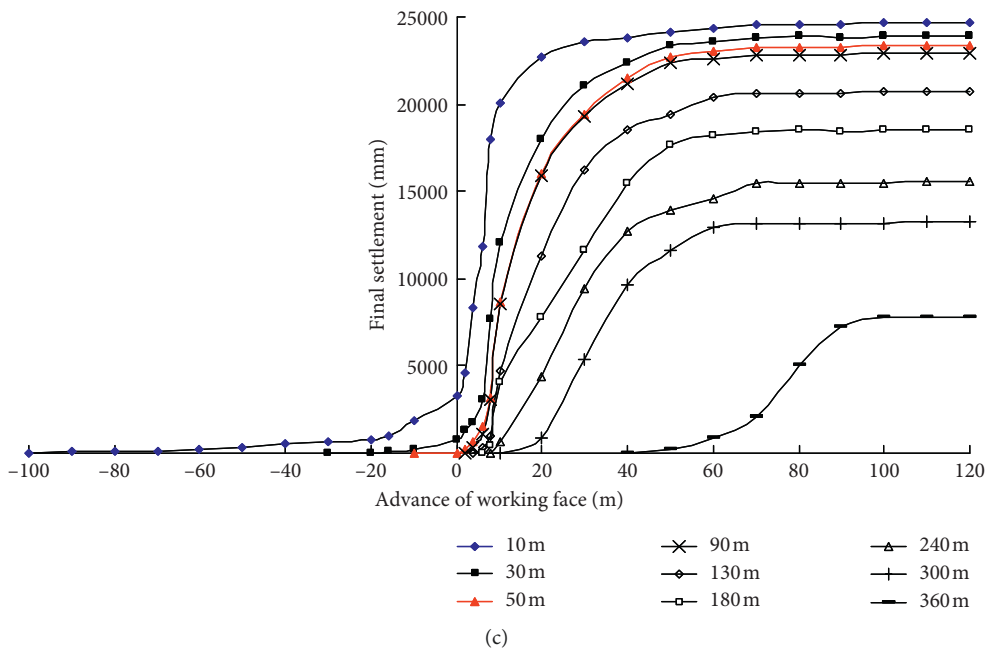


Figure 9: Variety of overlying strata convergence with advance of working face in different thick coal seams. (a) 9.0 m; (b) 15.0 m; (c) 25 m.

(3) All the measuring points have subsidence, and the subsidence is nonlinear. With the increase of mining thickness, the amount and coefficient of subsidence of the same layer of rock mass increase (as shown in Figure 11 and Table 4). It shows that the disturbance range of complex rocks increases nonlinearly with the increase of mining thickness.

(4) As the working face advances, the overburden movement height continues to expand upward, but when the working face advances more than 90 m, the accumulated subsidence amount remains basically unchanged.

(5) The morphological structures of the mobile deformation of different mining thick overburden movement characteristics are basically the same. As the distance from the coal seam increases, the disturbance range and the maximum subsidence amount of each rock layer decrease.

3.4.1. Variation of Working Resistance of the Support. According to the theory of mine pressure, the roof overburden structure of the fully mechanized caving for extremely thick coal seams can be obtained, as shown in Figure 12. Under normal circumstances, a higher main roof can form a "self-loading structure," while the destruction of the immediate roof and the lower main roof will induce the damage to the higher self-loading structure, and the fracture of the higher main roof directly passes to the lower main roof and leads to immediate roof fracture, and they work together in the form of dynamic pressure on the working face of the coal wall and the hydraulic support, as well as disasters occur due to the pressure formation. The relationship between the external load and the mounting of sinking is shown in Figure 13 [18, 19]. When overburden subsidence is large, the coal wall is easily subjected to high pressure leading to the rib fall of the coal wall. When the overburden subsidence is small, it is beneficial to control the coal wall, but it needs strong support resistance. The study on the relationship between support load and surrounding rock is focused on determining the size of the support load, and the key to determining the size of the support load in ultrathick coal seam mining is to determine the height of the rock strata to be controlled.

Support working resistances can be obtained from the material simulation test, taking coal with the thickness 9 m for example (Figure 14 and Table 5). When the working face advances 210 m, the main roof moves periodically and the support working resistance is 4616.81 kN. When the working face advances 230 m, the lower main roof is fractured and the support working resistance is 9767.04 kN. When the working face advances 260 m, bending and fracture occur in the lower main roof, and the deformation of the higher main roof is induced, while the support working resistance is 9307.20 kN. When the working face advances 270 m, the subsidence of the higher main roof fracture leads to the subsidence of the lower main roof and the immediate roof fractures, and obvious dynamic pressure appears on the working face; meanwhile, the support working resistance is 12664.05 kN. When the working face advances 280 m, the higher main roof complete fracture, and the support working resistance is 4754.77 kN. When the working face advances 280~350 m, a new round of periodic motion occurs on the main roof, and a lower main roof fracture occurs at 310 m and a higher main roof fracture subsidence occurs at 350 m, while the support working resistances are 9422.16 kN and 12549.09 kN. This shows that as the working face advancing, roof strata fracture initiation and propagation, the deformation under the pressure and stress state are in the process of continuous adjustment.

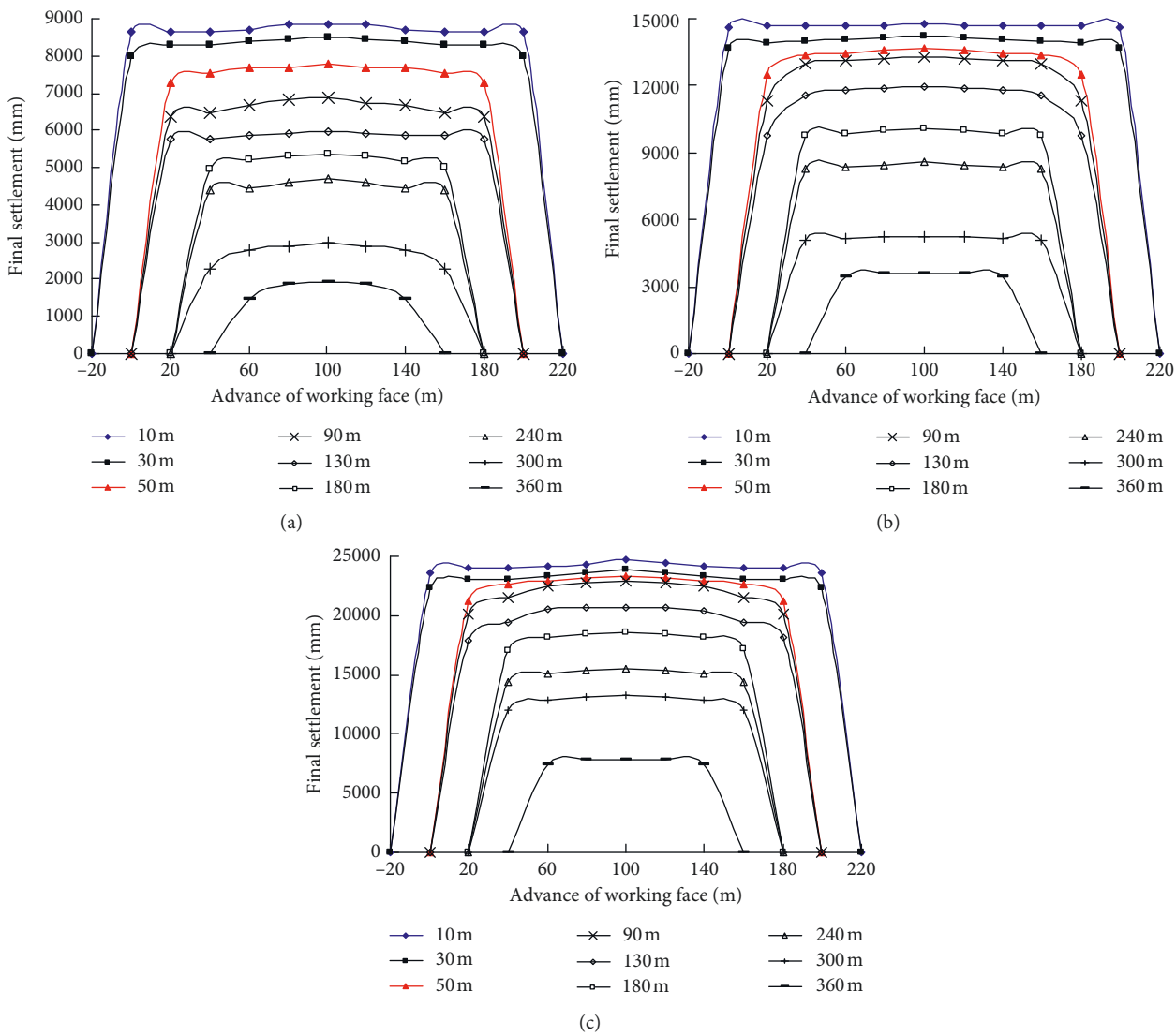


Figure 10: Curves of overlying strata movement with the advancing distance of 200 m in different thick coal seams: (a) 9.0 m; (b) 15.0 m; (c) 25 m.

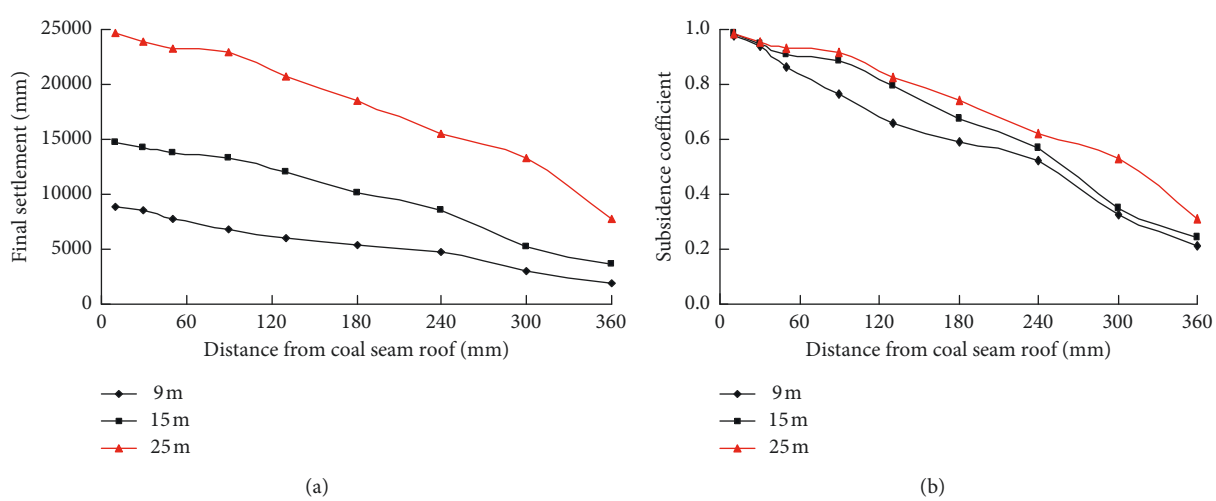


Figure 11: The subsidence variation law curves of overlying strata in different thick coal seams. (a) The final settlement curves of overlying strata with different mining thicknesses. (b) The subsidence coefficient curve of overlying strata with different mining thickness.

Table 4: Overlying strata convergence in different thick coal seams.

Distance from coal seam roof (m)	9 m		15 m		25 m	
	Final settlement (mm)	Subsidence coefficient	Final settlement (mm)	Subsidence coefficient	Final settlement (mm)	Subsidence coefficient
10	8847	0.983	14790	0.986	24675	0.987
30	8478	0.942	14235	0.949	23950	0.958
50	7767	0.863	13695	0.913	23325	0.933
90	6876	0.764	13290	0.886	22900	0.916
130	5952	0.661	11970	0.798	20700	0.828
180	5337	0.593	10095	0.673	18575	0.743
240	4689	0.521	8565	0.571	15525	0.621
300	2961	0.329	5235	0.349	13225	0.529
360	1908	0.212	3630	0.242	7800	0.312

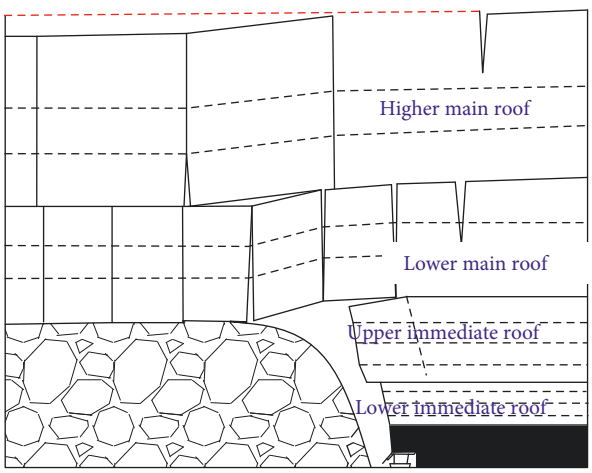


Figure 12: Overlying strata structure model.

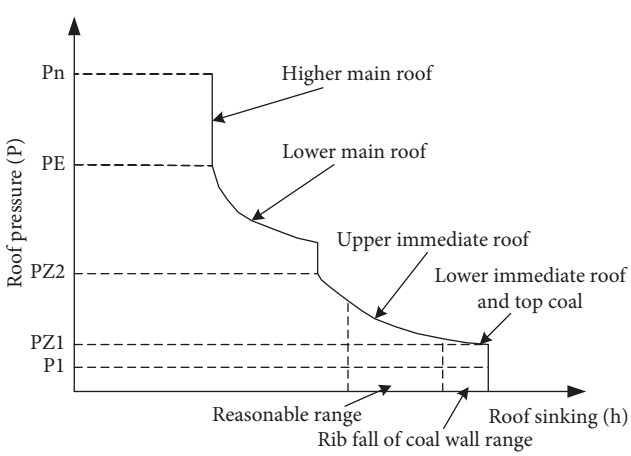


Figure 13: Relationship between support and sinking in extrathick coal seam fully mechanized top-coal caving mining.

It should be noted that the model support in the test does not have a safety valve, and the safety valve cannot be opened automatically when the pressure of the roof reaches the rated working resistance in actual production. However, the resistance of the hydraulic support measured in the test can

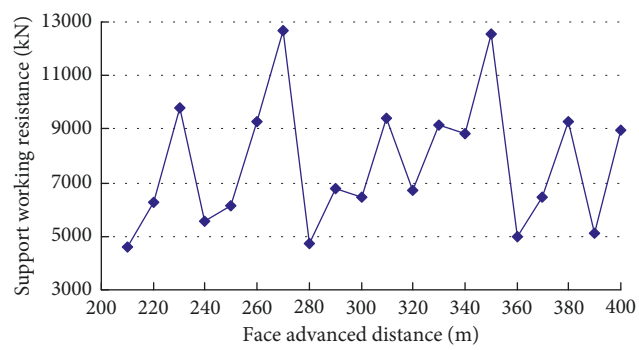


Figure 14: Variation in support working resistance under different mining distances of the working face in the 9 m coal seam.

more truly reflect the pressure of the roof under actual conditions.

The comparison results of rock pillar height and support working resistance under different conditions of main roof movement in different thick coal seams are shown in Tables 6 and 7. Conclusions can be drawn as follows:

(1) There are three working states of loading support in extremely thick coal seams, such as normal circumstance, lower main roof pressure, and higher main roof pressure; meanwhile, these states keep changing.

(2) Rock pillar height to be controlled and support working resistance increase with the increase of mining thickness. Taking the higher main roof pressure as an example, when the mining thickness is 9 m, 15 m, and 25 m respectively, the height of the rock pillar height to be controlled is 50.22 m~50.68 m, 68.34~71.75 m, and 82.51~93.64 m, and the average of support working resistance required is 12606.57 kN, 17503.89 kN, and 22010.9634 kN, respectively. In order to ensure the safety of the working face, the working resistance of the support should be appropriately increased.

(3) Under normal circumstances, because the roof pressure is small, the working resistance of the required support is small, and taking measures to

Table 5: Support working resistances obtained from the material simulation test in coal with a thickness of 9 m.

Face advanced distance (m)	Strain	Support load (MPa)	Model support resistance (kN)	Model height of rock pillar should be controlled, h (cm)	Actual height of rock pillar should be controlled, H (m)	Actual support working resistance (kN)
210	−219	0.16	0.79	9.24	18.47	4616.81
220	−146	0.21	1.07	12.60	25.19	6295.24
230	5	0.33	1.66	19.54	39.08	9767.04
240	−178	0.19	0.95	11.12	22.25	5559.49
250	−152	0.21	1.05	12.32	24.64	6157.28
260	−15	0.32	1.58	18.62	37.24	9307.20
270	131	0.43	2.15	25.34	50.68	12664.05
280	−213	0.16	0.81	9.51	19.03	4754.77
290	−125	0.23	1.15	13.56	27.12	6778.07
300	−138	0.22	1.10	12.96	25.93	6479.17
310	−10	0.32	1.60	18.85	37.70	9422.16
320	−127	0.23	1.14	13.47	26.94	6732.09
330	−21	0.31	1.56	18.35	36.69	9169.25
340	−35	0.30	1.50	17.70	35.40	8847.36
350	126	0.43	2.13	25.11	50.22	12549.09
360	−204	0.17	0.84	9.93	19.85	4961.70
370	−138	0.22	1.10	12.96	25.93	6479.17
380	−15	0.32	1.58	18.62	37.24	9307.20
390	−197	0.17	0.87	10.25	20.50	5122.64
400	−31	0.30	1.52	17.89	35.77	8939.33

Table 6: Rock pillar height which needs to be controlled under different conditions of main roof movement in different thick coal seams.

State of the roof	Actual height of rock pillar should be controlled (m)		
	9 m	15 m	25 m
Normal circumstance	18.47~27.12	24.27~39.73	31.63~50.22
Lower main roof pressure	35.40~39.08	51.23~59.69	52.24~68.89
Higher main roof pressure	50.22~50.68	68.34~71.75	82.51~93.64

Table 7: Support working resistance under different conditions of basic roof movement in different thick coal seams.

State of the roof	Support working resistance (kN)			Average of support working resistance (kN)		
	9 m	15 m	25 m	9 m	15 m	25 m
Normal circumstance	4616.8~6778.1	6065.3~9928.0	7904.7~12549.1	5812.4	8451.6	9885.8
Lower main roof pressure	8847.4~9767.0	12802.0~14917.3	13054.9~17216.5	9251.4	13905.6	15043.7
Higher main roof pressure	12549.1~12664.1	17078.5~17929.2	20619.3~23401.4	12606.6	17503. 9	22010.3

reduce the working resistance of the support properly under the condition of ensuring the safety of the mining field will be conducive to the top-coal production and release of ultrathick coal seams.

(4) During the lower main roof pressure, the working resistance of the support may reach or approach the rated working resistance of the support. At this time, properly increasing the working resistance of the support can balance the impact load when the lower main roof pressure is applied so as to ensure the effective protection of the support to the main roof.

(5) During the higher main roof pressure, underground pressure will be greater than the rated working resistance, while the present condition of hydraulic support technology cannot meet the requirements. It is difficult to control the roof strata effectively through changing the state of the support working resistance, and some measures are taken to ensure the stability of the stope support, such as releasing high pressure, lowering the cutting height appropriately, and lowering the recovery rate of coal.

4. Discussion and Conclusions

Through theoretical analysis, laboratory similarity simulation test, and other methods, this paper studies the effect of mining thickness on overburden movement and underground pressure characteristics for extremely thick coal seams by sublevel caving with high bottom cutting height. Some conclusions can be drawn as follows:

(1) The height of free space in goaf increases exponentially with the increase of mining thickness, and thickness the immediate roof needs to be controlled increase linearly with the increase of mining thickness. The main roof of ordinary mining height and fully mechanized caving face may be transformed into the immediate roof that needs to be controlled in the ultrathick coal seams. As the height of the corresponding fault zone increases, the immediate roof area that needs to be controlled increases, the main roof strata of "semibearing" is transferred to higher strata, and the main roof area is also increased.

(2) With the increase of mining thickness, the height of overburden movement damage increases, while the flattening rate of the collapse arch tends to decrease with the increase of mining thickness. With the increase of mining thickness, the thickness of roof entering the caving zone increases. The roof strata in the collapse zone and the height of the fault zone are mainly composed of key strata, which have obvious composite deformation, subsidence, and failure. The migration and collapse of top coal and roof is a dynamic process, in which the thickness of the rock pillar should be controlled in the immediate roof is changing, while the thickness and configuration of the lower main roof are also dynamically changing.

(3) A new concept has been proposed, rock pillar height which needs to be controlled can be used for reference to study the working resistance of support.

(4) There are three working states of loading support in extremely thick coal seams, such as normal circumstance, lower main roof pressure, and higher main roof pressure; meanwhile, these states keep changing. Rock pillar height to be controlled and support working resistance increase with the increase of mining thickness.

(5) Taking measures to reduce the working resistance of the support properly will be conducive to the top-coal production and release of ultrathick coal seams under normal circumstances. Properly increasing the working resistance of the support can balance the impact load when the lower main roof pressure is applied. Some measures are taken to ensure the stability of the stope support during the higher main roof pressure, such as releasing high pressure, lowering the cutting height appropriately, and lowering the recovery rate of coal.

Data Availability

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

Conflicts of Interest

The authors declare that there are no conflicts of interest regarding the publication of this paper.

Acknowledgments

This work was supported by the National Natural Science Foundation of China (Grant no. 51404167), State Key Laboratory of Coal Resources and Safe Mining (Grant no. SKLCRSM14KFB08), State Key Laboratory for Geo-Mechanics and Deep Underground Engineering (Grant no. SKLGDUEK1410), Key Laboratory of Safety and High-efficiency Coal Mining Ministry of Education (Grant no. JYBSYS2015105), China Postdoctoral Science Foundation Funding Project (Grant no. 2016M590151), and Scientific and Technological Innovation Programs of Higher Education Institutions in Shanxi (Grant No. STIP 201802047).

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