This paper presents a simulation study on parameter optimization of anchor net and cables of the B coal seam development roadway in Qinfa No.1 using numerical simulation methods. Mine and examines the roadway's stability (stress, deformation, and plastic zone) in the presence of fluctuating coal pillar width and mining dynamic stress. The findings indicate that the main roadway's vertical stress is significantly higher when the coal pillar's width is less than 30m and the mining stress coefficient is ≥3.0. The two gangways and the main roadway roof displacements gradually decrease with increasing coal pillar width. The larger the mining stress coefficient, the greater the reduction. It is thought that when the coal pillar width is not less than 40m, the interaction between the development of the main roadway will be less. The two gangways of the roadway and the roof's displacement under each coal pillar width change slightly when the mining stress coefficient is ≤2.5. After the coal pillar width is 40 m, there is minimal change in the displacement of the roof and the two gang ways of the roadway when the mining stress coefficient is >2.5. In other words, there is little mutual influence when the coal pillars in an open roadway are 40m wide. The plastic zone of the surrounding rock mass is within the anchor bolts and cables support range when the mining stress coefficient is ≤1.5. The plastic zone (deformation ≥120mm, plastic zone distribution) surpasses the support area of anchor rods and cables when the mining stress coefficient is ≥2.5.

**Keywords:** Roadway's Stability, Mining Stress, Gangways, Stress Coefficient, Simulation Research, Optimization Of Anchor Net And Cables.

**I. INTRODUCTION**

In recent years, with the development of coal mine roadway support theory and technology, the proportion of roadway driving along the coal seam is increasing, and the support form of coal seam roadway has changed greatly [1]. The stability control of soft rock roadways has always been a technical problem in coal mine production and construction, especially with floor heave in the nonlinear large deformation control of soft rock roadways [2-4].

Currently, domestic and foreign researchers have focused on the issue of roadways stability. Guo Dongjie et al. proposed a method of controlling the stability of tunnel surrounding rock by secondary reinforcing support after modifying the broken surrounding rock with grouting [5]. Wang et al. combined the advantages of bolts, anchor cables, and grouting, proposing the integrated support technology of bolt and grouting [6]. Jia et al. considered that high ground stress is an important factor leading to deformation of deep surrounding rock, analysed the failure mechanism of soft rock, and proposed a support method suitable for soft rock [7]. In addition, Kang Tianhe et al. advocate the combined support with bolt (anchor cable) and wire netting [8]. Yang Feng, Wang Lianguo et al. suggest improving the pre-tightening force of bolt (anchor cable) and strengthening support [9].

Xie Xiaoping proposed the asymmetric "anchor net, cable and spray" secondary joint support plan [10]. Shi Chaohong et al. analyzed the stress state around the isolated coal pillars of different-sized roadways [11]. Xiao Jiang et al. studied the overlying rock collapse characteristics under the influence of mining at different working faces in shallow coal seams [12]. Liu Huaidong et al. studied the stress field distribution of the surrounding rock in the floor tunnel and the damage characteristics of the coal seam floor rock layer during the cross-mining process of the working face [13]. Rong et al. proposed measures for controlling the stability of the surrounding rock in severely inclined and ultra-thick coal seams. They suggested that full-length anchoring should be implemented in the support of inclined coal seam tunnels, increasing the anchoring length of the anchor cables to enhance the integrity of the roof [14]. Zhang Xuanlei and He Weijie optimized the layout process of the two tunnels based on the layout of the two tunnels in the fully mechanized mining face [15]. Zhang Shiping analyzed...
the impact of factors such as different mining depths, coal thickness, and roadway spacing on the impact of isolated coal masses in roadways [16]. Pan Junfeng et al. revealed the mechanism of coal seam tunnel group rockburst based on time-varying characteristics, and proposed new prevention and control methods [17].

Although many successes have been achieved, the roadway stability under coal seams mining still needs further study. Due to the influence of intensive dynamic pressure, some common support techniques such as U-shaped steel, bolts, and cables may not apply or need to be improved. In conclusion, it is evident that the optimization of anchor net and cable support for Coal Seam Roadway have been the subject of relatively few studies. This study employs numerical simulation software to investigate the optimal of anchor net and cable support in the B coal seam at Qinfa No. 1 Mine. The main mine roadway’s stability is significantly affected by the research findings.

II. ENGINEERING OVERVIEW

Qinfa No. 1 Mine is located in the northwest of the SDE mining area. The SDE mining area is located in Kota Bharu County in the east of South Kalimantan Province, Indonesia.

As shown in Table 1, According to the drilling data in the mining area: the average thickness of the coal-bearing strata is 290.40m, and there are 5 coal-bearing strata (A, B, C, D, E), the average total thickness is 7.50m, and the coal content coefficient is 2.58%. The average thickness of B coal seam is 4.22m, and the coal content coefficient is 1.13%.

The geological structure of Qinfa No. 1 Mine is simple. It mainly mines the B coal seam. The burial depth is between 180 and 500m. The burial depth of the coal seam gradually increases from west to east. The coal seam thickness is 1.18~8.10m and the average thickness is 4.22m. The entire area can be mined, which is a nearly horizontal coal seam. The coal seam has the characteristics of shallow burial, shallow metamorphism, and easy weathering of the coal rock layer.

Table 1: Summary of characteristics of mineable coal seams in Qinfa No. 1 Mine

<table>
<thead>
<tr>
<th>Seam No.</th>
<th>Coal thickness range (average) (m)</th>
<th>Quantity of gangue (m)</th>
<th>Average thickness of gangue (m)</th>
<th>average layer spacing (m)</th>
<th>Roof/Floor Lithology</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>0.18-0.35 (0.26)</td>
<td>0</td>
<td>-</td>
<td>-</td>
<td>mudstone/siltstone</td>
</tr>
<tr>
<td>B</td>
<td>1.18-8.10 (4.22)</td>
<td>0-4</td>
<td>0.34</td>
<td>24</td>
<td>mudstone/siltstone</td>
</tr>
<tr>
<td>C</td>
<td>0.30-1.34 (0.97)</td>
<td>1-2</td>
<td>0.19</td>
<td>5</td>
<td>mudstone/siltstone</td>
</tr>
<tr>
<td>D</td>
<td>0.58-2.60 (1.41)</td>
<td>0</td>
<td>0</td>
<td>10</td>
<td>mudstone/siltstone</td>
</tr>
<tr>
<td>E</td>
<td>0.45-3.51 (1.50)</td>
<td>0</td>
<td>0</td>
<td>27</td>
<td>mudstone/siltstone</td>
</tr>
</tbody>
</table>

The lithology of the roof and floor of the B coal seam is interbedded with mudstone and siltstone, and mudstone accounts for a large proportion as shown in Table 1. The roof is 2.60~47.70m thick, with an average thickness of 25.15m, and the bottom plate is 8.30~58.90m thick, with an average thickness of 31.52m. The ground elevation is +20 to +160m, and the maximum relative elevation difference is 140m. The B coal seam and its roof and floor aquifers are buried shallowly, and the floor elevation is -105 to -420m. The mine is designed to exploit the B coal seam, and the mining level of the mine is determined to be -110m according to the elevation of the parking lot at the bottom of the mine. The B coal seam is 180~500m deep and the mining tunnel elevation is -110m, so the surrounding rock stress is about 6.55MPa. The horizontal stress is equivalent to 1.4 times the vertical stress, so the horizontal stress is 9.17MPa.
The main roadway is arranged along the roof of the B coal seam, with a rectangular section, a tunnel excavation width of 5.8m, an excavation height of 3.8m, a net width of 5.5m, and a net height of 3.5m. It adopts the active support form of anchor nets and cables, and the support parameters are: between anchors The row spacing is 900×1000mm, and the row spacing between anchor cables is 1800×1800mm. The cross-sectional support form is shown in Figure 1.

III. MODEL ESTABLISHMENT AND BOUNDARY CONDITIONS

We used AUTOCAD software to draw two-dimensional graphics and import it into ANSYS software to build a three-dimensional model. The model size is X×Y×Z=200m×100m×10m. The model is divided into 47844 nodes and 158921 units, as shown in Figure 2. A total of three main lanes is arranged in the three-dimensional model, with coal pillar widths of 20m, 30m, 40m, 50m and 60m respectively.

Stress boundary conditions are applied to the left, right, front, and top of the model. The bottom of the model is fixed and constrained. The upper boundary is free and bears the uniform load Q=γH exerted by the overlying rock on the boundary.
The distance between the main lanes is (a) 20m, (b) 30m, (c) 40m, (d) 50m and (e) 60m, respectively.

**Figure 2**: Schematic diagram of the model.

The model adopts the Mohr-Coulomb strength criterion, and the mechanical parameters of the coal and rock mass are shown in Table 2.

**Table 2**: Mechanical parameters of rock mass

<table>
<thead>
<tr>
<th>Lithology</th>
<th>Density</th>
<th>Tensile Strength</th>
<th>Cohesion</th>
<th>Angle of Internal Friction</th>
<th>Bulk Modulus</th>
<th>Shear Modulus</th>
</tr>
</thead>
<tbody>
<tr>
<td>Silt sand-sandy mudstone-claystone</td>
<td>2450</td>
<td>1.30</td>
<td>1.76</td>
<td>26.80</td>
<td>2.33</td>
<td>1.08</td>
</tr>
<tr>
<td>Silt-sandy mudstone</td>
<td>2500</td>
<td>1.50</td>
<td>1.48</td>
<td>25.00</td>
<td>1.82</td>
<td>0.94</td>
</tr>
<tr>
<td>A seam</td>
<td>1380</td>
<td>0.58</td>
<td>0.81</td>
<td>19.36</td>
<td>0.80</td>
<td>0.48</td>
</tr>
<tr>
<td>Mudstone-siltstone interbedded</td>
<td>2108</td>
<td>0.95</td>
<td>1.00</td>
<td>28.00</td>
<td>1.25</td>
<td>0.71</td>
</tr>
<tr>
<td>B seam</td>
<td>1380</td>
<td>0.90</td>
<td>1.10</td>
<td>22.00</td>
<td>0.96</td>
<td>0.60</td>
</tr>
<tr>
<td>mudstone</td>
<td>1800</td>
<td>0.80</td>
<td>1.50</td>
<td>29.31</td>
<td>1.74</td>
<td>0.99</td>
</tr>
<tr>
<td>C seam</td>
<td>1380</td>
<td>0.90</td>
<td>1.10</td>
<td>22.00</td>
<td>0.96</td>
<td>0.60</td>
</tr>
<tr>
<td>sandstone mudstone interbedded</td>
<td>2200</td>
<td>1.65</td>
<td>1.58</td>
<td>22.00</td>
<td>1.59</td>
<td>0.87</td>
</tr>
<tr>
<td>D seam</td>
<td>1380</td>
<td>0.90</td>
<td>1.10</td>
<td>22.00</td>
<td>0.96</td>
<td>0.60</td>
</tr>
<tr>
<td>sandstone mudstone interbedded</td>
<td>2200</td>
<td>1.65</td>
<td>1.58</td>
<td>22.00</td>
<td>1.59</td>
<td>0.87</td>
</tr>
<tr>
<td>clay rock</td>
<td>2100</td>
<td>1.45</td>
<td>1.11</td>
<td>24.00</td>
<td>1.74</td>
<td>0.99</td>
</tr>
<tr>
<td>E seam</td>
<td>1380</td>
<td>0.90</td>
<td>1.10</td>
<td>22.00</td>
<td>0.96</td>
<td>0.60</td>
</tr>
<tr>
<td>Claystone-sandstone interbedded</td>
<td>2250</td>
<td>1.68</td>
<td>1.80</td>
<td>26.00</td>
<td>1.79</td>
<td>1.13</td>
</tr>
</tbody>
</table>

In this numerical simulation, a total of three lanes were excavated, namely Lane 1, Lane 2 and Lane 3. The support simulation of anchor rods and anchor cables was carried out for three main roadways.

### IV. NUMERICAL SIMULATION AND MONITORING DESIGN

#### 1. Numerical simulation scheme design

The support method of anchor nets and cables is as follows: the spacing between anchor rods is 900×1000mm; the spacing between anchor cables is 1800×1800mm.

(1) The influence of coal pillar width on the stability of the main roadway. The coal pillar widths of the three main roadways are 20m, 30m, 40m, 50m and 60m respectively. Combined with the selected support parameters, the influence of different coal pillar widths on the stability of the roadway is simulated.

(2) The impact of mining influence on the stability of the main roadway. Combined with the selected support parameters, the impact of mining influence on the stability of the roadway under different mining influence conditions is simulated.

**Table 3**: Stress magnitude

<table>
<thead>
<tr>
<th>Horizontal stress/MPa</th>
<th>Vertical stress/MPa</th>
<th>Mining stress coefficient K</th>
</tr>
</thead>
<tbody>
<tr>
<td>9.17</td>
<td>6.55</td>
<td>1</td>
</tr>
</tbody>
</table>
2. Monitoring line design

We imported the calculation results into Tecplot result processing and analysis, and arrange the measurement lines in the horizontal direction and the forward direction. The specific arrangement is shown in Figure 3. Since survey line 1 and survey line 3 are symmetrically distributed, the vertical displacement, horizontal stress change pattern and size of each point on survey line 1 and survey line 3 are nearly equal. Therefore, this article only analyses the vertical displacement and horizontal stress of each point on survey line 1 and survey line 2, and analyses the horizontal displacement and vertical stress of each point on survey line 4.

![Figure 3: Measurement line layout plan.](image)

V. RESULT ANALYSIS

1. Main roadway's stability analysis when the coal pillar width is 20m.

a) Vertical stress

Figure 4 shows the vertical stress distribution cloud diagram of each roadway under different mining stress coefficients. It can be seen from Figure 4 that after the excavation of the roadway.

![Images showing vertical stress distribution](images)
Figure 4: Cloud map of vertical stress change of roadway surrounding rock under different mining stress coefficients. (a) Mining stress coefficient 1.0 (b) Mining stress coefficient 1.5; (c) Mining stress coefficient 2.0 (d) Mining stress coefficient 2.5; (e) Mining stress factor 3.0 (f) Mining stress factor 3.5.

Figure 5 illustrates how, after the main roadway has been excavated, the stress of the surrounding rock mass is released and transferred to depth, and the vertical stress appears to be concentrated in the horizontal direction, leading to a superposition of stresses between the main roadways. The vertical stress concentration value between the open roadways decreases with decreasing mining stress coefficient. The vertical stress concentration value between the open roadways progressively rises as the mining stress coefficient does. The coal pillar's width and maximum vertical stress value are extracted. In Figure 5, the change curve is displayed (b). As can be observed, the vertical stress between the main roadways under varying coal pillar widths is similar and all smaller than the stress when the coal pillar width is 20m when the mining stress coefficient is ≤ 2.5. The vertical stress of the roadway is significantly higher when the coal pillar width is less than 30m and the mining stress coefficient is ≥ 3. That is, in order to open up the roads and prevent them from affecting one another, the coal pillar's width should be at least 40m.

Figure 5: Variation curve of vertical stress with coal pillar width under different mining coefficients

b) Vertical displacement

The vertical displacement changes curves for each point on measuring line 2 under various mining stress coefficients are displayed in Figure 6 (the only coal pillar widths provided are 20 and 40 meters).

Figure 6: Relationship between the vertical displacement of measuring line 2 and the change of mining stress coefficient
As Figure 6 illustrates how the bottom plate bulges and the roof subsides following the main road's excavation. The amount that the main roadway's floor bulges and the amount that the roof subsidence gradually increase with the mining stress coefficient. The bottom plate displacement is 18.07mm and the roof plate displacement is 49.92mm when the coal pillar width is 20m and the mining stress coefficient is 1.0. In comparison to the displacement of the roof and floor at 1.0, at 1.5, the displacements increase by 27.12% and 40.62%, respectively. At 2.0, the displacements of the roof and bottom plates increase by 75.92% and 101.16%, respectively. At 2.5 mining stress coefficient, the displacements of the roof and bottom plates increase by 1.59 times and 1.73 times, respectively. The roof and bottom plate displacements increase by 3.04 and 2.79 times, respectively, when the mining stress coefficient is 3.0. When the mining stress coefficient is 3.5, the floor and roof displacements increase by 5.23 and 4.02 times, respectively; in contrast to the case where the coal pillar width is 20m, the floor and roof displacements increase by -2.26 and 2.32%, respectively, when the mining stress coefficient is 1.5. The bottom plates displacements increase by -4.71 and 2.52%, respectively, when the mining stress coefficient is 1.5. The displacements of the bottom plate and roof plate increase by 11.88% and 7.83%, respectively, when the mining stress coefficient is 2.0. The displacements of the roof and bottom plates increase by -10.94% and 2.77%, respectively, when the mining stress coefficient is 2.5. The roof and bottom plate displacements decrease by 16.59% and 0.16%, respectively, when the mining stress of the coefficient is 3.0. Based on the above-mentioned data, variation curves of the main roadway roof and floor displacement with the width of the coal pillar under various mining coefficients are drawn, as illustrated in Figure 7. When the mining stress coefficient is 3.5, the displacements roof and bottom plates are reduced by 22.34% and 1.13%, respectively.

(c) Horizontal displacement

The variation curve of the left and right sides of the main road displacement with the coal pillar width under various mining coefficients is displayed in Figure 8. The main alley’s left and right sides were unequally displaced, with varying degrees of displacement experienced after the excavation. The main roadway’s left and right displacements decrease with increasing coal pillar width under the same mining stress coefficient; the
reduction increases with increasing mining stress coefficient. The two sides of the main road shift slightly beneath each coal pillar width when the mining stress coefficient is ≤ 2.5. When the mining stress coefficient is > 2.5, there is little displacement between the two sides of the main roadway after the coal pillar width reaches 40m, which means that the mutual influence on the main roadway's development is minimal.

Displacements changing with coal pillar width under different mining stress coefficients

![Displacements changing with coal pillar width](image)

Figure 8: Variation curve of left and right-side displacement with coal pillar width under different mining stress coefficients

VI. PLASTIC ZONE

Figure 9 displays the distribution cloud diagram of the plastic zone of the surrounding rock surrounding the development roadway under various mining stress coefficients following anchor bolt and cable support (when the coal pillar width is 40 m). It is evident that a specific plastic zone remains in the surrounding rock mass surrounding the roadway even after it has been excavated and supported; this plastic zone primarily manifests as tensile failure and shear failure. At this point, however, the deformation is minimal because all of the surrounding rock mass and plastic zone areas are supported by anchor bolts and cables when the mining stress coefficient is ≤ 1.5. The plastic zone area of the surrounding rock mass around each roadway gradually increases as the mining stress coefficient increases, and it eventually surpasses the anchor bolt and cable support area. The plastic zone surrounding the roadway is larger (dispersed in a "butterfly" shape) and the deformation of the roof and bottom plates exceeds 150 mm, while the deformation of the two sides reaches 120 mm when the mining stress coefficient is ≥ 2.5. Several plastic zones surround the road when the mining stress coefficient approaches 3.5. Stress concentration happens as a result of the interactions between the three roads and the overlapping plastic zones surrounding them. At this point, the main roadway's floor and roof have deformed more than 300 mm, and the two gangs' deformation when the coal pillar width is 60 m has exceeded 250 mm. The deformation of the floor, roof, and two gangs exceeds 400 mm and 340 mm, respectively, when the coal pillar width is 20 m. More reinforcing measures are required because the main roadway's deformation is significant and has negatively impacted the stability of the main roadway's floor, roof, and two sides.
Figure 9: Cloud diagram of the plastic zone distribution of the surrounding rock of the roadway when the coal pillar width is 40 m.

VII. CONCLUSION

This article uses the B coal seam development roadway of Qinfa No. 1 Mine as the research background, uses FLAC 3D numerical software to carry out a simulation study on the coal pillar width optimization of the all-coal development roadway, the conclusions are as follows:

1. When the width of the coal pillar is less than 30 m and the mining stress coefficient is ≥ 3.0, the vertical stress of the main roadway is much higher than when the width of the coal pillar is 40 m. It is considered that when the width of the coal pillar is not less than 40 m, the mutual influence of the development of the main roadway will be less.

2. Under the same mining stress coefficient, the displacements of the main roadway roof and the two gangways gradually decrease as the width of the coal pillar increases, and the larger the mining stress coefficient, the greater the reduction.

3. When the mining stress coefficient is ≤ 2.5, the displacement of the roof and the two gangways gradually decrease under each coal pillar width. When the mining stress coefficient is > 2.5, the displacement of the roof and the two gangways of the roadway changes after the coal pillar width is 40 m. Smaller, that is, the mutual influence is smaller when the coal pillar width between the open roadways is 40 m.

4. When the mining stress coefficient is ≤ 1.5, the plastic zone of the surrounding rock mass is within the support range of anchor bolts and cables. When the mining stress coefficient is ≥ 2.5, the plastic zone exceeds the support area of anchor rods and cables (deformation ≥ 120 mm, the plastic zone distribution pattern is "butterfly-shaped").

VIII. REFERENCES


