A New Technique of Grouting to Prevent Water–Sand Mixture Inrush inside the Mine Panel—A Case Study

Rongjie Hu 1, Wanghua Sui 1,2,*, Daxing Chen 1, Yuxuan Liang 1,2, Ruijian Li 2,3, Xinhuai Li 2,4 and Ge Chen 2,4

1 Wanbei Coal-Electricity Group Co., Ltd., Suzhou 234000, China; tb23010012a41ld@cumt.edu.cn (Y.L.)
2 Institute of Mine Water Hazard Prevention and Control Technology, School of Resources and Geosciences, China University of Mining and Technology, Xuzhou 221116, China; 15697040086@163.com (X.L.); cg58@cumt.edu.cn (G.C.)
3 The Second Exploration Team of Jiangsu Coal Geology Bureau, Xuzhou 221000, China
4 Sinopeo Zhongyuan Oilfield Exploration and Development Research Institute, Puyang 457100, China
* Correspondence: suiwanghua@cumt.edu.cn

Abstract: Water–sand mixture inrush generally poses a significant threat to the safe operation of the quarry of coal mines. Therefore, proactive management initiatives are essential to enhance the impact resulting from mining operations. A novel approach involving grouting into the unconsolidated sand aquifer and the weathered zone was initially executed in the 1010-1 panel of the Wugou coal mine in Anhui Province, China. Considering the hydrogeological conditions of the study area, over 70 thousand tons of cement and fly ash were injected through 42 boreholes. Sampling, laboratory tests, similar materials model simulations, and numerical simulations of the trending and dipping profiles were all employed to elucidate the evolution and characteristics during the progression of the No. 10 coal seam. The outcomes illustrated that the grouting execution had transformed the structure of the porous media, weakened the watery media, and intensified the mechanical strength of the No. 4 aquifer and the weathering zone. This transformation proved beneficial in reducing the heights of the caving zone and water-conductive fracture zone, leaving more coal–rock pillars for safety. Twenty-seven underground detection drill holes and whole-space 3D resistivity exploration were adopted to verify its transformed property of low water content. During the mining process, the height of the caving zone at 19.70 m was measured through inter-hole parallel electrical detection. The pressure of hydraulic supports in the grouted area did not exceed the rated working pressure during mining. All of these findings highlight the significant impact of grouting in this study area. The successive safe mining of the 1010-1 panel demonstrates that grouting can be used to prevent water–sand mixture inrush during mining operations.

Keywords: water–sand mixture inrush; unconsolidated sand aquifer; weathered zone; grouting; mining activity

1. Introduction

Hydrogeological and engineering geological difficulties always exist and have an importance influence on the safety of coal mine construction and operations. Water–sand mixture inrush is a special geologic disaster that occurs when the mining quarry approaches an unconsolidated sand aquifer and the weathered zone in the coal mine. When the mining activities disturb the original geological framework, a mixture of water and sand can rush into the underground space, resulting in property loss and human casualties. Generally, four conditions (water–sand source, channels induced by mining activities [1], quarry space, and potential energy accumulation resulting from groundwater head) are the main factors contributing to water–sand mixture inrush [2]. In particular, in the context of large-scale and high-intensity coal mining near loose layers or weathering zones, the extensive roof cutting poses significant safety challenges. Notably, the risk of water–sand mixture inrush becomes a critical concern and jeopardizes the safety of mine production [3]. Therefore, it
is essential to investigate the mechanisms behind mining-induced water–sand inrush from loose water-bearing sand layers and weathered zones. Additionally, assessing the risk of these hazards and developing comprehensive disaster prevention and control management guidelines for water–sand mixture inrush are imperative.

The research on water–sand inrush in recent years has focused on various aspects, including prediction models, risk assessment, prevention and control strategies, and technological method innovations. Several studies have explored the use of advanced technologies such as artificial intelligence [4], remote sensing [5,6], and numerical modeling [7,8] to enhance prediction accuracy and develop early warning systems.

Currently, there is a limited body of research on water–sand mixture inrush occurrences. Nevertheless, some investigations have delved into innovative technologies such as drone-based monitoring [9], underground imaging [10], and 3D geological modeling to enhance the forecasting and control of water–sand mixture inrush incidents. An experimental inquiry was carried out to examine the sand generation mechanism of unconsolidated sand, taking into account variables like water content and injection pressure [11]. Based on recession analysis and the hydrodynamics of open channels [12], the discharge time after water inrush into the tunnel was assessed. Presently, the research in this area is somewhat fragmented, lacking a cohesive system for the prediction and prevention of water–sand mixture inrush.

Significant emphasis is placed on risk assessment methodologies to evaluate the vulnerability of coal mine formations to these hazards, considering factors such as geological conditions, mining activities, and hydrogeological characteristics [13–16]. Based on these risk assessments of water–sand mixture inrush, Chinese scholars have also studied diverse prevention and control strategies. These strategies encompass enhanced engineering practices, ground consolidation techniques, and the implementation of early warning systems to mitigate the consequences of water–sand mixture inrush.

During coal mining operations, in order to ensure the safety of mine production and minimize water–sand mixture inrush accidents, various passive and active prevention and control methods are often adopted. Passive methods leverage existing hydrogeological structures, while active engineering measurements focus on proactive prevention and control [17]. Among these, grouting transformation stands out as a typical active method. This technique involves transforming or blocking specific aquifer or potential water-guiding boundaries or channels to enhance the water-barrier strength, thereby meeting the requirements of safe mining operations. In China, grout-blocking technology has been extensively applied in coal mines to manage and regulate the aquifer water content, thereby reducing the risk of water inrush disasters during mining operations [18,19]. Despite grouting technology becoming increasingly mature, there are relatively few specialized evaluations of the risk of water–sand mixture inrush after aquifers are grouted. This paper primarily focuses on the risk assessment of water–sand mixture inrush for the No. 4 aquifer in the 1010-1 panel after grouting.

2. Hydro-Geological Condition of the Study Area
2.1. Description of the Study Area

The study area, known as the Wugou coal mine, covers an area of 21.65 km² and is located in the Linhuan coal mining region in the northern part of Anhui Province, China, as illustrated in Figure 1a,b. There are a total of 8 coal seams that can be mined in this area. However, currently only No. 10 coal seam is being mined with an annual production reaching 1.26 million tons [20].
The main target area is situated in the 1010-1 panel, which is at the western part of the No. 1 coal mining region within the Wugou mine, as shown in Figure 1c. The lengths of belt roadway and track roadway in the 1010-1 panel are, respectively, 662.1 m and 691.1 m. The opening-off cut length is 164.7 m. The roof of the No. 10 coal seam ranges from −346 m to −262 m in this panel, with an average thickness of 4.2 m. The dip ranges from 3° to 15°. The synthesized calculation of the geological resources is 604,582 tonnes and the available mining resources is estimated at 562,2618 tonnes.

The No. 10 coal seam within Shanxi Group was developed with one or two interlayered gangues, as shown in Figure 1d. The roof of the seam is approximately 60 m away from the overlying aluminous mudstone, while its floor is situated about 50 m from the lower Taiyuan Formation [21].
2.2. Hydro-Geological Framework of 1010-1 Panel

The mine water primarily originates from the sandstone aquifer of the Shanxi Group, the weathered zone, and the No. 4 unconsolidated sand aquifer (No. 4 aquifer) from the Cenozoic era, as shown in Figure 2a. These aquifers with poor yield are dominated by static groundwater reserves. After the coal was mined out, the conductive fracture zone was subsequently developed, which disturbed these aquifers, leading to the formation of water–sand mixture inrushes. An exploration survey with the pumping tests for the No. 4 aquifer demonstrated that its hydraulic conductivity ranges from 0.011299 m/d to 0.435720 m/d, and its specific flow is between 0.0017 L/(s·m) and 0.1352 L/(s·m). These hydrogeological parameters reveal that a high mud content is a property of the No. 4 aquifer [22]. The combination of poor recharge conditions and lower hydraulic conductivity are induced the static storage and poor yield. The local groundwater circulation between the roof sandstone aquifer of the coal seam, weathered zone, and the No. 4 aquifer is controlled by the regional streamline flow. As for this hydrogeological framework and the profiles of panel 1010-1 as shown in Figure 2b, the safe coal–rock pillar is higher when the stopping line is far from the opening-off cut. Therefore, the most threatened space is the opening-off cut, which may be influenced by the water–sand mixture inrush from the No. 4 aquifer.

![Figure 2](image-url) Hydrogeological profiles of (a) belt roadway and (b) track roadway in 1010-1 panel.

2.3. Hydro-Geological and Geological Engineering Conditions

The total area of the water–sand mixture inrush zone was 37,123 m², as depicted in Figure 3. The results from historical exploration boreholes indicated that the burial depth for the bottom boundary of the No. 4 aquifer ranged from 264.41 m to 275.18 m, with an average of 270.33 m. The thickness of the No. 4 aquifer ranged from 25.30 m to 47.88 m, and the average value was 32.80 m. The thickness was greater in the central and southern area compared to other regions, as shown in Figure 3. Similarly, the water yield was gradually higher from the north to the south and from the east to the west. The laboratory analysis of samplings from the No. 4 aquifer also revealed that the particle composition was uneven with a moderate to good grading. Furthermore, the No. 3 aquiclade from the Cenozoic era served as the overlying strata, with a thickness ranging from 36.0 m to 62.35 m, with
an average of 47.84 m. The multi-layered composite framework of these strata consisted of sandy clay, light clay, and sand. The unmanaged geological condition for coal resource operation is prone to inducing the occurrence of water–sand mixture inrush disasters [23].

Figure 3. Contour map depicting the thickness of the No. 4 aquifer in the 1010-1 panel.

3. Methods

3.1. Sampling and Laboratory Analytical Tests

Rock and groundwater samples were meticulously gathered from the Wugou coal mine, accompanied by comprehensive field investigations to ensure the preservation of sample integrity for the subsequent laboratory analyses. Specifically, sand samples were collected from the No. 4 aquifer from the Cenozoic era, and extracted from the designated No. 2 (at depths ranging from 42.6 m to 43 m), No. 4 (at depths ranging from 67.5 m to 69 m), and No. 5 (at depths ranging from 73.5 m to 74 m) boreholes. These samples were carefully preserved in sealed containers to maintain their pristine state. In addition to the solid samples, borehole effluent water samples were also systematically collected. The water quality analysis tests were able to provide the precise identification of the water source.

In order to determine the proportion of each particle group in the No. 4 aquifer in the Cenozoic loose layer at the Wugou coal mine, an analysis of the particle size ratios was conducted. Sand samples were collected and subjected to particle grading analysis to assess their grading characteristics. Furthermore, selected samples from the grouted No. 4 aquifer were magnified and examined under an optical microscope to showed the researchers the micro-particles. Meanwhile, a MATLAB (R2022a) program was developed to process the images using the YIQ color space, facilitating the extraction of grouted regions within the samples through binarization. Subsequently, the proportion of grouted regions within the samples was accurately calculated.

3.2. Material Simulation

Utilizing the data obtained from the engineering geology and hydrogeological exploration holes of the 1010-1 panel, samples were taken for physical and mechanical parameter tests. This process enabled the identification of the mechanical parameters of individual rock layers in the roof and floor strata of the No. 10 coal seam, as shown in Table 1.
Table 1. The geo-technical properties of roof strata in the study area.

<table>
<thead>
<tr>
<th>Lithology</th>
<th>Density (g/cm³)</th>
<th>Compressive Strength (MPa)</th>
<th>Tensile Strength (MPa)</th>
<th>Cohesion (MPa)</th>
<th>Internal Friction Angle (°)</th>
<th>Elastic Modulus (10⁶ MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>No. 3 aquiclude</td>
<td>2.30</td>
<td>8.9</td>
<td>1.5</td>
<td>1.9</td>
<td>32</td>
<td>0.5</td>
</tr>
<tr>
<td>No. 4 aquifer</td>
<td>2.40</td>
<td>2.6</td>
<td>0.8</td>
<td>0.6</td>
<td>27</td>
<td>0.4</td>
</tr>
<tr>
<td>Mudstone</td>
<td>2.33</td>
<td>1.8</td>
<td>0.5</td>
<td>0.6</td>
<td>26</td>
<td>0.5</td>
</tr>
<tr>
<td>Fine sandstone</td>
<td>2.36</td>
<td>8.8</td>
<td>0.8</td>
<td>1.2</td>
<td>26</td>
<td>1.6</td>
</tr>
<tr>
<td>Sandy mudstone</td>
<td>2.42</td>
<td>14.2</td>
<td>1.1</td>
<td>8.9</td>
<td>24</td>
<td>1.55</td>
</tr>
<tr>
<td>Siltstone</td>
<td>2.51</td>
<td>18.64</td>
<td>1.86</td>
<td>2.48</td>
<td>33</td>
<td>2.35</td>
</tr>
<tr>
<td>Mudstone</td>
<td>2.56</td>
<td>17.7</td>
<td>1.13</td>
<td>2.2</td>
<td>32</td>
<td>1.2</td>
</tr>
<tr>
<td>Fine sandstone</td>
<td>2.56</td>
<td>25.27</td>
<td>2.53</td>
<td>4.79</td>
<td>35</td>
<td>3.38</td>
</tr>
<tr>
<td>Mudstone</td>
<td>2.58</td>
<td>10.43</td>
<td>1.39</td>
<td>2.03</td>
<td>29</td>
<td>1.44</td>
</tr>
<tr>
<td>No. 10 coal seam</td>
<td>1.50</td>
<td>7.2</td>
<td>0.3</td>
<td>4</td>
<td>10</td>
<td>0.85</td>
</tr>
<tr>
<td>Mudstone</td>
<td>2.59</td>
<td>15.37</td>
<td>1.35</td>
<td>2.2</td>
<td>25</td>
<td>1.73</td>
</tr>
<tr>
<td>Fine sandstone</td>
<td>2.67</td>
<td>31.93</td>
<td>3.99</td>
<td>4.48</td>
<td>35</td>
<td>3.73</td>
</tr>
<tr>
<td>Siltstone</td>
<td>2.68</td>
<td>20.72</td>
<td>2.32</td>
<td>2.93</td>
<td>32</td>
<td>2.29</td>
</tr>
</tbody>
</table>

A similar model test was carried out utilizing an engineering geo-mechanical model frame, as illustrated in Figure 4. The dimensions of the trending profile were 300 cm × 30 cm × 180 cm; the dimensions of the dipping profile were 200 cm × 30 cm × 180 cm. The geometric similarity ratio of the model was established at 1:100, the time similarity ratio at 1:10, the capacity similarity ratio at 1:1.3, and the strength-to-stress similarity ratio at 1:130. Each stratum was arranged horizontally. The weights of the different materials in the model layering were calculated based on the specified material proportions and similar material ratios. The total amount of material in each layer of the model can be calculated using the following equation:

\[
Q = r \cdot l \cdot b \cdot m \cdot k
\]  

(1)

where \(r\) — capacitive of the material; \(Q\) — weight of the material in each stratification of the model; \(l\) — model length; \(b\) — model width; \(m\) — model stratification thickness; and \(k\) — material loss coefficient.

Figure 4. Engineering geo-mechanics model in the laboratory for the trending and dipping profiles.

The overlying Quaternary strata with a huge thickness need to be elucidated with a properly compensative load of 20.5 kN, which is also the top boundary of this model. Moreover, the left, right, and bottom boundaries of the model are fixed constraint boundaries.

In accordance with the material proportioning of analogous rock layers (groups), similar materials were selected for the model. The model was then constructed in layers, with each layer measuring 2 to 3 cm in thickness. The layers were compacted and the surface was scraped smooth. Subsequently, the next layer was laid, and sensors were simultaneously placed at the designed positions. To ensure accuracy and consistency in layer thickness, a layering marking line was drawn on the baffle when laying each layer. This practice is advantageous in guaranteeing that the thickness of the layers aligns precisely with the intended design. In order to simulate the structural surface and enhance...
the layering effect, mica sheets were evenly sprinkled at the interface of each layer. This helped to provide a more accurate depiction of the boundary between the layers.

Dynamic tracking video and photography techniques were employed to capture the mining process of the model, while manual measurements of relevant movement and damage state variables were taken. Concurrently, sensor data were also collected during the process. Subsequently, computer image processing technology was utilized to analyze the transient features of the images. This includes processing the images to track changes in the overburden rock movement and damage state during the mining process. The downward and fully mechanized mining method was used in the panel within 5 months. The focus was on observing the overburden rock damage, top plate displacement, and stress changes.

3.3. Numerical Simulation

The 3DEC numerical simulation model for the dipping and trending profiles was meshed differently. The dipping profile model was meshed at dimensions of $300 \times 170 \times 30$ m, while the trending profile model was meshed at dimensions of $200 \times 170 \times 30$ m. Both models were segmented into ungrouted and grouted scenarios, with the different critical parameters highlighted on the nodal surface between these scenarios showed in Figure 5. The trending profile model had a total of 100,390 blocks and 201,484 zones as depicted in Figure 5a, while the dipping profile model consisted of a total of 173,640 blocks and 427,151 zones, as showed in Figure 5b. All the other parameters in the models were consistent in terms of settings and values.

![Figure 5](image-url)  
**Figure 5.** Different profiles of ungrouted and grouted scenarios in the numerical simulation models. (a) Trending profile of the ungrouted model. (b) Trending profile of the grouted model. (c) Dipping profile of the ungrouted model. (d) Dipping profile of the grouted model.
The numerical simulation involving mining methods and boundary retention aligns with the similar material model tests. Downward mining with spontaneous roof caving was adopted in the simulation. The mining length of the No. 10 coal seam is 230 m, with a mining thickness of 4.6 m in the ungrouted area (Figure 5c) and 4.2 m in the grouted area (as shown in Figure 5d). During the mining process, the advancement was performed in increments of 10 m each time. The right boundary coal pillar in the trending section was left with 40 m, the left boundary coal pillar had 30 m, and each of the left and right boundary coal pillars in the dipping section was 35 m. The strength parameter of the jointed surface was assumed to be 1/20 of the rock mass. The numerical simulation was calibrated by adjusting the strength values of each stratum and comparing the simulated results with those from the engineering geo-mechanics model in the laboratory. This parameter plays a critical role in determining the response of the jointed surfaces to mining activities and boundary retention measures.

4. Results and Discussion

4.1. Design and Implementation of Grouting Boreholes

In the study area, the No. 4 aquifer is known for its relatively shallow burial depth, measuring an average of 270 m, along with its irregular thickness and intricate lithology. The weathering zone of the underlying bedrock varies significantly in thickness, ranging from 4.5 m to 45.6 m, with a prevailing range of 15 m to 25 m and an average value of 22 m. Additionally, there is a pronounced strong weathering zone with a thickness ranging from 0.41 m to 37 m, averaging about 13 m. In response to the challenges posed by these geological conditions [24], squeezing and splitting grouting techniques have been employed to create a consolidated layer with a fixed skeleton within the No. 4 aquifer, as shown in Figure 6. This process reduces the water saturation of the reformed strata, transforming the aquifer into a weakly water-rich or nearly drained Class III water body. Simultaneously, this grouted treatment enhances the overall strength of the No. 4 aquifer, thereby mitigating the risk of sudden water–sand mixture inrush [25] during underground mining operations. Moreover, this approach facilitates the extraction of resources from the shallow coal pillars. On the basis of the grouted treatment of the No. 4 aquifer, it is recommended to extend the grouted drill holes into the weathering zone of the bedrock to reinforce this zone. This proactive step aims to prevent water–sand mixture inrush incidents during underground mining activities, ultimately ensuring the safety and stability of such operations.

![Figure 6. Grouting borehole layout for 1010-1 panel.](image-url)
The construction process involves the utilization of the vertical boreholes arranged row by row to disperse the water and consolidate the sand. The boreholes, as indicated in Figure 7, have specific structural features. The first section of boreholes, extending approximately 25 m above the bedrock surface, has a hole depth of around 245 m and a diameter of 215.9 mm. These boreholes are encased in a Φ177.8 × 8.05 mm casing and cemented. The second section of boreholes features with a diameter of 152 mm, with downstream segmented grouting governance and a unit section length of 4 to 6 m. Finally, the third section of boreholes extends to approximately 10 m below the top surface of the bedrock.

The grouting project implemented a segmented downstream approach, utilizing both the orifice-closed static pressure grouting method and the orifice stopping method [26]. It incorporates a combined grouting process that combines continuous and intermittent grouting techniques. The No. 4 aquifer and the weathering zone of the bedrock are both the primary targeted layers for grouting. In terms of materials, the grouting mixture comprises the general silicate P.O 32.5 cement and fly ash obtained from coal-fired power plants.

The drilling, coring, grouting parameters, and operations adhered to the design specifications, yielding satisfactory results throughout the construction process. By 10 March 2021, the on-site drilling and grouting activities were completed after 191 days, encompassing the completion of 42 boreholes. A cumulative drilling length of 11,790.03 m, with coring of 243.03 m, and a total of 336 scans of boreholes were conducted. In addition, inclination measurements for boreholes spanned 11,790.03 m, accompanied by 44 pressure water tests. The installation of casing required 365.880 tons of cement, while the overall grouting volume for the No. 4 aquifer and the weathering zone of the bedrock reached 70,267 tons, comprising 57,742 tons of cement and 12,525 tons of fly ash [27].

Post-assessment after grouting confirmed that the checking boreholes from ground met the designated requirements. The grouting spread radius, which exceeded 50 m, demonstrated a diffusion effect that was more satisfactory than expected. Notably, the
water influx from each checking boreholes did not surpass 0.5 m³/h, aligning with the anticipated effectiveness of the project.

4.2. Field Detection after Being Grouted

The analysis for the feasibility of grouting the No. 4 aquifer based on the field project is presented in Figure 8. The overall grouting ability was measured at 50.8 t/m; with the injectability, it was reduced to 47.7 t/m when excluding the pre-existing grouting holes Z3 and Z4. The unit grouting amount of the coring inspection boreholes averaged 15.9 t/m. However, for boreholes J4, B1, and B2, the unit grouting amounts were 19.2 t/m and 43.6 t/m, respectively, with an average of 26.8 t/m. It is important to note that after grouting, the injectability of the inspection boreholes was significantly less than the overall injectability. The middle and lower sections were more injectable than the upper section, as shown in Figure 9.

Figure 8. Contour map of unit grouting volume distribution over boreholes.

Figure 9. Statistics for grouting quantity ratio in vertical sections.
The compressive strength of the samples from the weathering zone of the bedrock ranged from 3.03 to 11.10 MPa, significantly exceeding that of the bedrock section in the cored borehole before grouting, which ranged from 1.96 to 2.18 MPa. The grouting treatment enhanced the strength of the weathering zone of the bedrock by approximately 2 to 5 times. Compared to the test results of the cored borehole samples before being grouted, there was a slight increase in the true density and an slight improvement in the overall compressive, tensile, and shear strengths. Additionally, a marginal decrease in water content was observed in the samples following grouting.

According to the water pressure test data obtained from the boreholes, it was determined that the water permeability of the core boreholes decreased significantly after grouting compared to before the grouting process. Prior to the treatment, a pumping test was carried out on borehole Z12, followed by tests on boreholes J4 and B1 post treatment. The results from these tests allowed for the derivation of the hydrogeological parameters for the test section, which revealed that the No. 4 aquifer within the grouting area generally displayed a low water-yielding characteristic overall.

Upon entering the No. 4 aquifer, it was noted that the drilling holes were prone to collapsing and jamming, as reported in the underground disclosure. While water was initially present in the No. 4 aquifer, the water flow dwindled rapidly to 0 m$^3$/h in some boreholes, with others showing a complete absence of water. These observations suggest that the No. 4 aquifer had transitioned into a weakened aquifer state.

Samples of the water-bearing sand of the No. 4 aquifer were collected from the boreholes for analysis. The samples revealed a significant presence of cement, indicating the extensive spread of grouting slurry over a considerable distance into the No. 4 aquifer. Analysis of the 27 underground inspection boreholes indicated a diffusion radius of ≥51 m during grouting, with the 1#, 2#, and 3# inspection boreholes exhibiting a diffusion radius of ≥71 m during grouting. Furthermore, sampling and analysis of the weathering zone demonstrated that the cement spread to considerable depths during grouting, effectively enhancing the strength of the bedrock. The results from the 27 underground inspection boreholes illustrated the significant transformation effect of grouting, successfully extruding water from the No. 4 aquifer within the grouting section and solidifying the sand layer as intended.

4.3. Simulation of Similar Materials in Trending and Dipping Directions before/after Being Grouted

The study focused on analyzing the impact of grouting on the stability and integrity of structures in geo-technical engineering and underground mining projects. To achieve this, a simulation of the engineering geo-mechanical model was conducted to investigate the development of fissures, displacement, and stress evolution of the roof overburden during the mining progression of the No. 10 coal seam in the 1010-1 panel of the Wugou coal mine. The model, illustrating the mining progression from right to left, is depicted in Figure 10, showing two engineering geo-mechanical models for the trending profile. One model represents the No. 4 aquifer and the weathering zone without grouting, while the other model corresponds to the grouting condition.

Excavation in the No. 10 coal seam within the trending profile resulted in varying heights in the grouted and ungrouted areas, reaching 4.2 m and 4.6 m, respectively (Figure 10). The advancement of the panel caused stress disruptions within the quarry, leading to the initiation and upward development of cracks [28]. In the grouted area, the caving zone expanded to a height of 15.2 m, while the water-conductive fracture zone extended up to 56.6 m without fully penetrating into the No. 4 aquifer.

Additionally, the No. 3 aquiclude clay layer, which overlays the No. 4 aquifer and possesses better plasticity and a greater thickness, did not significantly impact the aquifers above it. Moreover, on the left side of the ungrouted area (Figure 10e,f), where the coal seam is buried deeper with an overburden thickness exceeding 40 m and the mechanical properties of the rock layer are superior, the overall damage was less compared to the right
side (Figure 10c,d). The measured height of the caving zone in this area was 18.6 m, with the height of water-conductive fracture zone being approximately 42.6 m, which was lower than that observed in the thinner bedrock area.

![Images](image_url)

**Figure 10.** Photos taken from material test in trending and dipping directions before/after being grouted. (a) Pre-mining of trending profiles. (b) Post-mining of trending profiles. (c) Pre-mining of dipping profiles (grouted). (d) Post-mining of dipping profiles (grouted). (e) Pre-mining of dipping profile (ungrounded). (f) Post-mining of dipping profile (ungrounded).

In the model, as shown in Figure 10b, the trending profile advanced to the left from the opening cut on the right side of the model. Once the panel was fully mined out, the caving zone and water-conductive fracture zone were basically developed and stabilized. The weathering zone, post grouting, displayed minor fractures, exhibiting significantly reduced overall damage compared to the ungrouted model, as depicted in Figure 10d,f. There were absciss layers at the bottom boundary of the No. 3 aquiclude, and the other absciss layers gradually tended to close. In the ungrouted model (Figure 10f), the caving zone reached a height of approximately 20.3 m after the completion of mining operations, with the water-conductive fracture zone extending up to 62.4 m, penetrating into the No. 4 aquifer. Following grouting (Figure 10d), the model forecasted the caving zone of the trending profile of the panel after mining to range from 16.4 m to 20.2 m in height, accompanied by a water-conductive fracture zone measuring 53.4 m [29].

Stress measurement points in the trending and dipping profile were designed and are shown in Figure 11. Figure 11b depicts the stress variation curves along the survey line at various advanced distances within the trending profile. The model indicates that the original stress equilibrium state of the overburden rock was disrupted upon the excavation of the panel from the opening cut. With the advancement of the panel, the stress state
of each measurement point along measurement line 1 underwent continuous alterations, with the stress value following a pattern of “stable–slowly rising–suddenly falling–stable”. Similarly, measurement line 5 demonstrated incremental changes in the stress values at each measurement point, with an overarching trend of “rise–fall–balance” changes. In comparison to the stress change pattern observed in line 1, certain measurement points in line 5 manifested a more moderate stress variation trend. This moderation can be attributed to the distance from the mining area and the presence of an overlying loose layer, resulting in a more moderate stress change trend within this region.

Figure 11. Stress measurement points in the model. (a) Layout of stress measurement points in the trending and dipping profiles. (b) Stress curves of survey line at different advanced distances in the trending profile. (c) Geostress curves of survey line at different advanced distances in the trending profile.
The stress evolution curves before and after being grouted revealed that the stress fluctuations at each measurement point were reduced to varying degrees post grouting. Moreover, the stress difference between the initial stress equilibrium state and the final equilibrium state decreased. This mitigation of the stress concentration indicates an improvement in the engineering properties of the weathering zone of the bedrock and No. 4 aquifer, leading to increased strength. Figure 11c shows the stress variation curves of the survey line at different advanced distances in the trending profile. The No. 4 aquifer forms a stable skeleton structure, which can, to a certain extent, share the load of the upper loose layer and transfer it to the areas on both sides of the quarry, thereby reducing the degree of stress concentration at each measurement point.

To monitor the evolution of roof displacement during mining, a camera was employed to observe the vertical displacement at each measuring point of the displacement-coded measuring line [30]. These observations were then simultaneously combined with the calibration of the rope-pulled displacement sensor to mitigate errors generated by the manual measurements, as illustrated in Figure 12a.

During the initial stages of mining, cracks propagated upwards along the quarry, continuously intersecting with the direct roof strata. This led to the gradual deterioration of the direct roof, which exhibited a tendency to collapse downwards. Upon completion of the panel advance, the caving zone and the water-conductive fracture zone became essentially developed and stabilized. This process is illustrated in Figure 12b, which depicts the dynamic evolution of the displacement of each line of measurement at different advanced distances in the trending profile. As a result of these developments, the emergence of a “double peak” phenomenon in the top plate sinking curve can be observed.

Figure 12c depicts the dynamic evolution of the displacement of survey lines at various advanced distances for the grouted and ungrouted trending profiles. The displacement of the middle measurement points was slightly greater than that of the measurement points on both sides of the model. This observation aligns with the “saddle-shaped” destruction of the roof overburden following mining activities. This tendency was consistent with the progression of the model from right to left. Compared with the displacement change curve of each measurement point in the ungrouted model, the displacement change of each measurement point after being grouted was slightly slower. After the grouting treatment of the No. 4 aquifer, the displacement of each measurement point decreased to varying degrees. Furthermore, as the quarry displacement stabilized, each measurement point in the ungrouted model entered a phase of smooth fluctuations of the displacement. This transition indicates an increase in the internal strength of the aquifer after the grouting treatment, consequently resulting in a slower trend of destruction of the overburden rock.

The phenomenon of a gradual decrease in displacement at measurement points further away from the quarry in the vertical direction can be partly attributed to the development of absciss layers, which were not completely closed. This gradual decrease was also influenced by the crushing and swelling of broken rock, as they can fill void areas and fissures. Consequently, the displacement of the measurement point decreased as the distance from the quarry increased, reflecting the combined effect of these factors [31].
The stress evolution curves before and after being grouted revealed that the stress fluctuations at each measurement point were reduced to varying degrees post grouting. Moreover, the stress difference between the initial stress equilibrium state and the final equilibrium state decreased. This mitigation of the stress concentration indicates an improvement in the engineering properties of the weathering zone of the bedrock and No. 4 aquifer, leading to increased strength. Figure 11c shows the stress variation curves of the survey line at different advanced distances in the trending profile. The No. 4 aquifer forms a stable skeleton structure, which can, to a certain extent, share the load of the upper loose layer and transfer it to the areas on both sides of the quarry, thereby reducing the degree of stress concentration at each measurement point.

To monitor the evolution of roof displacement during mining, a camera was employed to observe the vertical displacement at each measuring point of the displacement-coded measuring line [30]. These observations were then simultaneously combined with the calibration of the rope-pulled displacement sensor to mitigate errors generated by the manual measurements, as illustrated in Figure 12a.

Figure 12. Schematic layout of displacement coding point survey line for model test. (a) Layout of modeled displacement measurement points in the trending and dipping profiles. (b) Dynamic evolution of the displacement of each survey line at different advanced distances in the trending profile. (c) Dynamic evolution of displacement of survey lines at different advanced distances for grouted/ungROUTed trending profiles.

4.4. Numerical Simulation of Trending and Dipping Profiles before/after Being Grouted

Figure 13a presents a cloud view of the quarry displacement upon completion of the trending profile. In the pre-mining stage, grouting produced a beneficial effect on controlling the fractures of the rock formation and reducing displacement. The range of large displacement areas in the grouting model was significantly smaller than that in the ungrouted model, as shown in Figure 13c,d. Notably, through the opening cut and the left side of the stopping area, it was evident that the displacement-affected area in the ungrouted model was considerably larger [32]. This highlights that grouting into the No. 4 aquifer can effectively reduce the displacement around the panel, thereby reducing the subsidence area scope.
Figure 13. Quarry displacement of numerical models during mining operation. (a) Cloud view of quarry displacement at the completion of the trending profile (ungrounded on the left, grouted on the right). (b) Cloud view of quarry displacement at the completion of the dipping profile (ungrounded on the left, grouted on the right). (c) Schematic layout of displacement measurement points in the trending and dipping profiles. (d) Numerical simulation of displacement curves of some measurement points in the trending profile before/after being grouted. (e) Numerical simulation of displacement curves of selected measurement points in the dipping profile before/after being grouted.
Figure 13b presents a cloud view of the quarry displacement upon completion of the dipping profile. At the end of the simulated panel, the caving zone and water-conductive fracture zone were significantly developed and stabilized throughout the quarry. In the displacement cloud of the ungrouted model, the area with a larger displacement change was noticeably larger compared to the range of the grouted model on the right side, as shown in Figure 13c,e. This indicates that the overall displacement of the ungrouted model on the left side was relatively larger, which is basically consistent with the results observed in the model test. Furthermore, the subsidence level of the overlying rock of the ungrouted model was larger than that of the grouted model, and the degree of destruction was more severe [33].

In the early stage of mining, the model showed that as the panel advanced from the left side, the original stress equilibrium state of the rock was disrupted, leading to a redistribution of the stress field in the overburden. The stress distribution in the entire quarry exhibited a “landing funnel” pattern [34], which extended from the upper part of the open area to the model boundary. The upper part of the open area showed a parabolic pattern of stress reduction, while the two sides of the open area showed prominent concentration zones, indicated by light green coloring.

Figure 14a presents a cloud view of the quarry stress upon the completion of the trending profile. In the ungrouted model, as the panel progressed, the mudstone layer above the basic and key stratum was visibly broken, leading to stress release. Conversely, in the grouted model, there was an improvement in the geo-technical properties of the overlying weathering zone and the No. 4 aquifer. This enhancement is shown in Figure 14c,d where the No. 4 aquifer formed a stable “skeleton” structure within the zone, effectively preventing damage and disruption to the overlying strata. With the grouting treatment, the roof damage exhibited a degree of hysteresis when compared to the ungrouted model. Subsequently, after the completion of the panel, the roof strata made contact with the coal seam floor, resulting in the re-generation of stress connections.

Figure 14b depicts a cloud view of the quarry stress at the end of the dipping profile. As the panel progressed, the “landing funnel” extended towards the overlying No. 3 aquiclude. It was observed that the stress fluctuation state of the quarry remained essentially unchanged before and after being grouted, as noted by previous research [35]. A significant contrast, however, was evident in the reduction in the maximum compressive stress of the quarry after grouting in comparison to pre-grouting conditions, as shown in Figure 14c,e. In the middle section of the mining area, a stress concentration area formed due to the bending and sinking of the overlying strata. After the completion of the panel mining and stabilization to a steady state, there was a noticeable similarity in the distribution of stress figures in the quarry before and after being grouted.

The fissure development pattern in the trending profile at the end of mining is depicted in Figure 15a. The panel advanced from the right side (the opening-cut location) to the left (the stopping line), and as the buried depth increased, the fissures in the roof overburden gradually developed. The measurements indicate that in the ungrouted model of the trending section, the height of the caving zone was 18.91 m, and the height of the water-conductive fracture zone was 74.47 m. The maximum height of the water-conductive fracture zone was situated above the opening cut area. In the grouted model (Figure 15b), upon completion of mining, the height of the caving zone was 13.82 m, and the height of the water-conductive fracture zone was reduced to 61.14 m. Similarly, the maximum height of the water-conductive fracture zone remained above the opening-cut area. These reductions in the height of the caving zone and water-conductive fracture zone indicate that the process of grouting the No. 4 aquifer effectively transformed the stratum’s properties and mitigated the damage from roof overburden. Moreover, the grouting modification caused a certain degree of hysteresis in roof damage [36], decelerating the rate of roof degradation in comparison to the ungrouted scenario. Additionally, post-grouting modifications enhanced the overall integrity of the No. 4 aquifer and the weathering zone, ultimately increasing their strength [37]. This improvement led to
a decrease in the roof plate subsidence compared to the pre-grouting condition, which can consequently reduce surface subsidence to a certain extent.

Figure 14. Quarry geostress of numerical models during mining operation. (a) Cloud view of quarry stress at the completion of the trending profile (ungrouted on the left, grouted on the right). (b) Cloud view of quarry stress at the completion of the dipping profile (ungrouted on the left, grouted on the right). (c) Schematic layout of stress measurement points in the trending and dipping profiles during numerical simulation. (d) Numerical simulation of stress curves of some measurement points in the trending profile before/after being grouted. (e) Numerical simulation of stress curves of some measurement points in the dipping profile before/after being grouted.
Figure 15. Numerical simulation of fractures before and after being grouted. (a) Fissure development pattern after the end of mining in the trending profile (ungrouted on the left, grouted on the right). (b) Fissure development pattern after the end of mining in the dipping profile (ungrouted on the left, grouted on the right). (c) Numerical simulation of fractal dimensions of the fissure before/after being grouted with different advanced distances (trending profile on the left, dipping profile on the right).

Quantitative calculations were performed to determine the fractal dimensions of the overburden fissures at various mining stages, both before and after grouting. These calculations provided insights into the progression of the fracture generation, connection, and development induced by mining activities. In the numerical simulation of the trending profile, the evolution of the fractal dimensions of the fractures was analyzed by plotting them against the advanced distances before and after grouting, as shown in Figure 15c. It was evident from the analysis that the fractal dimension graphs of both the grouted and ungrouted models exhibited characteristic distribution patterns, demonstrating three variable dimensional cycles and one steady dimensional stage throughout the coal seam mining process [38].
Under the stress of the overlying rocks, geostress gradually concentrated at the opening cut and the foot of the quarry. When the geostress reached the compressive strength limit of the main roof, both the main roof and immediate roof broke, initially forming a caving zone. This marks the first “ascending–descending” dimensional cycle, as shown in Figure 15c. With further panel advancement, the overlying rock layer bent and sank as a whole, creating the second “ascending–descending” dimensional cycle as the mudstone layer overlying the main roof broke and collapsed downward. This process also led to rapid development of fissures in the longitudinal direction. Subsequently, at this point, the fractal dimension of fractures reached its maximum value, marking the formation of the third “ascending–descending” dimensional cycle. As the panel continued to advance, fissures developed more prominently along the lateral direction. The fissures in the advancing area continued to develop and expand, while those in the central area gradually compacted. Eventually, the mining activities induced the overburden disturbance to reach a stable state, where the fissures on both sides of the quarry were more developed, and those in the central quarry were mostly compacted [39]. During this stage, the fractal dimension curve stabilized, entering a phase of smooth fluctuation. It is worth noting that the fractal dimension of the fissures was slightly larger before grouting compared to after grouting, suggesting that the overlying fissures in the ungrouted model were more extensively developed, resulting in greater damage to the overlying rocks in the main roof.

4.5. Geophysical Validation and Analysis of the Mining Process

The geological column diagram of the 1010-1 panel indicated that mudstone and sandstone predominated in the 42 m overburden strata until the No. 4 aquifer’s layers are encountered. The data collected from the drill holes were processed using a full-space laminar model, and a whole-space 3D resistivity inversion method was applied to obtain horizontal resistivity imaging maps at various heights for the main roof of this panel. Through this analysis, relatively low resistance zones were primarily identified based on apparent resistivity values less than 14 Ω·m.

Figure 16 shows the results of the inter-hole electrodynamics for detecting apparent resistivity at various elevations, depicting horizontal slices ranging from 5 m to 50 m on the roof of the 1010-1 panel. The apparent resistivity slices displayed in Figure 16 revealed the presence of a predominantly low resistance area, which was primarily located amidst the sandstone and mudstone layers on the roof of the 1010-1 panel. This observation stems from the results of the inter-hole parallel electrical method exploration. From the integration of these findings with the geological data and on-site drilling observations, it was inferred that the low resistance area may be caused by the influence of grouting or fissure water within the sandstone layer on the roof. Furthermore, the resistivity slices at 45 m and 50 m on the roof portrayed the situation of the No. 4 aquifer within the management section. The majority of the regions in these slices exhibited high visual resistivity values and a low water content, indicating that the grouting transformation in this section had been successful.

According to the findings from the inter-hole parallel electrical method [40,41], the effectiveness of the grouting transformation in the overlying No. 4 aquifer and the weathering zone of the 1010-1 panel was evident. Although there were still localized areas with low resistivity values [42], the majority of the area showed higher apparent resistivity values, which points toward a remarkable improvement resulting from the grouting process.
Table 2 presents the height calculation results of the safe coal–rock pillar in the 1010-1 panel using different methods. At the Wugou coal mine, a total of eight geological detection holes were drilled to measure the caving zone and water-conductive fracture zone on specific closed panels. The maximum measured caving height ratio of the neighboring 1016 panel was 4.69 times higher. Consequently, the caving zone’s height in the shallow area of the 1010-1 panel was determined to be $4.69 \times 4.2 \text{ m} = 19.70 \text{ m}$.

Table 2. Calculation results of the height of caving zone of 1010-1 panel using different methods.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Mining Regulations</th>
<th>Measured</th>
<th>Material Test</th>
<th>Numerical Simulation</th>
</tr>
</thead>
<tbody>
<tr>
<td>Height of caving zone (m)</td>
<td>13.04</td>
<td>19.70</td>
<td>16.10</td>
<td>17.10</td>
</tr>
</tbody>
</table>

The 1010-1 panel was designed to mine up to an elevation of $-262.96 \text{ m}$, where the underlying bedrock elevation is $-240.11 \text{ m}$. The grouting transformation section of the No. 4 aquifer exceeded a thickness of 20 m. The actual upper limit elevation of this panel was $-262.96 \text{ m}$. The actual minimum height of the collapse prevention pillar was measured at 43.19 m, surpassing the calculated values in Table 2.
The stress monitoring data from the anchor bar at the track roadway and the working resistance data from the hydraulic support in the panel were analyzed. As the excavation of the 1010-1 panel progressed, the stress data from the track roadway anchor bar continued to change, as shown in Figure 17. Prior to January 29, 2023, the stress level of the anchor bar at the wind roadway was relatively stable [43]. However, as the 1010-1 panel advanced approximately 9.5 m, the stress on the anchor began to gradually increase due to the impact of mining on the surrounding rock at the roadway. The stress of the anchor at the track roadway increased rapidly at about 27.1 m of the panel advancement, reaching a peak at about 40 m of the panel advancement. Following this peak, the stress on the anchor suddenly decreased and stabilized rapidly. Moreover, the field measurements illustrated that none of the hydraulic supports reached their designed working pressures during the coal seam excavation in the grouted area.

![Figure 17. The stress curve of anchor at track roadway.](image)

5. Conclusions

The primary conclusions and findings can be succinctly summarized as follows:

(1) A thematic and systematic assessment of the hydrogeological and engineering geological conditions of the Wugou coal mine was conducted. This involved an analysis of the geological structural characteristics of the main roof of the 1010-1 panel and an examination of the water-related properties of the aquifers and aquicludes. The hydrogeological parameters were evaluated, revealing that the main roof of the No. 10 coal seam is susceptible to the influence of the water-conductive fracture zone, which poses a safety risk in terms of water–sand mixture inrush occurrences. Consequently, this necessitates the implementation of grouting in the No. 4 aquifer and the weathering zone to prevent and mitigate this hazard.

(2) Based on the information obtained from the grouting boreholes on the surface, and the underground detection drill holes in the 1010-1 panel, an analysis was conducted to assess the changes in the hydrogeological parameters and rock mechanical strength parameters after grouting. These results showed that after being grouted, the No. 4 aquifer and the weathered zone exhibited a low water content, and the mobility of sand was poor. The No. 4 aquifer and the weathered zone were transformed from an aquifer to a loose pore, weak aquiclude. This indicates that the grouting transformation was successful in achieving the desired outcomes of water expulsion from the No. 4 aquifer and consolidation of the sand in the grouting section.

(3) According to the geological prototype of the 1010-1 panel, we constructed engineering and geo-mechanical models and performed 3DEC numerical simulations of the geo-technical bodies in various profiles of this panel. Geostress and displacement monitoring were carried out, and the morphology of the roof damage process at different stages of the mining was simulated and quantified. Through the laboratory material model test and numerical simulation, it was found that the permeability of the geo-technical body weakened after grouting. Additionally, the overall mechanical strength properties were
enhanced, leading to a reduction in the height of the collapse and water-conductive zones resulting from mining activities.

(4) The observation information from the anchor at the track roadway and the apparent resistivity results obtained from the 3D whole-space resistivity inversion indicated that the No. 4 aquifer and the weathered zone had a low water content after grouting. The hydraulic supports in the grouted area did not reach the rated working pressure during the mining process. These findings collectively reveal the significant effect of grouting on the transformation of this study area.

Author Contributions: Conceptualization, performing simulations, and writing—original manuscript preparation: R.H.; methodology and conceptualization: W.S.; exploration of data and visualization: D.C.; validation and writing—review and editing: Y.L.; data collection and development of geo-technological model: R.L.; performing simulations: X.L.; underground sample collection and historical data description: G.C. All authors have read and agreed to the published version of the manuscript.

Funding: This research was funded by the National Natural Science Foundation of China, grant numbers 42130706 and 42202268.

Data Availability Statement: No new data were created or analyzed in this study. Data sharing is not applicable to this article.

Acknowledgments: The authors express their gratitude to the anonymous reviewers for their detailed comments and suggestions, which have greatly enhanced the quality of this paper.

Conflicts of Interest: Author Rongjie Hu and Daxing Chen were employed by the company Wanbei Coal-Electricity Group Co., Ltd. The remaining authors declare that the research was conducted in the absence of any commercial or financial relationships that could be construed as a potential conflict of interest.

References

2. Liang, Y.; Sui, W.; Jiang, T.; Shen, X. Experimental Investigation on the Transport Behavior of a Sand/Mud/Water Mixture Through a Mining-Induced Caving Zone. Mine Water Environ. 2022, 41, 629–639. [CrossRef]
9. Aboelezz, A.; Wetz, D.; Lehr, J.; Roghanchi, P.; Hassanalian, M. Intrinsically Safe Drone Propulsion System for Underground Coal Mining Applications: Computational and Experimental Studies. Drones 2023, 7, 44. [CrossRef]
15. Shi, L.; Ma, X.; Han, J.; Su, B. Identification of Limestone Aquifer Inrush Water Sources in Different Geological Ages Based on Trace Components. *Sustainability* 2023, 15, 11646. [CrossRef]


29. Li, Y.; Lei, X.; Wang, N.; Ren, Y.; Jin, X.; Li, G.; Li, T.; Ou, X. Study on the failure characteristics of overburden and the evolution law of seepage field in deep buried thick coal seam under aquifers. *Nat. Hazards* 2023, 118, 1035–1064. [CrossRef]


43. Li, Y.; Ou, X.; Ren, Y.; Wang, N.; Lei, X.; Jin, X. Research of the broken roof structure and supporting capacity of a shield in a deep and thick coal seam. Front. Earth Sci. 2022, 10, 961646. [CrossRef]

Disclaimer/Publisher’s Note: The statements, opinions and data contained in all publications are solely those of the individual author(s) and contributor(s) and not of MDPI and/or the editor(s). MDPI and/or the editor(s) disclaim responsibility for any injury to people or property resulting from any ideas, methods, instructions or products referred to in the content.