Test research on advance abutment pressure caused by coal mining based on optical fiber monitoring

Wengang Du1,2, Jing Chai 1, 2, Dingding Zhang 1,2, Zhe Ma1, Yongliang Liu1

1. College of energy engineering, xi’an university of science and technology, xi’an 710054, China
2. Xi’an university of science and technology, ministry of education of the western mining and mine disaster prevention and control of key laboratory, xi’an 710054, China;
3. College of safety science and engineering, xi’an university of science and technology, xi’an 710054, China

Abstract: The strata movement and deformation caused by underground mining activities is a typical "black box" problem, it is difficult to grasp the structural characteristics of complete overburden and the law of ground pressure. Abutment pressure is the vertical supporting force acting on coal and rock mass due to the redistribution of surrounding rock stress caused by underground engineering excavation. The distributed optical fiber and fiber Bragg grating are implanted into the physical similar model to monitor the distribution characteristics of abutment pressure in front of the working face during the whole excavation process. The results show that FBG monitoring to reach the peak strain needs to meet two conditions: the working face passes under it, and the cantilever beam structure above the working face is unstable and collapses, and the grating is located at the end of the suspended fixed beam. When FBG is located in front of the working face, the whole process is divided into four stages: 150-81 m from the grating is the weak influence area; 81-49.5 m is the significant influence area; 49.5-21 m from the grating is the severe influence area; 21-0 m from the grating is the influence recovery area. Based on this, the abutment pressure distribution model of FBG sensing detection is given. The influence range is within 81 m, the peak value is 21 m in front of the working face, and the pressure relief area is directly above the working face. The distributed optical fiber monitoring results show that the upward compressive stress value of the abutment pressure affected area decreases rapidly, and enters into the high roof tensile stress affected area. The tensile stress distribution presents a single peak shape, and gradually decreases after reaching the peak until it returns to the elastic area. The research results provide a new research idea for abutment pressure monitoring caused by mining.

Key words: mining rock mass; deformation detecting; abutment pressure; optical fiber sensing;
physical model test

1 Introduction

Abutment pressure is the vertical support force which is formed by the redistribution of surrounding rock stress caused by underground excavation. After coal mining, three distribution states of abutment pressure are formed around the working face, namely plastic zone, elastic zone and original rock stress zone, forming mining stress field. Abutment pressure distribution law, roof failure law and mine disaster control are the main research problems of mining engineering, and they are closely related. Accurate and reasonable calculation of abutment pressure distribution characteristics and influence range of stope has important engineering significance for optimizing advance support mode of mining roadway, accurately evaluating top coal fragmentation degree of fully mechanized top coal caving mining, accurately predicting coal and gas outburst and rock burst occurrence of deep roadway. At present, the research methods of abutment pressure mainly include roadway deformation observation, support load measurement, microseismic monitoring, laboratory similar simulation test, numerical calculation and theoretical calculation. The theoretical calculation methods mainly include limit equilibrium method, comprehensive movement angle method, elastic foundation beam hypothesis, continuous medium hypothesis and elastic-plastic theory.

Many researches have been done on the calculation and distribution of stope supporting pressure. In terms of theoretical research, Based on elastic-plastic theory, continuum theory and damage mechanics, Qian studied the distribution characteristics of abutment pressure, and considered that the movement of rock beam was the main factor causing the evolution of abutment pressure[1]. Based on theoretical analysis, numerical calculation and field measurement, the distribution law of the abutment bearing pressure of the working face in Yanzhou mining area is analyzed[2]. Liu put forward the viewpoint of dynamic and static abutment pressure in longwall mining stope. The formation of dynamic abutment pressure is the result of strata movement in stope fault zone, and the static abutment pressure is the result of load transfer of strata above fault zone[3]. By analyzing the forming process of abutment pressure, Song proposed a scheme that the comprehensive movement angle starts from the edge of abutment pressure, and established the relevant structural mechanics model[4]. Based on the viscoelastic damage model of abutment pressure distribution, Xie analyzed the advance abutment pressure and lateral solid coal abutment pressure[5]. Li thinks that the advance abutment pressure of working face is composed of original rock stress and concentrated stress caused by mining. In order to facilitate calculation, the concentrated stress is decomposed into constant abutment pressure caused by overburden weight and variable abutment pressure caused by roof rock movement[6]. Chen established a simple mechanical model by using statistical damage mechanics to study the distribution of abutment pressure under given deformation conditions[7].

The laboratory research of abutment pressure mainly includes physical similarity simulation test and numerical simulation calculation. Jiang used FLAC3D finite element software to simulate the vertical stress field of coal body during the advancing process of isolated island working face, and analyzed the dynamic variation law of abutment pressure of C-type stope[8]. Through numerical simulation, Si obtained the influence degree of working face advancing distance and length, coal seam thickness and buried depth on abutment pressure concentration factor and distance between peak point and coal wall[9]. In fact, at present, the main testing tools of abutment pressure distribution depend on
electronic stress sensors, whether for physical model or field measurement. The main disadvantages of this kind of sensor are complex wiring, easy to be affected by electromagnetic interference, and easy to rust and failure in underground humid environment. In order to achieve long-distance monitoring, a large number of sensors need to be arranged, which makes the monitoring system complex. With the rapid development of optical fiber sensing technology, it provides a new means for internal stress monitoring of coal and rock mass structure\cite{10-13}. The rapid development of optical fiber sensing technology (FOS) in recent years has provided a new channel for the deformation monitoring in geotechnical engineering field due to its advantages of small volume, light weight, anti-jamming, anti-corrosion, anti-electromagnetic interference, good durability and high resolution. Among them, the distributed optical fiber sensing technology based on Brillouin scattering principle (BOFS) is a hot research field in recent years. BOFS can realize long-distance distributed monitoring of structural health and is widely used in railways, bridges, tunnels, oil and gas pipelines, high-voltage lines, geotechnical structures and other fields. Since Horiguchi\cite{14} and Culverhouse\cite{15} et al. used the Brillouin scattering effect in optic fiber for distributed temperature and strain measurements for the first time, researchers in Japan, Canada, the United States, and other countries have conducted extensive research on both theory and experimentation\cite{16-20}. The distributed optical fiber sensor could be connected in parallel, or in series to install structure deformation monitoring network. BOFS monitoring of rock or soil structure deformation can be realized in the form of internal implant or surface adhesion\cite{21-22}. The commonly used optical fiber sensing technologies include Brillouin optical time domain reflectometry (BOTDR), Brillouin optical time domain analysis (BOTDA) and Fiber bragg grating (FBG). BOTDR has attracted much attention in engineering utilization because of its characteristic of being single ended and long distance monitoring\cite{23}. The advantage of FBG sensor is that it can accurately monitor the deformation and temperature distribution in a certain area. Its disadvantage is that it can not realize distributed monitoring because of its discrete monitoring mode\cite{24}. With the development of technology, the spatial resolution of BOTDA has reached centimeter level. High spatial resolution is very important for accurate monitoring of physical model test\cite{25}. Distributed optical fiber sensing technology has broad application prospects in geotechnical engineering.

However, the application of optical fiber sensing technology in the field of mining engineering is still in its infancy, so further exploration is needed to make the technology better promote the development of mining industry. In this paper, the three-dimensional model with the size of 1500\times600\times1200 mm based on optical fiber sensor detection were conducted. The distributed sensing fiber were embedded in the overlying strata, BOTDA was used to monitor the strain distribution and its variation as the mining progress. Through detailed analysis of the optical fiber implant pattern, temperature position method, and pre-stress apply principle, the deformation and distribution states of abutment pressure of overburden were analyzed.

2 Working principle of optical fiber sensing technology

2.1 Working Principle of BOTDA

The propagation of light in an optical fiber produces three scattering signals: Rayleigh scattering, Raman scattering, and Brillouin scattering. BOTDA technology based on Brillouin scattering principle,
inject pulsed light and continuous light at two ends of a complete fiber, stimulated Brillouin amplification occurs when the frequency shift between the two beams is equal to the Brillouin shift in a specific region of the fiber. The important feature of BOTDA is that ordinary single-mode fibers can be utilized as sensing media without special processing. Fig.1 shows the working principle of BOTDA. Compared with BOTDA system, PPP-BOTDA (Pulse-PrePump Brillouin Optical Time Domain Analysis) employs the pump pulse with leakage light to achieve a high spatial resolution. In order to gain higher spatial resolution, the usual way is to reduce the pulse beam width. However, as the excitation of phonons takes 28 ns, shortening the pulse width will lead to the decrease of Brillouin gain and the deterioration of spectral morphology. PPP-BOTDA technique excites phonons by changing the shape of the pump light and adding a pre-pumped pulse wave before the measured pulse light is emitted. With this method, the 1cm spatial resolution and the strain measurement accuracy of ±0.0025% can be obtained simultaneously. The frequency shift has a linear relationship with the strain and temperature at a certain location of the optical fiber. Therefore, by detecting the Brillouin frequency continuously, the strain and temperature distribution along the fiber can be obtained.

During measurement, the frequency of the two lasers is adjusted continuously to detect the power at the receiving end, and the frequency difference at the maximum energy transfer is determined [14]:

\[
\Delta V_B = V_{B(\epsilon)} - V_{B(0)} = C_1 \Delta T + C_2 \Delta \epsilon 
\]

\[
\epsilon = \left[ \Delta V_B - C_1 \Delta T \right] / C_2 
\]

In the formula: \(\Delta V_B\) is the change of Brillouin frequency shift; \(C_1\) is the sensitivity coefficient of the fiber to temperature; \(v_B(\epsilon)\) is the strain-induced Brillouin shift; \(C_2\) is the sensitivity coefficient of the fiber to strain; \(v_B(0)\) is the initial Brillouin frequency shift. The strain temperature information of the optical fiber is obtained by detecting the frequency shift variation of the Brillouin signal and the normalized signal power variation value to achieve distributed testing.

![Fig.1. working principle of Brillouin optical time domain analysis. The stimulated Brillouin amplification effect occurs when the optical fiber is subjected to stress and temperature changes.](image)

2.2 Working Principles of FBG

FBG is fiber Bragg grating sensor. This sensor can change the wavelength of the reflected light wave according to changes in ambient temperature and strain. Using internal writing or laser to form periodic defects on the fiber to change the refractive index of the core region, when the external parameters (temperature, stress) change, it will change the refractive index of the grating and cause the wavelength of the sensor to drift. It is obtained by detecting the wavelength drift; temperature and
strain satisfy the following relationship:

$$\Delta \lambda_B = K_\epsilon \Delta \varepsilon_g + K_T \Delta T$$  \hspace{1cm} (3)

Where: $\Delta \lambda_B$ is the FBG wavelength drift; $\Delta \varepsilon_g$ is the strain value, $\Delta T$ is the temperature value; $K\varepsilon$ is the sensor strain calibration coefficient; $K_T$ is the sensor temperature calibration coefficient.

3 Theoretical calculation model of abutment pressure

3.1 The forming stage of the abutment pressure

According to the existing mine pressure theory, the formation process of abutment pressure in long wall mining stope can be divided into five stages. The first stage: from the open-off cut to the coal wall in front of the working face before entering the plastic stage. Due to the gradual mining of the coal seam, the stress of the roof above the goaf is transferred to the surrounding coal wall. The stress of the coal wall in front of the working face increases gradually with the advancing of the working face, but at this time, the stress of the coal wall has not reached the yield limit of the coal body. The whole stress of the coal wall is still in the stage of elastic compression. The peak value of the supporting pressure is located above the coal wall in front of the working face, and decreases monotonously with the increase of the distance from the working face, As shown in Fig.2 (a). The second stage: from the plastic failure of coal wall to the end fracture of basic roof beam. After entering this stage, part of the coal body in front of the working face reaches the strength limit, plastic failure occurs, the bearing capacity decreases sharply, the overburden stress begins to transfer to the deep coal body, and the stress peak is formed at a certain distance in front of the working face. According to the elastic-plastic theory, the peak position is the boundary between the plastic zone and the elastic zone of the coal wall. Before the peak, the coal body is in the plastic state, and the abutment pressure distribution curve increases monotonously. After the peak, the coal body is in the elastic zone, and the abutment pressure curve decreases monotonously. The stress invariant section is the original rock stress zone, as shown in Fig.2 (b).

The third stage: the rock beam at the front of the basic roof breaks to the gangue in the middle of the basic roof rock beam. Before the rock beam fracture at the front end of the basic roof, the peak bearing pressure appears in the stress height concentration. After the rock beam breaks, the load transfers to both sides. The peak stress is formed before and after the fracture line is bounded. The internal stress field with the dead weight of the fractured rock block is formed on one side of the working face. The external stress field mainly consists of the whole overburden load transmission in front of the fracture line. The external stress field is much larger than the internal stress field. The fourth stage: from the high basic roof fracture to the gob "square" (the advancing length is equal to the working face length). With the advancing of the working face, the roof has periodic fracture and contact with gangue. The range of internal and external stress field increases with the advancing of the working face. When the advancing length is equal to the length of the working face, the range of internal and external stress field reaches the maximum. The fifth stage: from "seeing the square" in the goaf to stoppage. After entering this stage, the length of working face is limited, the height of fracture arch is no longer upward, and the roof rock layer is constantly suspended, fractured, sunk, rotated and touched with gangue. The height of fracture arch is basically 0.5-0.7 times the length of working face,
the dynamic balance of mining stress field with the advance of working face, the distribution of supporting pressure is constantly moving with the advance of working face, and the parameters such as distance between peak bearing pressure and coal wall and peak value are basically stable.

![Diagram of abutment pressure stages](image)

**Fig.2** The forming stage of the abutment pressure

### 3.2 Dynamic abutment pressure and static abutment pressure

The abutment pressure in front of working face is the superposition of original rock stress field and mining stress field. Therefore, the abutment pressure can be divided into constant abutment pressure and variable abutment pressure. Among them, the constant abutment pressure is caused by the self weight stress of overburden, which is related to the lithology and thickness of overburden; The variable abutment pressure is produced by the stress transfer of surrounding rock caused by mining, which is related to the range of rock movement and the severity of failure. Variable abutment pressure can be divided into static abutment pressure and dynamic abutment pressure. The static abutment pressure is caused by the continuous increase of the suspended area of the roof caused by coal mining, which is forced to transfer to the front and back and both sides of the coal wall by the load of the key strata under the roof, thus forming the static abutment pressure; The dynamic abutment pressure is that when the load acting on the basic roof rock exceeds its strength limit, the basic roof rock beam will fracture, resulting in the coal yield and loss of supporting capacity, and the stress will transfer to the deep part of the coal to form the dynamic abutment pressure. The difference between dynamic and static abutment pressures is that:

1. The formation mechanism is different. The static abutment pressure is formed by the load transfer
of the key strata, and the dynamic abutment pressure is formed by the stress transfer to the deep caused by the coal yield and loss of supporting capacity due to the fracture of the roof rock beam.

(2) The evolution cycle is different. The static abutment pressure is gradually formed with the increase of the critical layer area, and the formation time and evolution cycle are longer; The dynamic abutment pressure forms and evolves with the periodic fracture of roof, and the forming and evolution time is short.

(3) The scope of influence is different. Traditionally, the abutment pressure is considered as dynamic abutment pressure. Generally, the distance between the peak position and the working face is 2-3 times of the mining height. The influence range is 40-60 m and the lateral influence range is 30-40 M; The distance between static abutment pressure and coal wall is usually 6-10 times of mining height, and the influence range is 3-5 times of dynamic abutment pressure.

3.3 Calculation model of limit equilibrium method

When the overburden stress reaches the yield limit of the coal body in front of the working face, the plastic failure of the coal body will lead to the reduction of the bearing capacity, and the stress will transfer to the deep coal body, forming the three zone distribution of the bearing pressure. According to the limit equilibrium theory, the coal body in front of the working face forms plastic zone, elastic zone and original rock stress zone. The abutment pressure in plastic zone shows an exponential growth trend, while that in elastic zone shows a negative exponential trend. The stress level of coal body in original rock stress zone maintains the original in-situ stress state. The location of peak stress is the boundary of elastic-plastic zone.

![Fig.3 Theoretical calculation model of abutment pressure](image_url)

As shown in Fig. 3, the mechanical model of advance abutment pressure of working face is established. Among them, the advancing direction of working face is $x$ direction, and the vertical normal stress direction is $y$ direction, $\sigma_y$ is the normal stress in $y$ direction, $k\lambda H$ is the peak value of abutment pressure, $\lambda H$ is the original rock stress, $N_0$ is the vertical supporting force of coal wall, $x_0$ is the peak position, $x_1$ is the boundary position of elastic zone and original rock stress zone. Take the coal wall unit in the limit balance area for stress analysis as shown in the right in Fig.3, where $f$ is the friction factor between coal seams and $M$ is the thickness of coal seam, $\sigma_x$ is horizontal stress, and the stress analysis of horizontal direction is as follows:

$$M(\sigma_x + d\sigma_x) - M\sigma_x - 2\sigma_y f dx = 0$$

Assuming that the coal body in the plastic zone satisfies the Mohr-Coulomb criterion, the following
equation is satisfied:

\[ \sigma_y = \sigma_c + \frac{1 + \sin \phi}{1 - \sin \phi} \sigma_c \]  
\( (5) \)

Where: \( \sigma_c \) is the uniaxial compressive strength of coal, \( \phi \) is the friction angle in coal body. According to the condition of limit equilibrium zone, the abutment pressure of coal wall in plastic zone can be calculated according to the following formula:

\[ \sigma_y = \frac{N_0}{\lambda} e^{\frac{2\sigma_c}{\gamma}} \text{, } N_0 \leq \sigma_y \leq K\gamma H \]  
\( (6) \)

Where: the supporting capacity of the coal wall is:

\[ N_0 = \tau_0 \cot \phi \]  
\( (7) \)

The side pressure coefficient is:

\[ \lambda = \frac{1 - \sin \phi}{1 + \sin \phi} \]  
\( (8) \)

Order: \( \sigma_y = k\lambda H \). The distance between the peak bearing pressure and the coal wall can be obtained as follows:

\[ x_0 = \frac{m}{2f\lambda} \ln \frac{K\gamma H}{N_0} \lambda \]  
\( (9) \)

Where: \( k \) is the stress concentration factor, \( \gamma \) is the unit weight of floor rock, \( h \) is the buried depth. The calculation formula of abutment pressure in elastic zone is as follows:

\[ \sigma_y = K\gamma H e^{\frac{2f\beta(x_0 - x)}{K\gamma H}} \]  
\( (10) \)

The elastic zone is from peak point to original rock stress area \((x_t - x_0)\), and the end of elastic zone enters into the stress area of original rock, where \( \sigma_y = \gamma H \). Substituting into formula (9), the influence range of abutment pressure can be obtained as follows:

\[ x_t = \frac{m\beta}{2f} \ln K + x_0 \]  
\( (11) \)

3.4 Calculation results of abutment pressure under the background of this test

This paper takes the geological conditions of 52304 working face in Daliuta mine as the research background. The average thickness of coal seam in working face \( M=7.3 \text{ m} \), and the internal friction angle of coal seam \( \phi=22^\circ \). The friction coefficient between layers \( f=0.2 \), the field measured maximum stress concentration coefficient \( k=1.82 \), and the mining depth \( h=175 \text{ m} \); Unit weight of overburden \( \gamma =23.9 \text{ kN/m}^3 \); The cohesion of coal \( C=2.2 \text{ MPa} \); Uniaxial compressive strength of coal \( \sigma_c =0.45 \text{ MPa} \). According to formula (9), the distance between peak point and working face can be calculated:

\[ x_0 = \frac{m}{2f\lambda} \ln \frac{k\gamma H}{N_0} \lambda = \frac{7.3}{2 \times 0.2} \times \frac{1 + \sin 22}{1 - \sin 22} \times \ln \left( \frac{1.82 \times 23.9 \times 175}{2200 + 450 \tan 22} \times \frac{1 - \sin 22}{1 + \sin 22} \right) = 15.41 \text{ m} \]
By substituting the position of the peak point \( x_0 = 15.41 \) m into equation (10), the peak value of abutment pressure can be obtained as follows:

\[
\sigma_{y_{\text{max}}} = \frac{N_0}{\lambda} e^{\frac{2 \mu \lambda}{e}} = \frac{2.2 + 0.45 \tan 22}{1 - \sin 22} \times (1 + \sin 22) \times e^{\frac{2 \times 0.2 \times 15.41}{7.3}} = 11.3 \text{MPa}
\]

According to formula (11), the range of influence of the bearing pressure can be obtained:

\[
x = \frac{m \beta}{2 f} \ln K + x_0 = 56.8 \text{m}
\]

According to the limit equilibrium method, the peak value of abutment pressure in front of the working face is 11.3 MPa, the distance from the peak point to the coal wall is 15.4 m, and the influence range of abutment pressure is 56.8 m.

4 Construction of physical similarity model test

4.1 Construction of physical similarity model

The physical model test is intuitive compared with other research methods and can qualitatively or quantitatively reflect the mechanical characteristics of natural rock mass induced by underground engineering disturbance and the interaction between underground engineering supporting structures and underground engineering surrounding rock. It can truly simulate complex underground engineering structure, geological structure, underground strata combination relationship, and so on. With the increasing of underground coal mining engineering scale, the problems become more and more complicated, which makes the physical model test more and more practical engineering significance. According to the similarity theory, the experiment must be similar to the prototype system in geometry, kinematics and dynamics.

The simulated coal seam is 5-2 coal seam. According to the discrimination results of the key strata, the overlying strata structure of 52304 working face belongs to multi-key strata structure, the medium grain sandstone numbered 5 above the 5-2 coal seam is the sub-key strata structure, and the medium grain sandstone with the thickness of 13.43 m numbered 24 is the main key strata structure.

Considering the simulated rock formation thickness and the geometric size of the laboratory model frame, the geometric size of the model is 1500(length) × 600(width) × 1300 mm(height). Geometric similarity ratio is 1:150, the thickness of the overlying strata is 1.28 m when the thickness of the model coal seam is 5 cm, which meets the height limit of the model frame, and can be simulated to the surface loose layer without external load replenishment. On the one hand, the test system is simplified, on the other hand, the simulation similarity is improved. Bulk density similarity constant is the ratio of the prototype material bulk density to the model material bulk density. The average bulk density of the actual strata in coal mine is about 2.5 g/cm3. In the experiment, sand, gypsum, lime powder are used as the similar material. The average bulk density of similar material measured in laboratory is 1.6 g/cm3. So the bulk density ratio is 1:1.56. Physical model similar materials are usually prepared from several materials in a certain proportion. According to the requirements of similarity and previous research experience, it is determined that sand, pulverized coal, clay and mica (also as layered materials) are used as aggregates in each rock layer. Gypsum and lime powder are used as cementitious material and water as solvent material. According to the existing research conclusions, when the water cement ratio is 8%, the strength of similar materials is closest to that of prototype
materials. Because the main material of the model is river sand and the proportion of auxiliary material is relatively small, it does not affect the overall bulk density of the model material, so the density of all rock layers except coal seam is calculated according to 1600 kg/m³, and the density of pulverized coal is calculated according to 1300 kg/m³. In the proportion of coal seam, the proportion of sand, gypsum, lime powder and pulverized coal is 43%, 2%, 12% and 43%, respectively. After the selection of similar materials, according to the similarity constant determined above and the physical and mechanical properties of each rock layer, the proportion is determined by material ratio orthogonal experiment in order to meet the requirement of mechanical similarity. A real picture of the physical model is shown in Figure 4.

Fig. 4 Real photo of the three-dimensional physical model

4.2 Main measurement system in model test
Accurate scientific measurement is an important way to improve the reliability of physical similarity model test results. The physical and mechanical parameters of coal and rock mass affected by mining mainly include surface subsidence, displacement of overlying strata, internal strain of rock mass, distribution of abutment pressure and so on. The measurement methods used include BOTDA distributed optical fiber strain measurement system, FBG strain (temperature) measurement system, dial gauges ground surface subsidence measurement system, total station model displacement measurement system, three-dimensional digital speckle(DIC) model whole field deformation measurement system, pressure sensor measurement system and so on. The test preparation work mainly includes sensing optical fiber fusion, fiber spatial positioning, fiber grating connection, total station measuring point arrangement and instrument erection, dial gauges arrangement, model surface speckle rendering and DIC test system calibration and so on.

After years of development, optical fiber sensing technology has become more and more mature and has been widely used in large geotechnical engineering structures such as bridges, tunnels, dams, buildings and so on. With its unique advantages, it has become an excellent sensing technology to
replace the traditional mechanical or electromagnetic sensors. In this test, a total of 13 sensing fibers with diameter 2 mm were implanted in the model, including 4 vertical optical fibers and 9 horizontal fibers to monitor the deformation of rock mass in different layers and directions. In order to facilitate measurement, the optical fibers need to be welded together. The main factors considered in optical fiber fusion include: measurement time, convenience of measurement, system complexity, system risk and so on. If all the segmented optical fibers are connected into a complete loop, the measuring convenience and measuring time are optimal, but the measuring system is complex and the risk is high. If the optical fiber at a certain point in the model is broken, damaged and other irreversible mechanical damage occurs, the whole measurement system will fail. If there are too many optical fiber measuring loops, it will increase the unnecessary measurement time, and switching back and forth the measuring joint is easy to cause mechanical wear and tear to the interface of the testing equipment.

Considering the above factors, 13 sensing optical fibers are welded into 3 complete circuits. Loop 1 is four vertical fibers named V1, V2, V3, V4. Among them, the effective test length of V1 in the model is 1.29 m, V2 is 1.29 m, V3 is 1.28 m, V4 is 1.28 m. The sensing optical fiber needs to determine the specific position coordinates of each optical fiber in the model through spatial positioning, and the water bath heating method is used for spatial positioning. Based on the fact that the Brillouin frequency shift of distributed optical fiber has the characteristics of double sensitivity to temperature and strain, the spatial positioning can be realized by changing the temperature in the free part of optical fiber. The arrangement of optical fiber sensor is shown in Figure 5.
5 Analysis of similar model test results

5.1 Main test phenomena

When the working face is advanced to 63 cm, the strain increases to -244.7 με. In this process, the roof has no obvious change, only the gangue above the support rotates and collapses after the support is moved, which has a certain squeezing effect on the grating embedding position, causing strain growth. When the working face is advanced to 66 cm, the roof collapses violently, and the 200 mm thick rock layer on the upper part of the sub key layer collapses at one time. The collapsed rock layers are arranged neatly, forming an obvious boundary with the previous collapsed gangue, as shown in Figure 6 (a).

The monitoring data show that the pressure strain of grating is not rising and falling back, from -244.7 με reduced to -162.8 με. The comparison model shows that the new collapse rock and the stable rock mass on both sides are hinged to form masonry beam structure, and only a part of the gravity is borne by the gangue below. There is a large gap between the new caving rock and the existing gangue in the goaf, and the stress of the overlying broken rock block cannot be effectively transferred downward. On the other hand, because there are many gaps between the gangue where the grating is located, under the impact load, the rock block moves around, which reduces the compaction degree of the gangue in the position where the grating is located, which causes the pressure stress to decrease, but the process will end in the further compaction. The increase rate was 293.87%, 7.26 times that of the last promotion. The new fault block is closely hinged with the arc shaped rock beam in the fracture zone, and is located on the same arc track with the existing rock block. Due to the further expansion of the goaf area, the whole collapse zone and the fractured zone rock block are all subject to clockwise rotation and subsidence, and the goaf area is further compacted, and the separation gap above the grid is fully compacted and closed. Although the deformation of the whole overburden is not strong when the deformation is not promoted by 66 cm, the whole masonry beam structure is turned clockwise and sinking, which has a strong compaction effect on the position of grating, which causes the surge of pressure and strain.

![Image](image-url)

(a) working face advanced to 66 cm  (b) working face advanced to 69 cm

Fig. 6 FBG monitoring response of fracture compaction process in mined-out area

From propulsion 69 to propulsion 78 cm, the strain is basically maintained at -500 με. Nearby fluctuation, and it suddenly drops to -197.7 when it is pushed to 81 cm, the strain increment is 282.9 με and the increase rate is 65.38%, 3 times that of the last mining. When overburden is pushed to 81
cm, the failure form is returned from irregular form of "trapezoid and oblique parallelogram" to "approximate isosceles ladder shape", and new fracture line is formed along coal wall of working face, and the new fracture rock beam is rotated and sunk and the existing rock block in goaf forms hinge structure. Because there are many gaps in the grating, the right gangue will be lifted up after the right gangue is squeezed and sunk, which will cause the decrease of the pressure stress and decrease the absolute value of the measurement strain. With the advancing of 69 cm to 78 cm, the collapse range of goaf extends in the length direction, but it remains unchanged in the height direction. The influence of the collapse of the lower roof above the working face on the grating in the rear goaf is gradually weakened, and the change of grating strain is small. When the rock mass is pushed 81 cm, the newly broken rock mass has the extrusion action along the inclined direction to the goaf. The rock beam above the grating forms a warping structure, and the right side moves downward under the action of the extrusion pressure component, which causes the rock beam above the grating position to be warped up, which causes the pressure strain drop of the grating.

![Fig. 7 The impact of "diagonal quadrilateral" structure on FBG](image)

5.2 Analysis of abutment pressure monitored by fiber Bragg grating

FBG-02 is buried in the sub key layer, 50 cm away from the right boundary of the model and 12.5 cm away from the coal seam roof. The measurement results are shown in Figure 8. The graph on the left shows the change curve of strain with advancing distance, and the graph on the right shows the change curve of strain increase rate with advancing distance. It can be seen from the figure on the left that the strain value of the working face is small from the opening of the cutting hole to the advance to 84 cm, which is basically at the bottom ± 130 με changes within the system. It can be seen from the strain increase rate in the right figure that when the working face is far away from the grating stage, when the roof rock layer breaks and collapses, the strain increase rate will have a peak value, and the peak value corresponds to the intensity of the roof movement.

When the working face is advanced 24 cm, the horizontal direction is 66 cm away from the grating embedding point, and the strain increment is -14.38 compared with the last advance με. The strain increase rate is 273%, which is 24 times higher than that of 21 cm. The first collapse position of the top is directly detected at 24 cm, which indicates that the grating has high sensitivity to monitor the collapse process of the top with diameter 66 cm (99 m) away. The next convex peak in the right figure is that the strain increase rate reaches 487% when it advances to 33 cm. When advancing 33 cm, the basic roof of the working face falls for the first time, and the working face experiences the first
weighting process. After that, as the working face continues to advance, the strain increase rate reaches about 300% when it advances to 66 cm, 90 cm and 114 cm, which corresponds to the large period weighting process of the main key layer fracture working face of the model. When advancing to 45 cm, 54 cm, 108 cm and other positions, the strain increase rate is in the range of 100% - 200%, which corresponds to the small period weighting process caused by the sub key layer breaking.

Fig. 8 Measurement results of FBG-02 strain and strain increase rate of overburden deformation

Fig. 9 Monitoring results of advance abutment pressure of roof

Figure 9 shows the strain distribution measured by fbg-02 sensor during the period from cutting to advancing 84 cm. In this process, the strain is mainly negative, indicating that the measured strain is compressive strain. The grating is buried in front of the working face, which is mainly affected by the advance abutment pressure, resulting in the pressure strain of the rock. The whole process can be roughly divided into four stages: ab stage is from cutting to advancing to 36 cm, and the distance from grating is from 0 to 54 cm (150-81 m), which is the weak influence area; The BC section is 36-57 cm in advance and 81-49.5 m away from the grating, which is the significant influence area. The strain rate increases gradually and monotonically; The CD segment is from 57 cm to 69 cm, and the distance from the grating is 49.5 m to 21 m, which is a severe influence area; The de segment is 69 cm to 84 cm in advance and 21-0 m away from the grating, which is the recovery area of influence. The influence degree gradually decreases and the compressive strain gradually decreases. The maximum compressive strain appears at the position of advancing to 69 cm, that is, 21 m in front of the working face, and the minimum compressive strain appears at the position of advancing to 84 cm, that is, right above the working face. The distribution of pressure and strain measured by grating meets the distribution law of super front abutment pressure in front of general coal seam mining face. For the
simulated near shallow buried working face with large mining height, the influence range of advance abutment pressure is within 81 m, the peak position is 21 m in front of the working face, and the strain value is the smallest in the pressure relief area directly above the working face.

From 87 cm forward, the measured strain is changed from negative to positive, from -43 με Mutation 263.5 με, the increase rate was 116%, 1.84 times that of the last promotion. When pushing 87 cm, the working face has passed under the grating sensor, and the horizontal direction is 3 cm ahead of the embedded position of the grating. The strain is positive, indicating that the rock stratum has changed from compression to tension. The grating is located in the cantilever beam structure in front of the broken line. The fracture fracture is developed near it, indicating that the roof fracture mode is tensile failure. When the working face is advanced to 90 cm, the strain changes from 263.5 με 3 to 3799 με. The maximum value of the whole monitoring process of FBG-02 sensor is reached. When the working face is pushed 90 cm, the working surface has been buried at the lower part of the grating and 3 cm ahead of the time.

Compared with FBG-01 measurement results, the two conditions to reach the peak are basically the same: fbg-01 is the working face advancing to 45 cm to reach the peak (through the next advancing process directly below it), and it is also the first periodic weighting process of the working face; FBG -02 is the seventh periodic weighting process when the working face is pushed to 90 cm to reach the peak (through the next pushing directly below it). The law is summarized as follows: to reach the peak strain, two conditions must be met: the working face passes through the next advance below, and the cantilever beam structure above the working face is unstable and collapses, and the working face experiences the weighting process. The difference between the two is: the peak of FBG -01 is due to the small roof caving height, and the suspended fixed beam structure is directly above it. When FBG -02 reaches the peak, the fault zone has developed to the lower part of the main key layer, and its location is in the range of fracture zone, but the surrounding rock structure is complete, forming a masonry beam structure.

6 Conclusions
(1) FBG monitoring process can be divided into two parts, the first half is the grating located in the roof rock to monitor its stress state; The second half of the grating is located in the gangue pile, and the analysis of the gangue compaction process can reflect the overlying rock movement from the side. There are two conditions to reach the peak strain: the working face passes under the working face, and the cantilever beam above the working face is unstable and collapses, and the grating is located at the end of the suspended fixed beam.

(2) The whole process can be divided into four stages: the weak influence area is 150-81 m away from the grating; the weak influence area is 150-81 m away from the grating; 81-49.5 m is the significant influence area; 5-21 m away from the grating is the severe influence area; The distance from the grating is 21-0 m, which is the influence recovery area. Based on this, the abutment pressure distribution model of FBG sensing detection is given. The influence range is within 81 m, the peak value is 21 m in front of the working face, and the pressure relief area is directly above the working face.
(3) Because there are many gaps in the grating, the right gangue will be lifted up after the right gangue is squeezed and sunk, which will cause the decrease of the pressure stress and decrease the absolute value of the measurement strain.

References

[1] QIAN Ming, SHI Pingwu, XU Jialin. Mine pressure and rock formation control [M]. China University of Mining and Technology Press 2010.

[2] Yan L U, Fan S, Zou X. Distributing law of advanced abutment pressure in working face [J]. Journal of Liaoning Technical University, 2008.

[3] LIU Jinhai, JIANG Fuxing, ZHU Sitao. Study of dynamic and static abutment pressure around long-wall face and its application [J]. Chinese journal of rock mechanics and engineering, 2015, 34(9):1815-1827.

[4] SONG Zhenqi, LU Guozhi, XIA Hongchun. A new algorithm for calculating the distribution of face abutment pressure [J]. Journal of Shandong university of science and technology (Natural Science) 2006, 25(1):1-4.

[5] XIE Guangxiang, WANG Lei. Effect of mining thickness on abutment pressure of working face [J]. Journal of china coal society. 2008, 33(4):361-363.

[6] LI Hong, DAI Jing. Preliminary Solution to Calculation of Elastic Foundation Beams Supporting Pressure [J]. Journal of Mining & Safety Engineering 2005, 22(2):4-6.

[7] CHEN Zhonghui, XIE Heping. Damage mechanics analysis on the distribution of abutment pressure around a coal face [J]. Chinese journal of rock mechanics and engineering, 2000, 19(4):436-439.

[8] LIU Jinhai, JIANG Fuxing, FENG Tao. Numerical simulation of abutment pressure distribution of C-shaped stope [J]. Rock and Soil Mechanics, 2010, 31(12):4011-4015.

[9] SI Rongjun, WANG Chunqiu, TAN Yunliang. Numerical simulation of abutment pressure distribution laws of working faces [J]. Rock and Soil Mechanics, 2007, 28(2):351-354.

[10] KURASHIMA T, HORIGUCHI T, et al. Distributed strain measurement using BOTDR improved by taking account of temperature dependence of Brillouin scattering power [C]. ECOC 97, Sep 1997, Conference Publication, 1997.

[11] Du W, CHAI J, Zhang D, et al. The study of water-resistant key strata stability detected by optic fiber sensing in shallow-buried coal seam [J]. International Journal of Rock Mechanics and Mining Sciences, 2021, 141(6):104604.

[12] CHAI Jing, LIU Qi, ZHANG Dingding, et al. Temperature distribution and its evolution characteristics of physical similarity model by optical fiber sensing system

[13] SHI Bin, XU Hongzhong, ZHANG Dan, et al. Feasibility study on application of BOTDR to health monitoring for large infrastructure engineering [J]. Chinese journal of rock mechanics and engineering, 2004, 23(3):493-499.

[14] Du W, CHAI J, Zhang D, et al. Application of Optical Fiber Sensing Technology in Similar Model Test of Shallow-buried and Thick Coal Seam Mining [J]. Measurement, 2021.

[15] HORIGUCHI T, TATEDA M. BOTDA-nondestructive measurement of single-mode optical fiber attenuation characteristics using Brillouin interaction: theory [J]. Journal of Light wave Technology,
[16] Horiguchi T, Shimizu K, Kurashima T, et al. Development of a distributed sensing technique using Brillouin scattering [J]. Journal of Light wave Technology, 1995, 13(7):1296-1302.

[17] Bao X, Dhliwayo J, Heron N, et al. Experimental and theoretical studies on a distributed temperature sensor based on Brillouin scattering [J]. Journal of Light wave Technology, 2002, 13(7):1340-1348.

[18] Bao X, Webb D J, Jackson D A. Combined distributed temperature and strain sensor based on Brillouin loss in an optical fiber [J]. Optics Letters, 1994, 19(2):141.

[19] Parker T R, Farhadiroushan M, Handerek V A, et al. A fully distributed simultaneous strain and temperature sensor using spontaneous Brillouin backscatter [J]. IEEE Photonics Technology Letters, 1997, 9(7):979-981.

[20] Peled, Yair, Yaron, Lior, Motil, Avi, et al. Distributed and dynamic monitoring of 4km/sec waves using a Brillouin fiber optic strain sensor[J]. Proceedings of SPIE - The International Society for Optical Engineering, 2013, 8794(3):34.

[21] Hong C, Zhang Y, Li G, Zhang M, Liu Z. Recent progress of using Brillouin distributed fiber sensor for geotechnical health monitoring. Sens Actuators-phys. 2017,285:131-145

[22] Iten M, Puzrin A M. Landslide monitoring using a road-embedded optical fiber sensor[J]. Proceedings of SPIE - The International Society for Optical Engineering, 2008, 6933.

[23] Moffat R, Sotomayor J, Beltrán J F. Estimating tunnel wall displacements using a simple sensor based on a Brillouin optical time domain reflectometer apparatus[J]. International Journal of Rock Mechanics & Mining Sciences, 2015, 75:233-243.

[24] Jung E J, Kim C S, Jeong M Y, et al. Characterization of FBG sensor interrogation based on a FDML wavelength swept laser.[J]. Optics Express, 2008, 16(21):16552.

[25] Li W, Bao X, Li Y, et al. Differential pulse-width pair BOTDA for high spatial resolution sensing[J]. Optics Express, 2008, 16(26):21616-21625.