Experimental Investigation on the Influence of Temperature on Coal and Gas Outbursts

Xiaoqi Wang, Xiaohan Qi, Heng Ma, and Shengnan Li

1. Introduction

An instantaneous outburst is the violent projection of coal and gas away from freshly exposed coal during underground mining. Coal and gas outbursts are a dynamic phenomenon whereby the original equilibrium state of the coal is broken after the mechanical system of the coal rock is disturbed, and the accumulated elastic energy in the system is quickly and violently released after the strength limit is exceeded [1,2]. Nowadays, such outbursts have been observed in a number of coal basins, in salt, and during metalliferous mining activities in many countries in the world, and they are being observed more and more frequently [3–5]. When an outburst occurs, the rock/coal/gas system transforms from a stable to an unstable state with the release of a significant volume of gas over the duration of the outburst [6–10]. The occurrence of this phenomenon often leads to the destruction of tunnels or even human casualties [11–13]. Domestic and overseas studies have reported that the burst tendency is an inherent attribute of the rock and an internal factor in the occurrence of rock bursts [14]. Any change in the external environment evidently...
influences the burst tendency of coal, and the factors influencing the burst tendency have been extensively investigated [15].

Although China is rich in coal resources, many coal-related disasters have occurred, and some mines have seen two or more composite disasters, such as spontaneous combustion and impact ground pressure, gas outbursts, and spontaneous combustion [16]. These disasters not only cause a loss of resources but also cause problems that require further study. In mines prone to a composite disaster involving gas protrusion and spontaneous combustion, the pressure relief and extraction of the coal seam can help achieve stability; however, cracks develop in the coal seam, and the intensity of air leakage increases, which in turn oxidizes the coal body and induces spontaneous combustion of the coal. After spontaneous combustion, the coal body is damaged, and this type of working condition is commonly observed [17].

For example, as shown in Figure 1a–c, the remining protective layer can relieve the pressure, further develop fractures, and realize the local destabilization of the protected layer. However, oxygen enters the protective layer through the fractures and oxidizes the coal body, and the spontaneous combustion of the coal is further induced [18]. The floor gas extraction alley is used to extract and eliminate the breakthrough of the upper coal seam, resulting in air leakage in the upper coal seam and the gradual oxidation of the coal. With the long-term exposure of the coal column in the mine, the coal body comes in full contact with oxygen, which may induce spontaneous combustion of the coal [19]. In addition, the coal mining face is unsealed after the fire, and an open-pit coal mine is exposed to hidden fire areas, all of which can lead to oxidation or local spontaneous combustion of the coal. After the oxidation area is revealed, the deformation characteristics of the roadway and the appearance characteristics of the mine pressure may be different from those observed in an un-oxidized coal seam. In engineering design and research, the mechanical differences in oxidized coal bodies are often ignored, which may cause deviations between the design or research results and the actual engineering situation. Hence, studying the mechanical properties of high-temperature oxidized coal has become the basis and premise of engineering calculations and design. However, the mechanical properties and bursting liability of thermally damaged coal have rarely been investigated.

Scholars at home and abroad have conducted a series of studies on the influence of temperature on coal and rock damage, and many results have been achieved. Pan et al. [18] studied changes in the mechanical properties of coal after oxidation and proposed the change laws of the mechanical parameters of different oxidized coals. Zheng et al. [20] studied the effect of different temperatures on the mechanical properties of coal under axial compression. Xiao et al. [21] conducted cyclic loading tests under seepage and stress conditions on sandstone treated at different temperatures (25–800 °C). Wang et al. [22] jointly characterized the evolution and development of the pores and fracture structure of coal before and after inducing thermal shock, qualitatively and quantitatively analyzed the change in the width of cracks in the coal seam, and statistically analyzed the changes in the specific surface area and size of the pores before and after thermal shock. Li [23] studied the microscopic pores in lignite at different temperatures using scanning electron microscopy and fine micro-CT technology and analyzed the evolution laws of the pores and fissures in lignite subjected to pyrolysis from 2D and 3D aspects. Qi et al. [24] quantitatively described rock damage and analyzed the correlation between coal damage and mechanical properties. Zhang et al. [25] proposed the concept of a thermal damage coefficient. Hu et al. [26] pioneered the use of microwave radiation to crack coal. It provides a theoretical basis for effectively changing the physical and mechanical properties of a rock mass, thus preventing rock bursts.
Thermal damage to the coal body [18].

Figure 1. Thermal damage to the coal body [18].
Impact propensity refers to the sum of various physical and mechanical properties of coal rock when it can accumulate elastic energy and suddenly release it after exceeding its own strength. Bursting liability is an inherent property of coal rock, which is an internal cause and a necessary condition for the occurrence of coal rock dynamic shock. In recent years, scholars at home and abroad have done a lot of work on the bursting liability of coal rock. Zhang et al. [27] identified the impact tendency of coal with different moisture contents. Wang et al. [28] analyzed the influence law of gas on the shock propensity indicators of coal. Zhang [29] studied the effect of gas on the impact characteristics of coal seams and concluded that this effect should be fully considered in the process of measuring the impact tendency and impact risk assessment of gas-containing coal seams. Song et al. [30] investigated the correlation between gas pressure and energy dissipation after coal reaches its peak strength, and found that gas pressure is negatively correlated with the impact propensity index of coal. Du [31] found that conventional evaluation indicators cannot be used to accurately judge the degree of disaster risk, given the frequent and sudden nature of such disasters.

However, the influence of temperature on the tendency of coal rock bursts has rarely been reported. The study of the mechanism of coal bursting is crucial to the determination of liability, early warning, and the prevention of accidents. Based on the aforementioned studies, this paper employed a microcomputer-controlled electro-hydraulic servo universal material testing machine and a supporting high-temperature thermostat to investigate the influence of thermal damage at different temperatures on different burst tendency indicators, such as coal rock strength, dynamic failure time, and impact energy. The findings of this study can provide a reference for evaluating the burst tendency of coal seams after thermal damage and can have a certain guiding significance in preventing dynamic disasters.

This paper researches the influence of coal treated at different temperatures on coal and gas outbursts during mining. Section 2 studies the bursting liability of coal by introducing multiple bursting liability indicators, such as the energy index of coal and gas outbursts, which provides a certain basis for studying the influence of different temperatures on coal bursting liability. Section 3 uses a servo universal testing machine and a high-temperature constant temperature incubator to carry out mesoscopic damage and mechanical characteristics of the coal body after different temperature treatments. Section 4 uses the box dimension as the fractal descriptor and combines the damage factor to quantitatively describe the thermal damage, and explores the correlation between the thermal damage of coal and the change of mechanical properties after different temperatures. Section 5 is calculated by the full stress–strain curve of the coal sample combined with the classification indicators in the second part. Based on the calculation results of each indicator, the coal treated at different temperatures is discussed and compared. Section 6 concludes the paper.

2. Classification Index of Coal and Gas Outbursts

Bursting liability is an inherent property of coal and rock, which can be expressed by the impact energy index. At present, the judgment of coal seam bursting liability by a single index is very simple and not comprehensive. Therefore, scholars also mostly judge bursting liability from many aspects, such as energy, time, deformation, etc. By establishing multi-faceted indicators, we can gain a comprehensive understanding of the bursting liability of coal bodies.

Many indices can be used to determine the burst tendency of coal, and mature indices include uniaxial compressive strength, dynamic failure time, elastic energy, and impact energy [32]. Related studies have also proposed various comprehensive evaluation indices to address the shortcomings of any single index, in that it may not fully reflect the burst tendency of the coal rock. Examples include the effective impact energy index [33], residual energy index [34], elastic deformation index [35], stiffness ratio index [36], brittleness coefficient [37], surplus energy index, change rate index [38], and impact energy velocity index [39].
This paper provides a clear basis for judging the bursting liability through the following five types of indexes, including intensity index, time indicators, energy indicators, deformation index, and comprehensive indicators. Therefore, the evaluation results are more objective and accurate. These are explained in detail below:

(1) Strength index

The uniaxial compressive strength ($R_c$) of coal is a parameter often used in coal mine engineering, and because it is closely related to the impact trend, it can be used as an indicator to evaluate the bursting liability of the coal seam. The greater the uniaxial compressive strength, the stronger the bursting liability. Specifically, based on this index, coal seams can be divided into no-impact-tendency coal seams, when $R_c < 7$ MPa; medium-impact-propensity coal seams, when $7 \leq R_c < 14$ MPa; and strong-impact-tendency coal seams, when $R_c \geq 14$ MPa.

(2) Time indicators

The dynamic failure time of coal $t_D$ refers to the time required by the coal to transition from ultimate strength to complete failure under conventional uniaxial compression test conditions. Figure 2 shows the acquisition of typical dynamic destruction times. For no-impact-tendency coal seams, $t_D > 500$ ms; moderate-shock-prone coal seams, $50$ ms $< t_D \leq 500$ ms; and strong-impact-tendency coal seams, $t_D < 50$ ms.

![Figure 2. Load-time curve of coal rock specimens.](image)

(3) Energy indicators

As expressed in Equation (1), the elastic performance index $W_{ET}$ is defined as the ratio of the elastic energy ($A_e$) to the unloading energy ($A_{ex}$) at 80–90% of the strength before the peak, and this index is calculated as follows:

$$W_{ET} = \frac{A_e}{A_{ex}} = \frac{1}{2} \sigma_c \Delta \varepsilon_e = \frac{\int_{0}^{t_e} \sigma(\varepsilon) d\varepsilon - \frac{1}{2} \sigma_c \Delta \varepsilon_e}{A_{ex}}$$

Here, $\sigma_c$ is the stress at 80–90% of the strength before the peak, MPa; $\Delta \varepsilon_e$ is the elastic strain variable corresponding to coal accumulation at 80–90% of the strength before the peak, %. Based on the $W_{ET}$ value, coal seams can be divided into strong-impact-tendency coal seams, when $W_{ET} \geq 5.0$; shock-prone coal seams, when $2.0 \leq W_{ET} < 5.0$, and no-shock-propensity coal seams, when $W_{ET} < 2.0$. 
The impact energy index $K_E$ is the ratio of the deformation energy $(A_s)$ accumulated before the peak from the full stress–strain curve of the coal under uniaxial compression conditions to the deformation energy $(A_x)$ lost after the peak, as expressed in Equation (2):

$$K_E = \frac{A_s}{A_x} = \frac{\int_0^{\varepsilon_{\text{max}}} \sigma(\varepsilon) \, d\varepsilon}{\int_{\varepsilon_{\text{max}}}^{\varepsilon_{\text{max}}} \sigma(\varepsilon) \, d\varepsilon}$$

In Figure 3, $K_E = S_{O\text{DFE}}/S_{\text{FDHG}}$; generally, for strong-impact-tendency coal seams, $K_E \geq 2.0$; for medium-shock-tendency coal seams, $1.0 \leq K_E < 2.0$; and for no-shock-tendency coal seams, $K_E < 1.0$.

![Figure 3. Total stress–strain curve.](image)

(4) Deformation index

The elastic deformation index $K_i$ is the ratio of the elastic deformation obtained under repeated loading and unloading cycles to the total deformation when the coal is loaded to 80–90% of its peak strength, as expressed in Equation (3).

$$K_i = \varepsilon_{\text{ei}} / \varepsilon_i$$

In the formula, $\varepsilon_{\text{ei}}$ is the elastic deformation of the coal after the $i$th cycle, and $\varepsilon_i$ is the total deformation of the coal after the $i$th cycle.

The stiffness ratio index $K_{fb}$ is defined as the ratio of the stiffness $K_i$ and $|K_b|$ before and after the extreme strength point in the $\sigma – \varepsilon$ curve of the entire process of coal, as expressed in Equation (4).

$$K_{fb} = K_i / |K_b|$$

The stiffness $K_i$ and $K_b$ before and after the peak strength, can be characterized by Equation (5).

$$K_i = \frac{\sigma_i}{\varepsilon_i}, \quad K_b = \frac{\sigma_b}{\varepsilon_b}$$

Here, $\sigma_i$ and $\sigma_b$ are the axial stress values of the coal before and after the peak, MPa; $\varepsilon_i$ and $\varepsilon_b$ are the axial strain values of the coal before and after peak, %, respectively. From the stiffness ratio in Equation (6), we have:

$$K_{fb} = \frac{\sigma_b \varepsilon_i}{\sigma_i \varepsilon_b}$$
On the $\sigma - \epsilon$ curve of the coal, the tangent slope before and after the peak can be used to characterize the magnitude of the stiffness. The coal will have a bursting liability when $K_{fb} \leq 1.0$ and no bursting liability when $K_{fb} > 1.0$.

The brittleness coefficient $B$ is defined as the product between the ratio of the compressive strength to the tensile strength of the coal and the value of the strain ratio before and after the peak, as expressed in Equation (7).

$$B = \alpha \frac{R_c}{R_t} \frac{\epsilon_f}{\epsilon_b}$$

where $R_t$ is the tensile strength of the coal, MPa; $\epsilon_f$ and $\epsilon_b$ are the strain values before and after the peak, %, respectively; $\alpha$ is the adjustment parameter to make the order of the magnitude of $B$ comparable to the other indicators. The higher the value of $B$, the greater the bursting liability of the coal. Specifically, based on the $B$ value, coal seams can be divided into strong-impact-tendency coal seams, when $B \geq 5.0$; weak-to-medium-impact-tendency coal seams, when $3.0 < B < 5.0$; and no-impact-tendency coal seams, when $B \leq 3.0$.

(5) Comprehensive indicators

The effective impact energy index $K_{efx}$ combines the elastic energy index $W_{ET}$ and impact energy index $K_E$, considering the ratio of the elastic energy stored before the peak ($A_{ef}$) to the dissipated energy ($A_x$) after the peak, as expressed in Equation (8).

$$K_{efx} = \frac{A_{ef}}{A_x} = \frac{1}{f_{\epsilon_{max}}} \sigma_1 \Delta \epsilon_{ef} \int_{\epsilon_{max}}^{\epsilon_n} \sigma(\epsilon) d\epsilon$$

Here, $\sigma_1$ is the peak stress, MPa; $\Delta \epsilon_{ef}$ is the elastic strain variable of coal accumulation before the peak, %. $\epsilon_{max}$ and $\epsilon_n$ are the peak strain and the response value when the coal is completely destroyed, %, respectively. The ratio of the area $S_{FBD}$ under the peak unloading line FB to the area $S_{FDHG}$ under the post-peak curve can be calculated, as shown in Figure 3, by $K_{efx} = S_{FBD} / S_{FDHG}$.

The surplus energy index $W_R$ is the ratio of the elastic strain energy $A_{ef}$ accumulated before the ultimate strength of the coal and to the total failure dissipative energy $A_x$ (i.e., the surplus energy $W_s$), which can be calculated using $W_R = (S_{FBD} - S_{FDHG}) / S_{FDHG}$ in Figure 3.

The rapid release of energy is why the impact pressure leads to a disaster. Therefore, in the evaluation of the bursting liability of coal, in addition to the energy factors, the release rate of the energy, that is, the time effect, should be considered.

The change rate $K$ of the residual energy index is a comprehensive evaluation index that combines the surplus energy, post-peak failure dissipative energy, and dynamic failure time. It can characterize not only the relative magnitude between the surplus energy and failure dissipation energy of the coal but also the dynamic failure time. The rate of change in the surplus energy index $K$ is the ratio of the surplus energy divided by the elastic strain energy and the dynamic failure time, as expressed in Equation (9).

$$K = \frac{W_s}{A_x t_D}$$

where $W_s$ is the surplus energy, J; $t_D$ is the dynamic destruction time, ms; $A_x$ is the deformation energy consumed in the post-peak failure process, J. The greater its value, the greater the relative value of the energy released per unit time during the destruction process, and the stronger the bursting liability.

The impact energy velocity index $W_{st}$ is a new comprehensive discriminant index of bursting liability and can be defined by considering three factors: the elastic energy storage $A_t$, loss energy $A_x$, and dynamic failure time $t_D$ during coal rock failure. Its physical meaning is the ratio of accumulated and absorbed energy per unit of time during the
compression process of the coal, which indicates the impact ability of the failure energy during the uniaxial compression process, as expressed in Equation (10).

\[ W_{st} = \frac{K_E}{t_D} \]  

where \( W_{st} \) is the impact energy velocity index, \( K_E \) is the impact energy index, and \( t_D \) is the dynamic destruction time. The higher the impact energy velocity index, the stronger the impact ability.

Some indexes have dual attributes of correlation and independence, reflecting the impact characteristics of coal from different perspectives. Since the comprehensive evaluation index covers some single indicators and considers the convenience of the test method, in the analysis presented below in this paper, six indicators are mainly used: uniaxial compressive strength index, dynamic failure time, stiffness ratio index, effective impact energy index, residual energy index change rate, and impact energy velocity. How to use these indicators to judge the bursting liability of coal, which requires mechanical tests on coal, and these bursting liability indicators are calculated through the full stress–strain curve obtained by the test.

3. Test Equipment and Methods

The experimental process mainly includes the preparation of standard coal samples, the coal samples required for screening experiments, and the mechanical testing and damage detection of coal samples.

3.1. Specimen Preparation

The coal was sampled from the 24,130 working face at a mining depth of 1073 m in the Pingding Mountain coal mine. This is shown in Figure 4. The processed coal had dimensions of \( \phi \times L \) and met the requirements of the international processing standards. To reduce the influence of the discreteness of the test results, the coal was screened using the HC-U7 non-metallic ultrasonic detector, as shown in Figure 5.

The results of the coal industry analysis showed that the coal was fat coal, the moisture \( M_{ad} \) mass fraction was 1.37%, the ash \( A_{ad} \) mass fraction was 18.86%, the volatile content \( V_{ad} \) mass fraction was 23.27%, the fixed-carbon \( F_{ad} \) mass fraction was 43.09%, the calcite mass fraction was 2.31%, the pyrite mass fraction was 0.60%, and the clay ore mass fraction was 10.50%. The results of the physical properties of the coal were as follows. The porosity \( n_0 \), permeability \( k_0 \), density \( \rho_0 \), cohesion \( \phi_0 \), and inner friction angle \( \alpha \) were 0.068, \( 2.96 \times 10^{-17} \) m², 1.55 g/cm³, 1.98 MPa, and 35.84°, respectively.

Since the ignition point of bituminous coal (the coal in this study was mainly fat coal, which belongs to bituminous coal) is in the range of 320–380 °C, to prevent the coal from being heated and burned, the maximum thermal damage temperature was set to 300 °C. The heating range of the experimental design was divided into six temperature grades in the 50–300 °C interval.

3.2. Test Protocol

The coal samples were processed on a self-built oxidation experiment platform. Figure 6 shows the test flow. The specific steps of the test are as follows:

(1) The retrieved coal block was cored using an electric drilling core machine, and the coal was processed into standard specimens for testing using a cutting and smoothing-integrated machine and a double-end face grinding machine.

(2) Since the coal block in the sampling process was in a water-drenched state, the coal needed to be dried in a drying box (maintained at a constant temperature of 50 °C) for 6 h. The resulting coal samples were denoted by MY1–MY53. To reduce the discreteness of the test results, the coal was screened using an HC-U7 nonmetallic ultrasonic detector. The screened coal samples were grouped such that there were no fewer than three in each group.
(3) The screened specimens were packed in a sealed plastic bag and placed in the same environment for later use.

(4) Thermal damage to the coal (50–300 °C) was induced using an electric blower drying oven, where the coal was heated to the target temperature at a rate of 1 °C/s. The coal used in the test was placed in a drying box for thermal damage for 2 h, and the specimen was removed and placed in a well-ventilated position in the laboratory until it cooled to room temperature.

(5) When the specimen reached room temperature, its mass and volume were measured, and its wave velocity was tested using an HC-U7 ultrasonic detector. Before the ultrasonic wave velocity test, the ultrasonic sensor was coupled with butter at both ends of the coal sample, tested individually, and recorded.

(6) After thermal damage, the six groups of coal specimens were subjected to uniaxial compression tests at a loading rate of 0.001 mm/s until they lost their bearing capacity. The stress, strain, time, and other related parameters were recorded until the coal was damaged, and the full stress–strain curve was plotted. The transverse and longitudinal strains of the coal were obtained using a stress–strain gauge to determine the Poisson's ratio of the coal at different temperatures.

(7) A control specimen of the coal was selected. Optical measurement and imaging of the meso-crack state were conducted using a 4 K scientific research camera by inducing thermal damage on the control coal specimen, so as to obtain local fracture images, which were finally stitched to plot a microscopic crack map of the end face of the coal. This helped study the crack evolution law of thermally damaged coal. Subsequently, the complex fracture network was quantitatively described by performing a crack analysis on the stitched full-fracture image with the help of Image J software.
Processes 2023, 11, x FOR PEER REVIEW

Figure 5. The detection principle of non-metallic ultrasonic detector [40].

Figure 6. Experimental process.

4. Analysis of Test Results

In the third section, by taking the coal of the 24,130 working face of Ping-mei Ten Mine as the research object, using HC-U7 non-metallic ultrasonic detector, electric blower drying box, 4 K scientific research camera, servo universal testing machine, and other instruments, the loading test of heat-damaged coal at different temperatures was carried out, aiming to study the influence of thermal damage on mechanical parameters, such as uniaxial compressive strength, elastic modulus, and Poisson’s ratio of coal body, and analyze the meso-structure and macroscopic failure state of thermally damaged coal.

4.1. Load Stress—Strain Curve of Thermally Damaged Coal

Figure 7 shows the full σ–ε curves of different thermally damaged coal samples obtained experimentally. The total stress–strain curve contains rich energy-related infor-
mation, reflecting the conversion, accumulation, and dissipation of the energy in the coal during loading. Figure 7 shows the curve obtained under the action of thermal damage at different temperatures in the laboratory. The uniaxial compressive $\sigma$–$\varepsilon$ curves of the coal rock generally pass through the following five stages: Primary pore fracture compaction stage, linear elasticity stage, plastic weakening stage, weak surface damage failure, and comprehensive failure stage. Multiple sets of bursting liability indices could be obtained by calculating the total stress–strain curves of the coal, and a comparative analysis was conducted. The energy index and mechanical parameters were obtained from the total $\sigma$–$\varepsilon$ curve of the coal.

Figure 7. Thermal damage coal $\sigma$–$\varepsilon$ curve.
4.2. Analysis of the Mechanical Properties of Thermally Damaged Coal from a Mesoscopic Perspective

Figure 8 shows the interaction of thermophysical parameters with temperature during thermodynamic coupling. The problem of thermal damage to the coal body is mainly caused by thermal stress caused by changes in coal temperature. The change in physical and mechanical properties of coal bodies after high temperature thermal damage is essentially the evolution process of microcracks from scratch, from germination to penetration. Based on experimental research, the evolution of the mechanical properties of coal rock mass under thermodynamic coupling is studied in order to reveal the deformation and failure mechanism of thermally damaged coal rock, which is of great significance for the stability analysis of deep underground coal rock mass under temperature.

![Figure 8. Process of coal and rock damage under thermomechanical coupling](image-url)
4.2.1. Analysis of the Influence of Thermal Damage on Coal Mechanical Characteristics and Parameters

As shown in Figure 9a, the 50–200 °C interval $E$ tends to be stable. After 200 °C, the average modulus of elasticity decreased from 3.49 GPa to 0.96 GPa, a decrease of 72.49%, and the decline rate of $E$ increased.

![Graphs showing elastic modulus, Poisson's ratio, and compressive strength vs. temperature](image)

(a) Elastic modulus of coal after thermal shock of different temperature  (b) Poisson’s ratio of coal after thermal shock of different temperature  (c) Uniaxial compressive strength of coal after thermal shock of different temperature

**Figure 9.** Changes in the mechanical parameters of coal rock with temperature.

Owing to the change in the state of the 200 °C thermally damaged coal from elastic-plastic to brittleness, $E$ was improved. Once the thermal damage temperature crossed 200 °C, the coal significantly lost its cohesion, the stress was significantly reduced, the strain variable increased, brittle-to-ductility transition occurred, and $E$ was significantly reduced.

The coal was affected by thermal damage, and the internal dehydration, pyrolysis, and volatilization of the organic matter and minerals were analyzed [42–44]. During thermal damage, the main components of the desorption gas released from the coal were $\text{H}_2\text{O}$, $\text{CH}_4$, $\text{CO}_2$, $\text{C}_2\text{H}_6$, $\text{N}_2$, and $\text{O}_2$. A high-precision electronic balance was used to record the change in mass $M$ after thermal damage and cooling, and the mass $m$ of the emission gas was calculated, as shown in Figure 10.

In the temperature range of 50–200 °C, Figure 10 shows that the increase rate of water and gas emissions was low, mainly for water removal and a small amount of attached gas removal. Above 200 °C, the emissions were exacerbated, and the concentrations of gases, such as $\text{CH}_4$ and $\text{CO}_2$, increased significantly, indicating the generation of a large amount of organic matter and mineral pyrolysis. At 300 °C, the coal body accumulated 28.99 g of mass during its internal dehydration, pyrolysis, and volatilization processes, accounting for 11.75% of the total mass of the coal.
Coal is affected by long-term geological activities. Heating and cooling produce heterogeneous deformation and heterogeneous stress, resulting in thermal cracking [45]. To model this phenomenon, two microbodies in coal were analyzed and assumed to be constrained. The corresponding mechanical models can be described as follows: (1) Two different particles are close to each other, where $\lambda_1$, $\lambda_2$, $E_1$, and $E_2$ represent the expansion coefficient and elastic modulus of the two components, respectively; (2) The temperature is increased from room temperature to $\Delta \theta$. The thermal stress [46] is calculated as follows:

$$\Delta \sigma = (\lambda_1 + \lambda_2) \Delta \theta \frac{E_1 E_2}{E_1 - E_2}$$

(11)

when $\Delta \sigma$ exceeds the ultimate strength [$\sigma$], the coal will break. $\Delta \theta$ determines $\Delta \sigma$, and after setting a predetermined temperature inside the electric blower drying box, the coal was placed inside the box for thermal damage.

During the damage process of the coal, there will be a temperature difference between the high temperature and the internal temperature of the coal, i.e., $\Delta \theta$. The existence of this temperature difference determines the degree of internal rupture of the coal body; that is, compressive and tensile stresses will occur between the internal granular coal bodies during the thermal damage process at different temperatures, resulting in evident changes in the mechanical properties of the coal (such as the moisture content, deformation ability, and other parameters), resulting in thermally induced fracture. After the thermal damage treatment of the coal, the porosity and moisture content of the coal, the internal structure of the coal body, coal density, and the original micro-fractures and pores inside the coal changed compared with that before the treatment [47,48].

As shown in Figure 9b, in the temperature range of 50–200 °C, the $\sigma$ decline rate was low, and the decrease range was 11.63%. After 200 °C, the average compressive strength decreased from 7.60 MPa to 3.60 MPa, a decrease of 52.63%, and the $\sigma$ decline rate increased. The thermal damage considerably reduced the pore structure of the coal and the cementation degree between the coal particles, and a decrease in the cohesion, softening coefficient, and deformation parameters of the coal was noted.

As shown in Figure 9c, in the temperature range of 50–200 °C, the overall change in the slope was small, and the growth range was 18.18%. This may be because, with the increase in temperature, the thermal expansion coefficient of the particles inside the coal varied, resulting in a large circumferential deformation of the coal. Because the coal is bound by axial force, the longitudinal deformation of the coal changes little. In the temperature range of 200–300 °C, $A$ decreased rapidly with the increase in temperature, and the average Poisson’s ratio decreased from 0.31 to 0.25, a decrease of 18.59%. This is because large amounts of organic matter and minerals in the coal were pyrolyzed, the axial bearing capacity was reduced, and the axial strain variable per unit of time was greater than the ring response variable.

![Figure 10. Change in coal rock mass with temperature after thermal damage.](image-url)
4.2.2. Quantitative Analysis of Mesoscopic Thermal Damage Images of Coal Rock

Under the action of temperature, the internal organic matter in coal is pyrolyzed, and the release of pyrolysis products significantly changes the internal structure of the coal, forming a large number of pore fractures [49]. At the same time, because coal is a heterogeneous body, the thermal expansion coefficient of each block is different under high-temperature conditions, and there is compression of the large-deformation areas and tension of the small-deformation areas, forming structural thermal stress in the coal rock; thus, the connections between the heterogeneous blocks are fractured, resulting in microfractures.

Due to its limited field of view, an optical microscope can only perform detection in a narrow range, and its practicality is poor. The best method is to move the microscope sequentially along the observation surface to capture consecutive images and then stitch them sequentially. To realize the above method, combined with existing related equipment and technology, the optical microscope was modified to be placed on a motorized platform. The microscope itself can adjust the focus perpendicular to the observation surface, and the 3D movement of the microscope can be realized through the cooperation of the motorized stage and the optical microscope. Before the test, the scanning range and scanning method were set through a computer. In this method, the microscope automatically scans, captures, stores, and stitches images in real time after the test starts. The principle of image stitching is that the obtained image has a certain overlap with the adjacent image. The image processing software can determine the relative position relationship between the images based on similar characteristics of the overlapping part, so as to realize image stitching. After stitching, the image is then distorted, stitched, and fused, and the overlapping parts and gaps in the image are disposed of, thus obtaining a complete and clear image of the observation surface.

Figure 11 shows a mesoscopic thermal damage image of a coal sample captured using a 4K scientific measurement camera after applying different temperature shocks to its end face. The development width of the fracture was measured. Taking the fracture in the box shown in Figure 12 as an example, the fracture width $L$ changes with the thermal damage temperature.
4.2.3. Coal Rock Damage Factors and Meso-Fractal Description

For coal rock materials, the damage factor $S$ can be defined using the elastic strain method, and for fully elastic materials, the stress–strain satisfies the following formula:

$$\sigma = E\varepsilon_e$$  \hspace{1cm} (12)

where $\varepsilon_e$ is the coal rock strain, $\sigma$ is the coal rock stress, $E$ is Elastic modulus.

The damage factor is expressed in Equation (13),

$$S = 1 - \left(\frac{V_f}{V_p}\right)^2$$  \hspace{1cm} (13)

**Figure 12.** Change in primary crack width after thermal damage.
Here, \( V_p \) and \( V_l \) are the velocities of the longitudinal waves propagating in the rock after and before damage, m/s, respectively.

It is limited to describing the development of fractures with fracture width and density, which cannot accurately depict the evolution characteristics of fractures. The calculation of the fractal descriptors can be binarized using the pixel overlay method to binvize the end face diagram of the coal after different temperature shocks, as shown in Figure 13. The grid is gradually refined to check the number of coverages that need to be changed so that the number of box dimensions can be calculated [50].

As expressed in Equation (14), when the side length of the grid is \( r \), the total space is divided into \( N \) grids. The meshing and fracture calculation methods of the fracture surface of the fracked coal body are shown in Figure 13. With this, the box dimension can be expressed as follows:

\[
D_B = \frac{\ln N(r)}{\ln (\frac{r}{e})} \tag{14}
\]

Here, \( D_B \) is the box dimension, and \( r \) is the dividing scale. \( D_B \) embodies the efficiency of covering the entire area with small boxes of the same shape.

Fracture density is an important parameter for evaluating the number and distribution of fractures on the surface of the coal body. The fracture density \( \rho \) can also be calculated from the cross-sectional area \( S \) at the total fracture length of the section:

\[
\rho = \frac{\sum_{i=1}^{n} L_i}{S} \tag{15}
\]

where \( L_i \) is the length of the \( i \) fracture, and \( n \) is the total number of fractures.

As shown in Figure 14, the fracture evolution formed by the thermal rupture at the end face of the coal rock mass can be divided into three stages: the primary crack propagation stage, the new single macroscopic fracture development stage, and the fracture network connectivity stage. Under the action of thermal damage, thermal cracking occurs inside the coal rock mass, and the original cracks first expand and then produce new microcracks. With the intensification of thermal cracking, small cracks gradually extend and penetrate, forming macroscopic cracks. A large number of propagating cracks coalesce to form a
complex fracture network. The development process of the three types of fracture structures reflects the basic form of the thermal fracture evolution of coal rock.

![Full-section fracture network](image)

**Figure 14.** Mosaic diagram of the crack network on the full face after thermal damage.

The ultrasonic test results showed that the damage factors of the coal rock under thermal damage at different temperatures can be obtained using Equation (13), combined with the full-section fracture network $D_B$, as shown in Figure 15. The relationship between $S$ and intensity $R_c$ is plotted in Figure 15.

![Graph](image)

**Figure 15.** Correlation between quantitative depiction factors and intensity of thermal damage.

Figure 15 shows the $S$ change characteristic after experiencing high-temperature shock. The ultrasonic wave velocity inside the coal rock decreases with the increase in the temperature: the higher the thermal damage temperature, the greater the decrease in the ultrasonic velocity and the greater the $S$ value. It only slightly reduces at 100 °C; this is because the coal contains argillaceous components at this temperature, with expansion, extrusion, and filling of the pores of the specimen. The variation characteristics of the end-
face fracture network $D_B$ are most correlated with $R_c$. When the end fracture $D_B$ is between 50 and 200 °C, the growth is slow, and when it exceeds 200 °C, the $D_B$ value increases rapidly, consistent with the change law of $E$, indicating that the threshold temperature of this batch of coal is 200 °C. Since the tests were conducted after the coal rock was cooled after thermal damage, the damage to the coal rock after impact cooling was irreversible.

As shown in Figure 16, with the increase in temperature, the compressive strength of coal $\sigma$, ultrasonic velocity $A$, and temperature $T$ had a negative correlation. The increase in the number of cracks led to an increase in the resistance of the elastic waves. With the increase in the thermal damage temperature, the change in the microstructure significantly impacted the mechanical properties of the coal, resulting in a gradual decrease in its compactness.

![Figure 16](image)

Figure 16. Variation of fracture width, fracture density, wave velocity, mass, and density with temperature.

As shown in Figure 16, the ultrasonic velocity $A$, coal mass $m$, coal density $\rho$, and temperature $T$ had a negative correlation, while the fracture width $L$ and fracture density $\psi$ were positively correlated with temperature $T$. During the thermal damage to the coal, the desorption gases released from the coal included $\text{H}_2\text{O}$, $\text{CH}_4$, $\text{CO}_2$, $\text{C}_2\text{H}_6$, $\text{N}_2$, and $\text{O}_2$. In the temperature range of 50–200 °C, there was water removal and a small amount of adherent gas discharge, resulting in the release and connection of the pore space occupied by the two, and the coal matrix became soft under the action of temperature, resulting in extrusion deformation between the coal particles. In this stage, the porosity of the thermally damaged coal increased gradually, the meso-structural complexity of the coal body increased, the amplitudes of $L$ and $\psi$ were flat, $L$ increased from 0.02 mm to 0.038 mm, and $\psi$ increased from 12/mm$^2$ to 34/mm$^2$. In the temperature range of 200–300 °C, the amount of water discharge decreased, the concentrations of $\text{CH}_4$ and $\text{CO}_2$ gradually increased, large amounts of organic matter and minerals in the coal were pyrolyzed, and large amounts of coal moisture and pyrolysis gases were lost, resulting in a decrease in coal quality. Moreover, $L$ increased from 0.038 mm to 0.144 mm, $\psi$ increased from 34 pieces/mm$^2$ to 80 pieces/mm$^2$, $L$ and $\psi$ changed in a wide range, and $L$ and $\psi$ increased by 279% and 135%, respectively. In this stage, the coal body underwent physical and chemical changes. The coal decomposed at high temperatures to produce a large number of gases, which continued to heat and cause the volume to expand sharply. The coal body
produced a high local tensile stress, and there was typical reaming and opening of new holes [51].

In summary, with the gradual increase in the thermal damage temperature, the pores, roar size, and number of roars gradually increased. Thermal damage changes the structure of the coal body by eliminating the cement inside the coal, which manifests as gradual weakening of the cementation strength and cohesion between the internal skeleton structures of the coal. The meso-structural change in the coal body causes the internal defects to increase and result in damage, which expands to the weak structural surface. The degree of damage to the coal becomes increasingly high, which increases the heterogeneity of the coal specimens, increases the spatial distribution density of the cracks in the raw coal, and has significant randomness in terms of the scale and location of the fractures. The heterogeneity causes a deterioration in the macroscopic mechanical proper ties of the coal, thereby significantly decreasing the load-bearing capacity of the coal skeleton.

5. Analysis of the Shock Tendency Index of Coal

All kinds of bursting liability indicators are from the whole stress–strain curve. According to the definition of various indicators in the second section, the corresponding energy can be obtained by integrating each characteristic segment of the curve, and then the value of the bursting liability index at different temperatures can be obtained by simple calculation.

5.1. Effect of Thermal Damage on the Uniaxial Compressive Strength Index

Figure 17 shows that after the coal rock was thermally damaged at 50 °C (hereinafter referred to as the normal temperature) below the ambient temperature, the uniaxial compressive strength increased with the decrease in the temperature, and the coal was categorized as weakly impacted coal at room temperature. After the temperature range of 50–200 °C, the compressive strength of the coal rock showed a slow downward trend with the increase in temperature, whereas the bursting liability of the specimen did not change. The average peak stress of the coal rock decreased rapidly after 200–300 °C, and the average peak stress decreased from 8.13 MPa at 200 °C to 6.40 MPa at 250 °C and 4.0 MPa at 300 °C, with a decrease of 21.28% and 50.80%. Compared with the compressive strength of 9.92 MPa at room temperature, the average decrease in the compressive strength in the 250–300 °C interval was in the range of 35.48–59.68%. In terms of the uniaxial compressive strength index, after the temperature crossed 200 °C, the bursting liability of the coal rock reduced from medium impact to no impact. When the temperature of the coal matrix gradually dropped to laboratory room temperature, in this process, the internal heat energy was gradually dissipated, and the inhomogeneity of the different components in the coal body increased, resulting in a pyrolysis effect, causing more damage.

Figure 18 shows the thermal distribution of the thermally affected area after borehole ignition, where the high-temperature area spreads outward from the hole wall in a spherical shape; close to the nearest position of the hole wall ignition area, the temperature is the highest, exceeding 200 °C; the coal strength in this range is significantly reduced under the action of the surrounding stress, and the local strain softening zone formed by the surrounding rock of the borehole is further expanded. The strength of the peripheral coal, which is less affected by thermal damage, is higher, and the bursting liability remains unchanged. According to the theory of deformation system instability, the impact ground pressure undergoes a dynamic instability process under external disturbance when the medium in the high-stress zone of the rock body exhibits locally formed strain softening, and the medium that has not yet formed strain softening will be in an unstable state. The ignition core area forms a local strain softening region, producing an unstable state with the surrounding coal body that is unaffected by high temperatures, which increases the risk of local coal seam impact.
damage temperature, the change law of the impact energy index first increases, peaks at temperatures. The impact energy index of the coal at room temperature shows that it is in

and after a large number of calculations, the average values of the seven indicators under ing each characteristic segment of the curve from the definitions of the above indicators,

5.2.1. Change Law of Comprehensive Index of the Bursting Liability of Coal Rock

Temperature distribution in the affected area of thermal damage.

Figure 17. Uniaxial compressive strength of coal rock after thermal damage at different temperatures.

Figure 18. Temperature distribution in the affected area of thermal damage.

5.2. Effect of Thermal Damage on Comprehensive Indicators of Coal Shock Tendency
5.2.1. Change Law of Comprehensive Index of the Bursting Liability of Coal Rock

According to Equations (1)–(3), the corresponding energy can be obtained by integrating each characteristic segment of the curve from the definitions of the above indicators, and after a large number of calculations, the average values of the seven indicators under each thermal damage temperature are obtained, as shown in Figure 19. Figure 19 shows the change law curve of the bursting liability index of coal after thermal damage at different temperatures. The impact energy index of the coal at room temperature shows that it is in a state of transition from medium shock to strong impact. With the increase in the thermal damage temperature, the change law of the impact energy index first increases, peaks at
200 °C, and then gradually decreases. The bursting liability is zero at 300 °C. The change law of the impact energy rate is consistent with that of the impact energy index. The change rates of the effective impact energy index and surplus energy index are slightly different from those of the impact energy index, and the specific points are as follows:

1) In the temperature range of 50–150 °C, the impact energy index increased slightly to 1.58 times that at 150 °C. The effective impact energy index, surplus energy index change rate, and impact energy velocity index also increased with the increase in temperature, and their distributions increased to 1.73, 5.42, and 2.87 times, with the coal transitioning from weak bursting liability to medium bursting liability.

2) In the temperature range of 150–200 °C, the impact energy index and the impact energy rate increased with the change in the thermal damage temperature. The impact energy and impact energy velocity indices increased by 71% and 80%, respectively. The effective impact energy index was reduced by 21%, and the rate of change in the surplus energy index was reduced by 30%.

3) In the temperature range of 200–300 °C, all the indicators decreased significantly with an increase in the temperature, with the coal transitioning from medium-bursting liability to no-bursting liability.

4) Based on the stiffness ratio index, the normal temperature coal showed a bursting liability, as shown in Figure 18. In the temperature range of 50–100 °C, the bursting liability of the coal specimens was enhanced. In the temperature range of 100–150 °C, the bursting liability weakened. In the temperature range of 150–300 °C, the bursting liability increased.

Figure 19. Coal rock bursting liability index at different temperatures.

5.2.2. Comparative Analysis of Shock Propensity Indicators

A morphological analysis of coal failure during the test showed that the effective impact energy index, surplus energy index change rate, and impact energy velocity index can better reflect the real bursting liability, and the change laws of the three comprehensive indicators are more consistent with the difference in the thermal damage temperature. However, since the above comprehensive bursting liability indices are the ratio of energies or the ratio of energy to the dynamic failure time, the bursting liability inferred from them is a relative concept, while the uniaxial compressive strength is an absolute index. When determining the impact risk of coal, it is necessary to combine the comprehensive index with the uniaxial compressive strength for a comprehensive evaluation. The change law of the bursting liability of the coal with the temperature, as characterized by the dynamic failure time, is consistent with the impact energy index and the impact energy velocity index.
The stiffness ratio index only considers the ratio of the stiffness before and after the peak, ignores the weakening degree and weakening rate of the stiffness after the peak, and cannot fully grasp the change characteristics before and after the peak. The correlation between this index and the other indicators is poor, and there is a significant error in the judgment of the bursting liability of coal.

Notably, the internal cause of the bursting liability of coal is the occurrence of impact pressure, while the external cause is whether the environmental stress of the coal mass reaches critical stress. This shows that the bursting liability index can only reflect one nature of the coal seam, and whether the impact occurs and the impact intensity also depends on whether the external load reaches the critical value and the absolute value of the accumulated energy in the coal system. Therefore, the identification and discrimination of bursting liability involves dividing the bursting liability degree in the impact pressure area, and the impact risk degree of the impact pressure area should be tested on site.

5.3. Correlation Analysis of Mesoscopic Damage and Energy Response of Thermally Damaged Coal

From the typical failure process curve of the coal shown in Figure 20, it can be seen that the bursting liability of the coal varied, and its fracture instability propagation form after peak strength was also different. The lower the thermal damage temperature of the coal, the greater the intensity of the energy accumulation before the peak and the energy release after the peak, compared with that of the coal with a higher thermal damage temperature, and the intensity is inversely proportional to the degree of thermal damage. Compared with that at 50 °C, after the peak strength of the coal at 50–100 °C, the stress falls in a step manner, and after the first fall, the stress returns straight back to a certain value and then falls again, and quickly loses the ability to resist after three repeated falls, so that the elastic energy accumulated before the peak is released gradually.

![Figure 20. Stress–strain curve of a typical coal.](image)

The stress–strain curve of the coal in the temperature range of 150–200 °C fell in a slope-like manner, and two evident linear drops occurred. In the temperature range of 250–300 °C, the elastic deformation stage was short, and the plastic deformation stage was long during the pre-peak deformation process of the coal. During the loading process, most of the energy inputted by the external system existed in the form of irreversible dissipation energy, the energy dissipation increased, the energy release during destruction decreased, the stress decreased on a gentle slope, and there was no sudden drop phenomenon. The release of coal energy tended to flatten, the dynamic failure time increased significantly, the residual strength was evident, and the failure mode was visco plastic failure.
5.4. Macroscopic Failure State before and after Loading Failure of Thermally Damaged Coal

As shown in Figure 21, when the temperature is in the range of 50–200 °C, the coal is smooth overall, the distribution of the minor cracks is lower, and a small number of semi-penetrating cracks appear on the surface of each part. As the processing temperature decreases, the cracks decrease in number and become more regular.

![Figure 21. Macroscopic failure state of coal after thermal damage at different temperatures.](image)

When the temperature was between 200 and 300 °C, with the increase in the treatment temperature, microscopic cracks appeared in multiple areas in the coal meso-structure due to the external energy; these cracks were approximately parallel to the axial direction. The newly formed small cracks merged with the main cracks inside the coal to form a crack network until they were damaged, causing the coal to pop out. Because there were multiple damages before the coal was stressed, a large number of cracks could be visually seen; when loaded to failure, the cracks in the coal were staggered and penetrated, and there was a higher degree of fragmentation.

When the temperature was 200 °C, the failure mode of the coal was X-shaped conjugate breakage, and when more crushed coal blocks were ejected, there was more damage before the coal was not stressed, and a large number of cracks could be visually seen. After exceeding 200 °C, the coal body was close to the molten state after loading, the coal body was relatively soft and broken, the homogeneity was poor, cutting was developed, the form of deformation failure manifested as tension failure, the coal skeleton structure continued to fail, the coal rupture surfaces increased, the ductility deformation increased, the energy dissipation increased, and the energy release during destruction decreased.

5.5. Discussion

In recent years, some scholars at home and abroad have paid special attention to the fracture development characteristics of high-temperature thermal damage coal rocks and the changes in their petrophysical and mechanical properties caused by them. The macroscopic fracture of coal rock mass depends on the distribution of mesomicrocracks in the coal body. High temperature will lead to the propagation of these microcracks, which in turn affects the macroscopic fracture of the coal body, and it is of great significance to study the physical and mechanical characteristics of coal under high temperature or high temperature treatment for evaluating the surrounding rock failure under the coupling conditions of high temperature and high stress. However, the existing research focuses
on the influence of high temperature thermal damage on the macroscopic mechanical properties of coal rock, and rarely explores the influence of the difference of thermodynamic properties of different minerals on the macro-meso mechanical properties of coal from the mineral level. However, the influence of the mesostructure and joint characteristics of some minerals inside coal on the macroscopic failure mechanism of coal rock cannot be ignored. Therefore, in future research, this paper can be used as a basis to reveal the influence of high temperature thermal damage on the deformation and failure mechanism of coal samples with the help of more advanced instruments and other means, in order to provide useful references for deep coal mining.

6. Conclusions

In this study, uniaxial compression tests were conducted on coal samples obtained from a deep mine, and the burst tendency of the coal was analyzed under thermal impact at different temperatures. The main conclusions of this study are as follows:

(1) Wave velocity $A$, coal mass $m$, coal density $\rho$, and temperature $\theta$ showed a negative correlation. The fracture width $L$ and fracture density $\psi$ were positively correlated with the temperature $\theta$. In the temperature range of 50–200 °C, the complexity of the meso-structure of the coal body increased, and the variation amplitudes of the fracture width $L$ and coal density $\rho$ were flat. In the temperature range of 200–300 °C, the coal body had the most complex meso-structure, the fracture length was large, the distribution range was wide, the fracture shape was complex, many fractures appeared to penetrate each other, and the fracture distribution was relatively dense.

(2) Thermally induced mesoscopic damage of the coal showed a good positive correlation with the mechanical properties and energy response during thermal loading. The higher the thermal damage temperature, the greater the pore development in the coal, and high-stress zones were easily formed around the pores to superimpose each other, which deteriorated the mechanical properties to a greater extent, and the accumulated strain energy was easily consumed through plastic deformation or rupture.

(3) When the temperature was between 50 and 200 °C, in terms of the failure mode, there were multiple splitting fractures along the axial direction. In the temperature range of 200–300 °C, there were more damage before the coal was not stressed, and a large number of cracks could be visually seen. After loading to failure, the coal came close to the molten state. With the thermal damage temperature exceeding 200 °C, the rupture surfaces of the coal increased in number, the ductility deformation increased, the energy dissipation increased, and the energy release during failure decreased.

(4) The comprehensive impact index of the normal temperature coal showed that it was in a transition state from medium shock to strong impact. With the increase in the thermal damage temperature, the impact energy index first increased, reached a peak at 200 °C, and then gradually decreased. The bursting liability decreased to no impact at 300 °C.

(5) Comparing the six shock tendency indicators, the effective impact energy index, surplus energy index change rate, and impact energy velocity index could accurately reflect real bursting liability. With the increase in the thermal damage temperature, the change laws of the three comprehensive indicators were relatively consistent. When determining the impact risk of coal, it is necessary to combine comprehensive indicators with uniaxial compressive strength. The change law of the bursting liability of coal with temperature, as characterized by the dynamic failure time, was consistent with the impact energy index and impact energy velocity index. The stiffness ratio index ignored the degree of stiffness weakening and the weakening rate after the peak, could not fully grasp the change characteristics before and after the peak, and had a weak correlation with the other indicators.

Based on the impact of thermal damage on the burst tendency and the failure form of coal, the effect of thermal damage on the impact characteristics of coal seams should be
considered in the process of measuring the burst tendency and evaluating the impact risk of coal seams that have been drilled and fired.

Author Contributions: X.W.: Methodology, Validation, Formal analysis, Investigation, Data curation, Writing—original draft, Supervision. X.Q.: Conceptualization, Methodology, Resources, Writing—original draft, Supervision, Project administration, Funding acquisition. H.M.: Validation, Resources, Data curation, Supervision. S.L.: Validation, Data curation, Supervision. All authors have read and agreed to the published version of the manuscript.

Funding: This project was funded by the Natural Science Foundation of China (No. 52074148). Xiao-qi Wang is supported by the China Scholarship Council (No. 202208210240).

Data Availability Statement: The datasets generated during and analyzed during the current study are available from the corresponding author on reasonable request.

Conflicts of Interest: We declare no financial or personal relationships with other people or organizations that could unduly influence our work. We have no professional or other personal interests of any nature or kind in any product, service, and/or company that could be construed as influencing the position presented in, or the review of, the manuscript titled, ‘Experimental Investigation on the Influence of Temperature on Coal and Gas Outbursts’.

References
21. Xiao, W.; Yu, G.; Li, H.; Zhan, W.; Zhang, D. Experimental study on the failure process of sandstone subjected to cyclic loading and unloading after high temperature treatment. Eng. Geol. 2021, 293, 106305. [CrossRef]
42. Xu, Y.; Lin, B.; Li, Y. Change Laws of Pore–Fracture Structure of Coal under High-Temperature Steam Shock. ACS Omega 2022, 7, 44298–44309. [CrossRef]

51. Martirosyan, A.V.; Ilyushin, Y.V. The Development of the Toxic and Flammable Gases Concentration Monitoring System for Coalmines. Energies 2022, 15, 8917. [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.