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Mathematical Modeling of Gas Monitoring and Control in the Coal Mine

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Aim To monitor dynamically and control precisely the gas content during coal mining operations as well as to reduce relevant accidents. Method Using mathematical modeling to decide whether a coal mine is high-gas and to judge its safety degree, thus establishing mathematical and improvement model of the targeted coal mine. Results and conclusions By mathematical modeling, the air volume required is obtained by calculation under the condition to ensure maximum security, solving the problem of gas monitoring and control in the coal mine to some extent.

1. Introduction

The issue of safe production in the coal mine has been a hot topic of social concerns in recent years. At present gas has become one of the main incentives for coal mine accidents, so effective monitoring and control is an important way to ensure coal mine safe production.
To simplify the model, this paper assumes that the oxygen underground is enough for accidents and neglects the impact of other conditions on gas explosion. In addition, it assumes that there’re people operating underground 24h a day and machinery operation doesn’t exert any impact on the space and air rate underground iterative algorithm of the simulator.

2. Literature review

Coal is an important source of energy in China. There are many important hazard sources in the safety production of coal mine and gas is one of the most important hazard sources. How to prevent the occurrence of gas disasters and ensure the safe production of the coal mine has always been an important subject for the research of coal science and technology workers. The transmission distance between the sensor and the branch station is lengthened, and the increasing of the mining concentration makes the gas change more constant, and all these bring new problems to the gas monitoring. It is the problem needs to be solved urgently how to effectively detect the field information, how to carry on the long distance transmission and real-time processing and how to ensure the long and stable operation of the system.
Foreign monitoring technology began to develop in 1960s. Up to now, there are 4 generations of products, basically updated for 5-10 years. From the perspective of technical characteristics, the development stage of monitoring system is mainly divided based on the progress of information transmission. The information transmission of the earliest coal mine monitoring system adopts air separation system. Jiping mentioned in the article that most of the British coal mine transport machine control and the fixed equipment control in the Japanese coal mines used this technology. The most representative is the CCT63-40 coal mine environmental monitoring system in France, which can measure gas, carbon monoxide, wind speed, temperature and other parameters, and 40 points are measurable at most (Jiping, 2015). Mellors et al. pointed out that the system was equipped with more than 150 sets in some Western European countries (Mellors et al., 2016). Jiping said in the article that Poland introduced technology from France in 1970s, withdrew the CMM-20 system that can be measured at 20 points, and then expanded the test point to 128 points to form a CMC-1 system, which is the first generation of coal mine monitoring system (Jiping, 2016). The main technical characteristic of the second generation of coal mine monitoring technology is the application of the frequency division technology

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of the signal channel. Because the number of cable cores used in the frequency division transmission channel is greatly reduced, it quickly replaced the frequency division system. Fu-bao and others mentioned in the study that the coal mines in the United Kingdom and other countries widely used frequency division technology in the late 1960s (Fu-bao et al., 2015). Among them, the most representative and still influential are the TST system of SIEMENS in Germany and the TF2000 system of F+H company, all of which are audio transmission systems. The application of frequency division system embodies the development of information transmission technology based on transistor circuit, which is a great step forward than frequency division technology. The development of the time division system is promoted by the emergence of integrated circuits, and the third generation coal mine monitoring system based on time division system is produced, in which the UK develops the fastest. In the study of Xie and others, it can be seen that the British Coal Research Institute launched a sensational MINOS system based on the time division system in 1976, which is quite mature, widely promoted in the UK and sold to the United States and India (Xie et al., 2015). Wang and others talked about the high speed development of modern technologies, such as computers, large-scale integrated circuits, digital communications, and other modern technologies (Wang et al., 2017). After the successful application of the MINOS system software launched by the UK Coal Research Institute, the HSDE, HUWOOD and other companies in the UK produced the system based on time division to match them. And the United States took the lead in modern computer technology, large-scale integrated circuit technology, data communication technology, and so on with its strong high-tech advantages. The new generation of high and new technology is used in the coal mine monitoring system, and the fourth generation of coal mine monitoring system with distributed and unprocessed base is formed.

The development of coal mine safety monitoring technology is late in China. The first coal mine monitoring system developed by China is the system developed by the Changzhou Institute of Automation of the Coal Science Academy. Lu and Qin pointed out that China introduced two sets of CTT63-40 from France, which were equipped with Yangquan No. 1 Mine and Yanzhou Dongtan Mine (Lu and Qin, 2015). Alvarado et al. talked about the introduction of two sets of CMC-20 from Poland at the later stage, which was equipped with Fushun Longfeng coal mine and Kailuan Zhao Ge Zhuang mine. The above is the first generation of coal mine safety monitoring system in China (Alvarado et al., 2015). Since 1990s, China closely followed the development trend of the world monitoring and control system, and developed a number of monitoring systems with advanced level in the world, such as KJ66 system jointly developed by Beijing Xian Dao New Technology Research Institute and Fushun Safety Instrument Factory, and KJ90 system of branch of Coal Science Academy and so on. Their main characteristics are: the intelligent level of the measurement and control station is further improved; it has the function of network connection; the system software generally adopts the windows operating system. With the rapid development of electronic technology, computer technology and software and hardware, as well as the requirements of the development of the enterprise itself, the Fushun branch KJF2000 of the Coal Science Research Institute and the Beijing Ruisai, as well as the integrated digital network monitoring and management system for coal mine safety such as MSNM and WEBGIS, represent the technical level of the coal mine monitoring and control system in China. At the same time, under the guidance of the policy of the coal mine safety work system and *The Coal Mine Safety Regulations*, the mine monitoring and monitoring system were installed in all large, medium and small-sized coal mines with high gas or gas outburst. At present, there are more than 20 companies and hospitals that produce monitoring and control systems in China. They mainly monitor environmental safety parameters and warn or cut off when the safety parameters exceed the limit. The monitoring and control of production equipment is mainly limited to the monitoring of the opening and stopping state of the equipment, and there is no way to monitor the whole process of coal production. It is one of the tasks for the development of monitoring and control system in China to develop a wider and larger monitoring and control system to lay a good foundation for the realization of the comprehensive automation of coal production.

To sum up, in the above research work, a deep research is made on the coal mine monitoring technology and monitoring system at home and abroad, but the research on the application of mathematical modeling methodology to the analysis of coal mine gas monitoring and control system is very few. Therefore, based on the above research status, the mathematical modeling method is used to determine whether the coal mine is high gas or not, and judge its safety degree, thus establishing the pertinent mathematical model and improved model of the coal mine.

3. Notation

Q^{ijk}: absolute gas emission of the kth monitor point at shift j on the ith day (m3/min)

Q^{k}: average absolute gas emission at the kth monitor point (m3/min)

V^{ijk}: wind speed at the kth monitor point of shift j on the ith day (m/s)

C^{ijk}: gas concentration at the kth monitor point of shift j on the ith day (%)

S^k: tunnel cross-sectional area (m2), among it, when k=6, Sk=5; when K≤ 5, Sk=4
W^{ijk}: relative gas emission (m³/d.t)
Ai: daily coal yield (t/day)

4. Model Establishment and Solution

4.1 Model I: Judging Whether High-Gas or Low-Gas Coal Mine

To judge whether a coal mine is high-gas or low-gas, the relative and absolute gas emission must be calculated and compared with the standard in Coal Mine Safe Rules, thus modeling the problem and facilitating description. Six monitor points were set: coalface I, coalface II, driving face, return airway I, return airway II, total airway, numbered as k (k= 1,2, … ,6); morning,middle and night shifts were numbered as j== 1,2,3.

(1) Calculation of Absolute Gas Emission

The absolute gas emission at the k^{th} monitor point of shift j on the i^{th} day can be obtained with gas volume passing through cross section minus the volume percentage concentration, i.e. C^{ijk}%, thus obtaining the gas volume produced within a certain time, which can be described as notation in formula below:

$Q^{ijk}= 60V^{ijk}S^kC^{ijk}\%60$ (V^{ijk}: path wind passes every minute (m))

Based on Q^{ijk} obtained above, the average absolute gas emission of this month at the k^{th} monitor point can be calculated by below formula:

$$Q = \frac{1}{90}\sum_{i=1}^{30}\sum_{j=1}^{3} Qijk$$

(2) Calculation of Relative Gas Emission

According to the definition of gas emission, the gas emission on the i^{th} day can be expressed by below formula:

$$Wijk = \frac{24*60Qijk}{Ai}(i=1,2,3...,30)$$

The absolute gas emission of various monitor points can be calculated with the model above. Standard in *Coal Mine Safety Rules* can be used as reference to judge whether a coal mine is high-gas or low-gas.

4.2 Model II: Judging Insecurity Degree of the Coal Mine

Explosions in the coal mine may be caused by two reasons: gas explosion or coal dust explosion. So we need analyze the probability of these two categories. (1) probability detection of coal dust explosion: fitting was conducted to the data in figure 1, thus obtaining the below function relationship between gas concentration C^{ijk} and y, the minimum explosion concentration of coal dust:

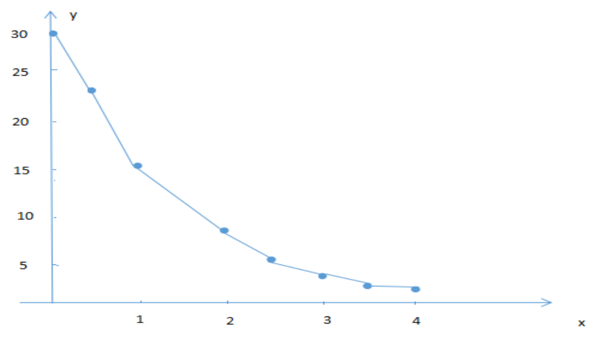

Figure 1: The function relationship between gas concentration

The equation is: y=30.3136 -19.067x+ 4.33999x2-0.34029x3
y: the minimum explosion concentration of coal dust.
x: gas concentration, i.e. C^{ijk}.
According to the function above, program was written to calculate the coal dust concentration R^{ijk} at every monitor point of very shift on every day to compare it with the minimum explosion concentration of coal dust

defined by the function y= f(C^{ijk}), thus testing whether the coal dust concentration in the checking table exceeded the corresponding minimum explosion concentration defined by the gas concentration Cijk and finally obtained the probability of coal dust explosion.
Probability of Gas Explosion:
According to the *Coal Mine Safety Rules*, we think that there is risk of explosion when it reaches the alarming concentration (≥ 1. 0%). To judge whether there is risk of explosion, among all the gas content in the Attached List 2 obtained through testing, 1.18 is the highest, i.e. all the data in the list falls within the area surrounded by the four lines of 0< x≤ 4, 0< y≤ f(x), x= 0 and y= 0, and the points within the area surrounded by y= 0, y≤ f(x), x= 1, x= 1. 18 still owns risk of gas explosion, so the probability of gas explosion is the percentage of the shadow area S to the area below the curve, i.e. a problem of geometric probability. Figure 2 shows Geometric probability.

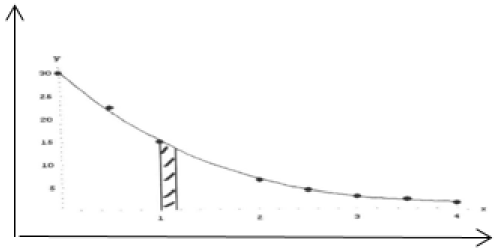

Figure 2: Geometric probability

Area of these two parts can be calculated by integral:
Shadow S=∫1.18130.3136-19.067x+ 4.33999x2-0.34029x3dx
S=∫4030.3136-19.067x+4.33999x2-0.34029x3dx
Insecurity degree of gas explosion is approximately as below:

$$S = \int_{1}^{1.18} 30.3136 - 19.067x + 4.33999x \wedge 2 - 0.34029xdx$$

Therefore, the insecurity degree (no risk of coal dust explosion) is around 6.49%.

4.3 Model III: calculating the total ventilation and sub-ventilation of each working face

To facilitate description, below symbols were defined:
x^1: ventilation required for coalface I;
x^2: ventilation required for coalface II;
X^3: fresh air flow required for the tunnel where the local fan is situated
x^4: fixed air flow required by the local fan
Model above was established according to the wind diffluence at the tunnels as shown in the layout diagram of the mine:

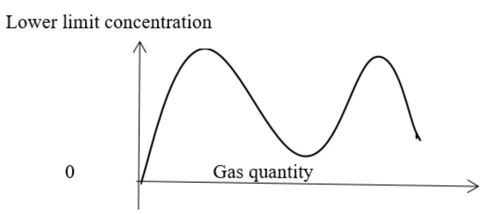

Figure 3: Function model

(1) Total air flow (the sum of air flow at each ventilation tunnel) can be calculated by below formula:
z=∑4i= 1xi
(2) Air flow required for coalface can be calculated as below:
x^i= 100 QiK(i=1,2)(K is spare air coefficient, normally K=1.2~1.6)
The minimum wind speed of mining working face defined in the *Coal Mine Safety Rules* can be used to estimate the minimum air flow required in one minute:

$x_i \geq 60 \times 0.25S_i (i = 1,2)$ (The maximum wind speed of coalface defined in the *Coal Mine Safety Rules* can be used to estimate the maximum air flow required in one minute: $x_i \leq 60 \times 4S_i (i = 1,2)$)
(3) Air flow required at the driving face can be calculated as below:
$x^4 = 100Q3k0$(k^0 is spare air coefficient, normally $k^0 = 1.5\sim2.0$)
At least 15% of fresh air flow is required in the tunnel where the local fan is situated to ensure that wind flows normally there and that there won't be counter current due to dirty air which would be hence absorbed into the driving face. So the limit to x^3 is as follows:
$x^3 \geq (x^3 + x^1)15\%$
As wind speed influences the concentration of coal dust, the bigger the wind speed is, the larger the amount of coal dust will be. There exists function relationship between them, i.e. $R^{ijk} = g(V^{ijk})$. In order to assure no coal dust explosion, the concentration of coal dust must be lower than its minimum explosion concentration (decided by the amount of gas): $R^{ijk} \leq f(Q^3)$. Based on above, below model as figure 3 has been established:

4.4 Object function analysis

Initial analysis shows that the ultimate goal of the third problem is to obtain the optimum total air input, which can be solved with the goal programming model. Safety degree in the well can be selected as the object function. And by adjusting the control damper to control the air flow reaching the mining points and excavation working points, it can be assured that the well reaches the maximum degree of security. Safety degree for the well= degree of gas safety + degree of coal dust safety.
In terms of the gas, considering the power-off required concentration for working face I is 1.5%, when the gas concentration reaches 1.5%, degree of gas safety can be considered to be 0%; when gas concentration reaches 0%, degree of gas safety can be considered to be 100%. so 1.5%-Y21.5% has been selected as the degree of gas safety.
Similarly, in terms of the coal dust, when its concentration Y^1 reaches the minimum coal dust explosion concentration Y^3, its safety degree can be considered to be 0%; when its concentration Y^1 is 0%, its safety degree can be considered to be 100%; therefore Y^3-Y^1Y^3 has been selected as the safety degree of coal dust, i.e. the object function is 1.5%-Y21.5%+Y^3-Y^1Y^3.

4.5 Constraint condition analysis

Data fitting was conducted with Matlab and reached below relational expressions:
1) Fitting function between wind speed x and coal dust concentration Y: Y^1=3.02321+2.06861x
2) Fitting function between wind speed and gas concentration: Y^2=3.3174e-0.5121x
3) Function between minimum coal dust explosion concentration and gas concentration: Y^3=24.5138-7.06967Y^2.
According to the air volume in every tunnel as shown in Figure 1, and the requirement of wind speed in every tunnel as defined in Table 2 of Provision 101 in the Attachment 2, it can be known that Vfi in the coalfaces <<5m/s; Vfi in the major intake and return airways <8m/s; Vfi in the mining areas and the return airway fell between 0.25m/s~6m/s; Vfi in the mining working and driving faces fell between 0.25m/s~4m/s; Vfi of stone drifts being excavated fell between 0.15m/s~4m/s. meanwhile, when gas concentration is 1.5%, wind speed is 0.872m/s, so the appropriate wind speed x should be >0.872m/s.
Based on above, a programming model can be obtained with the maximum safety degree as the object function and the wind speed as constraint condition.

4.6 Model establishment Max1.5%-Y21.5%+Y3-Y1Y

$$S.T.\begin{cases} Y1 = 3.02321 = 2.06861x \\ Y2 = 3.3714\mathrm{e} \\ y2 < 1.5 \\ x < 4 \end{cases}$$

Lingo8.0 the solution turns out to be x=4.0, i.e. the safety degree is the largest when the wind speed is 4m/s. then the gas concentration is 0.4277492%.

5. Model improvement

Through online searching and induction it can be known that among well explosion, coal dust explosion occupies around 33% while gas explosion occupies around 67%. The ratio is around 1:2. However, the power of coal dust explosion is 10 times that of gas explosion, i.e. the harm from coal dust explosion is 5 times that of gas explosion. Then the weight of harm from gas explosion to the coal dust explosion in the well is around 1:5. The model can be established as below concerned with the problem of weight:

$$\max 1/6 * \frac{1.5 - y2}{1.5\%} + \frac{5}{6} * \frac{y2 - y1}{1.5\%}$$

Solution with the Lingo 8.0 turns out to be x=2.551252, i.e. when the wind speed is 2.551252m/s it's the safest with gas concentration of 0.8982429%, and 2.551252 is the optimal ventilating speed of the working face. Considering that air flow in the driving tunnel depends on the fixed air volume of the local fan 150U400m3/min, i.e. adding a wind speed limiting condition, x falls within 0.625U1.67m/s.
With the same method the mining working face II and the air volume of the local fan is shown as table 1.

Table 1: Air volume of a local fan

	Coal mining face	Local ventilator
Do not consider the weight of harm	2.113855m/S	0.872m/S
Consider the weight of harm	2.115091m/S	0.872m/S

Considering there is at least 15% of ample air volume in the tunnel of local fan where 209m3/min of air volume is required while the air volume at Air Door 2 should be 240m3/min. Local demand for wind is as shown as table 2.

Table 2: Local demand for wind

	Coal mining face1	Coalmining face2
Do not consider the weight of harm	848m/min	507m/min
Consider the weight of harm	487m/min	507m/min

6. Conclusions

This paper adopts mainly the method of mathematical modeling to calculate the gas content under a certain conditions. After calculation, it has been found that the optimal total air volume required in the well is 1595m3/min without no regard to the weight of harm; taking the weight of harm into consideration, the optimal total air volume required in the well is 1178.656m3/min.

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