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Designing The Ventilation System For Galandroud Coal Mine

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ABSTRACT: Galandroud colliery, which was recognized as a highly gasous coal mine in north of Iran, is one of the important mines in terms of its coal properties. In view of the fact that methane was the major existing gas in the mine and its natural flow is upwards, the ascending ventilation was believed to be the most efficient. In addition to this, due to presence of a number *of* adits in upper levels of mine, the *forcing* ventilation system acting from the lower level of mine was chosen. Firstly, the total required fresh air quantity was empirically calculated to meet the ventilation requirements. Secondly, the total pressure lost was found using analytical methods. Furthermore, computer model network simulations were conducted so as to evaluate the effects of different network configurations and ventilation adjustments. Finally, an axial blower type of fan which meets the above needs as well as the future ventilation requirements was also selected according to an economic challenge. All results obtained from different methods were compatible.

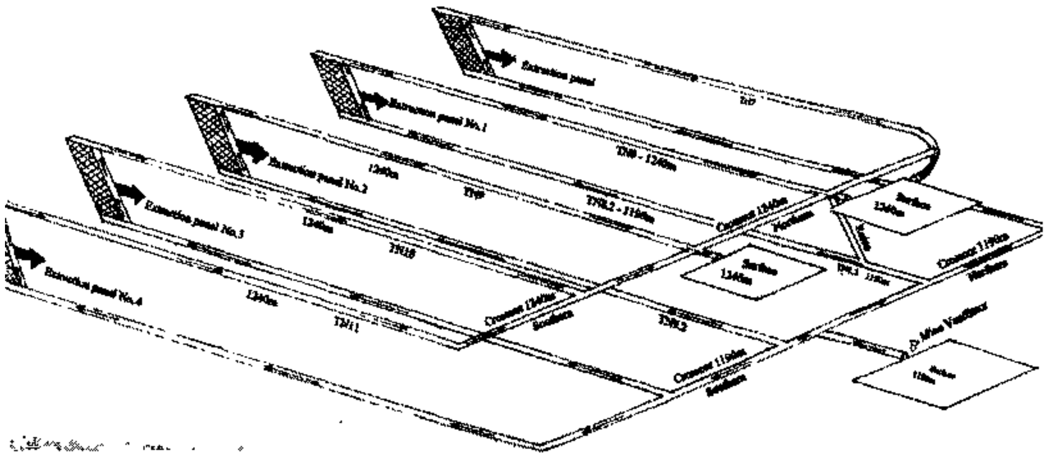
I INTRODUCTION

The Galandroud coal mine, affiliated with National Iranian Steel Company (NISCO), has been opened and developed quite recently. Having been completely explored, the mine was divided into 2 main levels namely +l 190 and +1240. Due to the topography of the area, horizontal adits having E-W orientation were considered in order to access the deposit. Crosscuts with orientation of N-W were derived in order to make the extraction panels. Owing to the thickness and dip of coal layers, five conventionally diagonal retreat longwall faces were regarded as the main method of mining. The total annual production from the mine was approximately 150,000 tons. Since the methane emission rate per ion of coal mined was IOnr/t, Galandroud coal mine is categorized as a highly gaseous coal mine (GIS GeoincTustria, 1998).

Design methods of mine ventilation system can be categorized into three groups: (1) analytical methods, 12) empirical methods and (3) observational methods (Osgoui, 2000).

Analytical methods deal with fundamental concepts of fluid mechanics in conjunction with applied mathematics. In cases where the ventilation network is simple (initially planed ventilation network), analyses are carried out easily; on the other hand, in the case of complex networks (investigating or upgrading the in-use ventilation network), handy methods would be inapplicable: accordingly, computer-aided and numerical methods are taken into consideration to be more appropriate. Empirical methods include the governmental, health and safety regulations as well as experimental rules. Nowadays, empirical methods are best known as a prerequisite for analytical methods. Observational methods rely on consistently monitoring the ventilation system in the course of time. Incessantly measuring and inspecting the amount *of* harmful or explosive gases, air velocity and quantity and air pressure are the chief essence of this method. Therefore, a primary ventilation system can not be developed unless observational methods are taken into account. Using the above mentioned methods appropriately, a perfect ventilation system for both initial and further stages of mining will be achieved.

In the Galandroud coal mine, planning the ventilation system was carried out based on analytical and empirical methods; however, computer model network simulations were conducted in order to evaluate the effects of different network configurations and ventilation adjustments. A overview of the ventilation network layout is presented in Figure l.



Figuie I Mine ventilation network layout

2 REQUIRED QUANTITY OF AIR

The amount of air needed should be determined in such a way that it will satisfy different requirements such as breathing, dilution of harmful gases and dust produced in the mine (Hartman et al. 1997 & Madani, 1987).In the Galandroud coal mine, air quantity for working places was estimated based upon the rate of gas emission and the number of workers. In view of the differences between the rate of gas emission per coal mined at the beginning and end of every work shift stem from workers' fatigue, the rate of product at the beginning of the shift is therefore counted twice as much as that at the end of the shift. According to a survey conducted at longwall faces, the rate of gas emission at the beginning of the work shift was found to be 0.0572m"/s. Consequently, the amount of fresh air for any longwall face to keep the gas concentration below 0.75% is calculated as follows! Osgoui. 2000):

$$Q = \frac{100}{0.75} p\lambda$$

Where Q is the required air quantity (m/s), p coal production of any stope (l/h) and x is gas volume acquired from one ton mined coal *(mit)* and therefore:

$$Q = \frac{100}{0.75} \times 0.0572 = 7.63 m^3/S = 457.77 m^3/\min$$

Considering 30% air leakage from different parts of mine, required fresh air necessary for each stope is computed:

$$Q = \frac{7.63}{1-0.3} = 10.9 m^3/S$$

For unpredicted problems, which cause the inadequacy of the ventilation requirements as well as in order to meet the federal regulations, estimated air quantity has to contain a safety factor of 20%. Accordingly, given 5 longwall faces, the total air quantity for all stopes (or the total air quantity for mine) will be 65nrVs. The manner of total air distribution is demonstrated in Figure2.

3 PRESSURE LOSS ESTIMATION

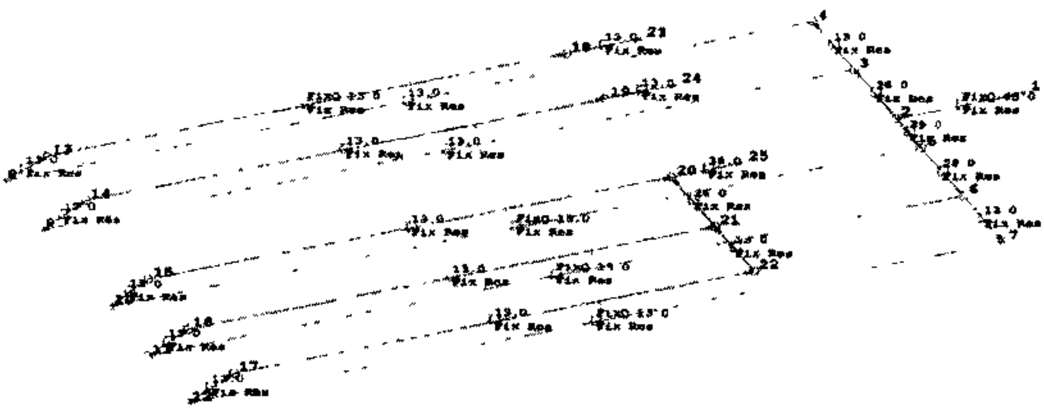
In order to calculate the pressure loss in each branch of mine airways it is necessary to compute the friction factor and the resistance. Based on physical characteristics of airways as well as empirical equations, the friction factor values utilized were taken into account as following:

For main adits and major tunnels (A= **11.2 m^2)** K=().01484kg/m''

For minor tunnels and drifts (A= 7.2 in") fc^O.OIHSSKg/m[1]
For working places (A= 4.66 m^2) K=0.04638 Kg/m'

$$R = K\frac{(L+L_e)}{A^3}P$$

Where L_L. ıs equivalent length of airway which is being obtained from literatures. Therefore, statistical



Figuie 2 An quantity distribution (m'/s)

It was interesting at this point that while back analyzing the data obtained from monitoring the ventilation system; the K values acquired by both empirical
expressions and inverse equations were in reasonable agreement.

Having been calculated the friction factor, resistance of airways were to be computed by Atkinson equation(Hartman et al. 1997 & Ramani 1992a, b):

$$R = K\frac{LP}{A^3}$$

Where R is resistance of airway (NS[3]/™*), K is the friction factor of airway (Kg/m), L is length of airway (m), P is airway perimeter (m), and A is cross-sectional area of airway (nr).

Note that pressure loss in each branch of the ventilation network is equal to the summation of corresponding factional pressure loss and shock head loss: accordingly, in order to determine the total pressure lost, the value of shock loss has to be taken into consideration. In the Galandroud coal mine, the effect of shock loss is surveyed by means of equivalent-length method. This method, indicated by below equation, has been strongly suggested by Hartman et al.(1997).

pressure loss of each branch of ventilation networks is determined by:

$$\Delta P_s = RQ^2$$

Where Q is the air quantity passing through airway (m /s). R is airway resistance, and AP, is statistic-pressure loss (Pa or mm-water).

3.1 Adjusting the ventilation network

When airways are arranged in parallel and a prescribed quantity of air is made to flow through each branch, controlled splitting is being utilized. Controlled splitting is used to obtain the desired quantity of air through workings and airways forming splits, rather than allowing natural splitting to prevail and the air to apportion itself. In order to solve mine-ventilation networks employing controlled splitting, the number of required meshes for applying Kirchhoff s Second Law must be calculated as follows:

$$N_m = N_b - N_j + 1$$

Where N_m is the minimum number ot meshes, N_b is the number of branches, and N, is number of junction. Therefore, the minimum number of meshes were determined as follows:

$N_{,,}= 12-9 + 1 =4$ independent loops

Having applied the first and second Kirchhoffs law, desired pressure losses provided by modifiers (negative regulators) in defined branches were computed. Adjusted ventilation network is illustrated in Figure3 and the total mine statistic pressure loss was determined to be 320 Pa (32.58 mm.water).

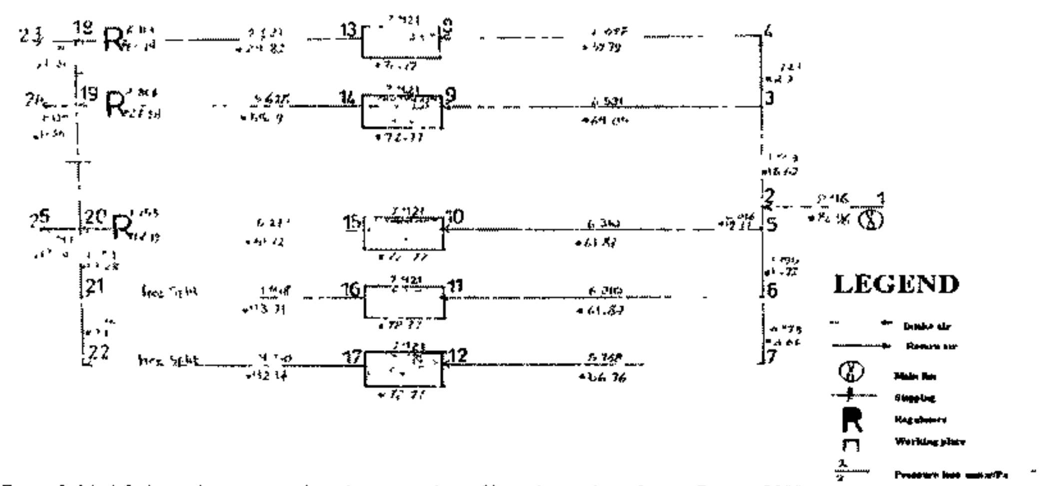


Figure 3 Modified ventilation network with pressure loss of branches and regulators (Osgoui, 2000).

3.2 Estimation of dynamic and total pressure loss of inme

In order to determine the total pressure loss, it is essential to calculate the dynamic pressure loss in all exit airways. Because of forcing system of air distribution in such a way that total input air enters the system by tunnel 6 and exits the system through tunnels 7.8,and 9, it is therelore imperative to determine the dynamic pressure loss in each exit tunnel. The maximum dynamic pressure loss obtained is regarded as the dynamic pressure loss of the mine. In view of the fact that maximum intake air quantity passes through exit tunnel 9, it is therefore expected that maximum dynamic head loss will occur in that tunnel. Hence, dynamic pressure loss will be:

$$\Delta P_1 = \gamma \frac{V^2}{2g}$$

Where y is unit weight of air, Vis air velocity ,and g is gravitational acceleration. Hence, for exit tunnel 9:

$$\Delta P_1 = \gamma \frac{V^2}{2g} = 1.2\times\frac{3.48^2}{2\times 9.806} = 0.7409 mm.water$$

As a result, the total head loss of mine will be obtained from summation of both dynamic and static pressure loss, hence:

$$\Delta P_t = 32.58 + .7409 = 33.4\, nim.water\,(328pa)$$

4 SELECTION OF THE MAIN FAN

In general, the main fan to be selected must be capable of delivering 65mVs at a minimum pressure of 328Pa. In order to meet the ventilation requirements, two types of main fans were therefore chosen. According to their adeptness and capital cost, it was advisable to purchase a Iranian-made fan instead of importing one. Consequently, an axial blower fan with blade setting ability was exclusively preferred to be more efficient. As a result of relatively lower pressure loss at the first stage of the ventilation system, at the blade set to 30" and little efficiency of fan, ventilation needs of system is readily satisfied. Technical characteristics of the chosen main fan are demonstrated in the Table I.

C- analyzing the ventilation network in the case of employing fan made by USA.

Table I - Technical charactei istics of axial main fan (2CQ-200)

F.m du	Vdcx ity	Power	Efficiency		Provided pressure		Provided air quantity		
			Max	Mm	Max	Mm	Ave	Max	M in
2000	900	5-250	80	62	2 1	0 1'J	70	120	35
mm	ipm	kW	</<		kPa		mVs		

Over the life of the mine, the demand for head and quantity will vary; therefore, the duty on the fan may dramatically increase due to new operating sections or decrease due to the addition of new shafts and tans. The selected fan will be able to meet all future ventilation requirements. Based on the operating point of mine, the required power of the fan can be calculated as follows(Hartman et al. 1997 & Ramanı 1992a, b):

$$N = \frac{Q_t \Delta P_t}{\eta_t \eta_2 1000}$$

Where Q, is total air quantity (m /s). AP, is total head loss (Pa), ŋi fan efficiency, n, electromotor efficiency, and N required fan power. Hence fan power will be:

$$N = \frac{65 \times 328}{0.66 \times 0.70 \times 1000} = 43kW$$

In view of the required fan, it seemed another alternative, Jeffrey fan 8HU-72, could have been a suitable choice, satisfying the initial ventilation requirements. The fan characteristics and operating point of mine when using Jeffrey fan 8HU-72 illustrated in the Figure4.

5 VENTILATION NETWORK SIMULATIONS

Generally, computer model network simulations are realistically essential in order to appraise the effect of different network configurations and ventilation adjustments (McPherson, 1993 & Bandopadhyay, 1992). Ventilation network simulation were conducted by three simulator softwares namely; MINVENT, VNETPC, and VENTSIM. Ventilation network was then surveyed based on the following cases:

A- analyzing the ventilation network without use of a fan.

B- analyzing the ventilation network in the case of utilizing fan made by Iran.

In all mentioned categories, the primary layout of the ventilation network was chosen in the way that it would be in agreement with other mining operations in addition to ventilation adjustments. Based on results obtained Irom simulating the ventilation network by three simulators, it was concluded that they are in reasonable agreement (Osgoui, 2000).

In the case where the software has to select the main fan, all three simulators suggest delivering 65mVs air quantity at a pressure of approximately 300 Pa through tunnel 6. Moreover, for delivering the fixed air quantity through working places, both VNETPC and VENTSIM advise that booster fans must be installed in tunnel 9, 10, and 11 and a regulator has to be put in tunnel 7. According to MINVENT's results, however, in tunnel 8, 9. 10, and 11 the booster fans must be established. It can be easily concluded that while VENTSIM and VNETPC used both negative and positive adjustment (combination regulation), MINVENT only utilizes positive regulation.

In the case where the Iran fan is used, all three simulators have identical results. Installation of main fan at tunnel 6 with air quantity of 65mVs at the range of pressure between 260Pa and 275 Pa are elucidated by them. However, to satisfy the operating point of the mine, it is necessary to use the booster fans in tunnel 8, 9, 10, and 11 and utilize regulator in tunnel7.

On the other hand, as far as the jeffry's fan is employed, all three simulators have also indistinguishable conclusions. The significant difference between the results of VENTSIM and MINVENT is that in

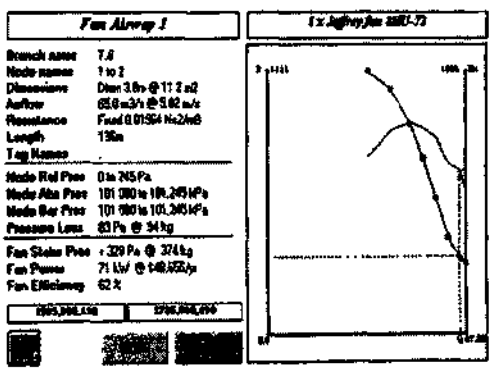


Figure 4 Operating poiiu of mine by simulating the ventilation network in the ease of using Jeffrey fan' fan static piessure =328MPa. an quantity=65m /s. fan efficiency =62% (Osgoui. 2000)

the latter one, the main fan load is reduced and this deficiency will be compensated by booster fans.

5 CONCLUSIONS

The design of ventilation system in Galandroud coal mine was conducted base upon main network of mine in the way that no additional airway was driven. However, to open and develop the mine, the ventilation effects were notably taken into account. Moreover, in regard to mine network configuration and adjustment, multi split system was recommended to be more appropriate. The main advantage of this system is that a new potential panel can be easily either linked or separated from the other ventilation circuits. All results obtained from ventilation network simulation are in remarkable compatible with those acquired from handy calculation. With the help of ventilation simulation, as the mine is extended, so the ventilation parameters such as required air quantity and pressure loss of each and then whole airways can be determined resulting in new operating point of mine would be clarified.

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