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Coal and Oil Sands

▲ Assessment of the efficiencies of auxiliary ventilation systems using empirical methods

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KEYWORDS: Underground mining, Auxiliary ventilation systems, Methane gas, Air flow, Coal.

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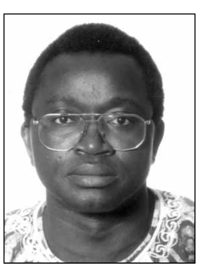
ABSTRACT

Increasing depths and mechanization of underground mines have led to the production of large amounts of gaseous and particulate contaminants. Over 100 000 lives have been lost due to methane gas and dust explosions in coal mine workings in both the United States and Canada since 1900. There is, therefore, the need to constantly assess and evaluate the performance of existing mine ventilation systems to maintain safe and acceptable mine environmental conditions. This paper advances research initiatives in the control of methane gas in underground mine environments. It uses the results of continuous monitoring of methane gas concentrations conducted in selected coal mines in North America to assess the effectiveness of existing auxiliary ventilation systems to control methane gas concentrations. The results show that the average quantities of fresh air required to dilute, disperse and remove methane gas concentrations within set levels of one minute varied from 5.43 m^3/sec. to 27.97 m^3/sec. in the development headings. The average dilution times in the headings studied were less than eight minutes. The calculated dilution efficiencies of the auxiliary ventilation systems in the headings varied from 12% to 139%. These efficiencies ranged from poor to excellent. This implies that the auxiliary ventilation systems were capable of controlling the methane gas concentrations below statutory levels but may not be able to cope with large and unusual methane gas concentrations in the headings. This study is significant in the control of methane gas and coal dust explosions in coal mines.

Introduction

The ever-increasing lengths of underground mine headings and their limited cross-sectional areas lead to large variations in the environmental conditions within these headings. It is therefore necessary to constantly assess and evaluate the performance of auxiliary ventilation systems in development headings to ensure safe working conditions. The main objectives of this work are to: use the results of continuous methane gas monitoring in development headings to calculate the methane gas emission rates; calculate the required quantities of fresh air required to dilute methane gas concentrations in mine headings to set limits; and determine the dilution times of methane gas and assess the efficiencies of the auxiliary ventilation systems under set conditions. This is necessary to maintain safe and acceptable mine environmental conditions in workings as required by federal, state or provincial laws. Accordingly, the methane gas concentrations were monitored in six development headings in coal mines in North America*. Empirical relations on mass flow and dilution of a contaminant in an opening were used to calculate the efficiencies and performance of the auxiliary ventilation systems studied under set conditions.

The following two sections deal with methodologies for determining the dilution requirements and liberation and concentration build-up rates for methane gas contaminants. The two sections thereafter deal with the empirical relations for determining dilution, efficiency and fresh air requirements and auxiliary ventilation efficiency under two general conditions. An application of the model to real-world cases with discussions and analysis of the results have been provided in the two final sections. A guide to practicing ventilation engineers on how to use the equations developed in this paper to assess the efficiencies of the auxiliary ventilation systems in mines is given in the Appendix.



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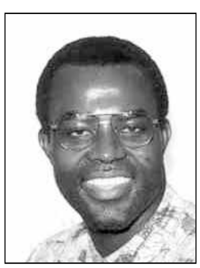

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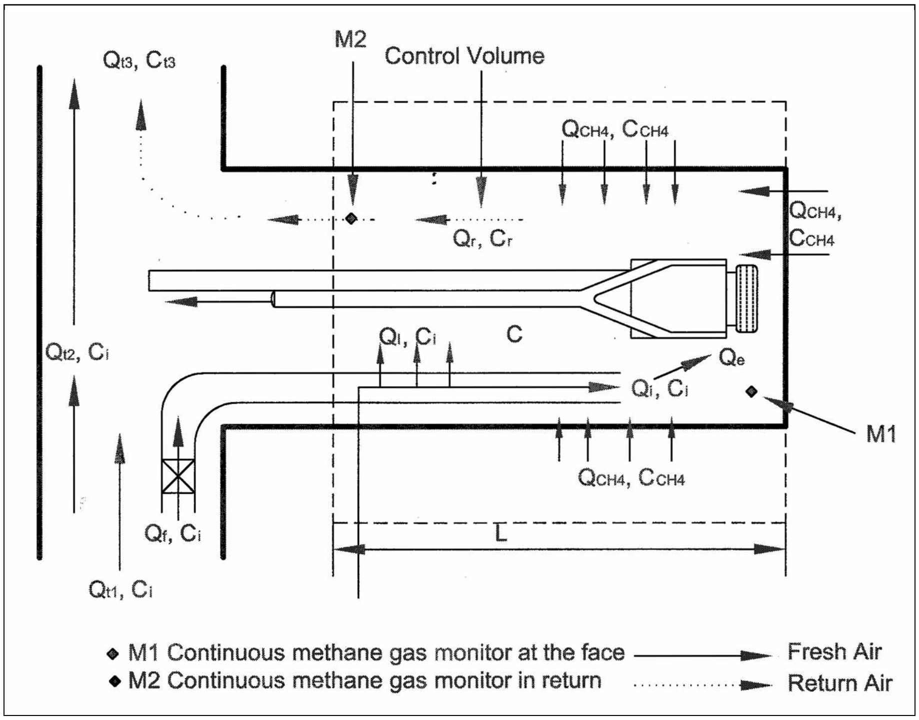
* The names of the mines are being withheld for confidential reasons.

Fig. 1. Schematic of air flow parameters in a mine development roadway.



Determination of Dilution Requirements

Certain critical design factors are required to determine the dilution requirements in any mine opening that will effectively and safely dilute the concentration of any contaminant to below statutory levels. These include the rate of contaminant emission into the opening (room), the volume of the room, and the nature, composition and concentrations of the mixtures of gaseous and particulate matter in suspension in the room.In addition,it is necessary to know the explosibility of the mixture of gases in the opening, if any. The Coward triangle (for methane gas) expresses the graphical relation between the composition and explosibility of methane-air mixtures (Raybould and Hughes, 1960). Knowledge of the explosibility of a particular mixture of gases allows the determination of the dilution requirements and how to effectively and safely dilute the concentrations to safe levels. Special dilution techniques are required to safely deal with explosive atmospheres when detected. The ultimate objective of any ventilation system is to maintain the concentration of oxygen in all workings as close as possible to 20.95% and to maintain the concentrations of the mixture of gases in the working below statutory levels.

Methane Gas Liberation and Concentration Build-up Rates

Methane gas, which is the product of the coalification process of coal, is retained or stored in coal in two different ways. Methane gas is retained as free gas compressed in the pores, fissures, cleats and fractures that are almost always present in the coal, and as adsorbed gas on the microscopic surfaces of coal as well as in the micropores (Murray, 1990; Curl, 1978; Feng and Augsten, 1980; Konda,1985).At very high pressures, methane gas dissolves to a considerable extent in the free water that exists in the cleats and fractures of a coal seam (Greig, 1982).When this water enters the mine openings at lower pressures, the dissolved gas is released into the adjacent air. The free methane gas in the pore spaces and fissures in coal is at equilibrium with the adsorbed gas on the surfaces of the pores and fissures.There is a constant interchange of molecules between the free gas and the adsorbed gas phase (Curl, 1978).

It is difficult to prevent the emission of methane and other strata gases into mine openings as a result of the natural processes involved. Thus, it is necessary to adopt ways of reducing the emissions and concentrations of methane below 1.5% by volume in air using various methods. These methods include methane drainage in advance of mining, sealing off mined out or very gassy sections and areas, and dilution by the main or auxiliary ventilation systems. The latter is the most commonly used in coal mines when the methane gas emission rate is low as it is more versatile, least expensive and the most effective of the three options (Miller and Dalzell, 1982). The other options become economically feasible at high methane gas emission rates (Vutukuri and Lama,1986).

Empirical Relations

Most of the auxiliary ventilation systems studied in this work were predominantly the primary forcing systems with secondary exhaust overlaps. Figure 1 shows a roadway that is being developed from a main heading using a roadheader. It shows an overlap auxiliary ventilation system. The auxiliary forcing ductings are installed on one side of the roadway while the secondary exhaust overlap system is mounted on the heading machine. As a result, the intake end of the secondary exhaust ducting was constantly within 1 m from the face when the machine was cutting coal. The properties of the air are analyzed as it enters and leaves the control volume as shown in the figure. The definitions of the symbols used in all equations are given in the Nomenclature at the end of the paper. Part of the total air quantity flowing through the main drive (Q_{t1}) passes through the forcing fan (Q_f). However, due to leakages and possible short-circuiting of the air, only a portion of the intake air (Q_e) gets to the face of the heading. Methane gas is also emitted into the heading from the roof, floor and walls of the heading at a rate of Q_{CH_4} and at a concentration of C_{CH_4}. The vitiated air in the heading (Q_r) is exhausted from the heading and joins the rest of the fresh air at the last through cross-cut.

Under steady state conditions where there is neither compressibility nor absorption/adsorption of gases in the system, the following constraints are applicable in estimating methane gas build-up rate in a control volume (G, m^3) within a roadway in a coal mine: $G > 0$, $C_{CH_4} \geq 0$. The efficiency of the forcing auxiliary ventilation system (E_f) is given by equation (1):

$$E_f = \frac{Q_i}{Q_f} \times 100\% \quad \ldots\ldots\ldots\ldots (1)$$

From Figure 1, $\quad Q_f = Q_i + Q_l \quad \ldots\ldots\ldots\ldots (2)$

The overall efficiency (E_0) of an overlap auxiliary ventilation system is given by equation (3) as:

$$E_o = \frac{Q_e}{Q_f} \times 100\% \quad \ldots\ldots\ldots\ldots (3)$$

Employing the law of conservation of mass, mathematical relations can be derived on the airflow through a region in a space called the control volume (Fig. 1). The properties of the air are analyzed as it enters and leaves the control volume. Assuming instantaneous and thorough mixing of the gases, the concentration of methane gas in a room under steady state conditions is expressed as (Wala and Kim, 1985):

$$C = \frac{Q_{CH_4}}{Q_i + Q_{CH_4}} \times 100\% \quad \ldots\ldots\ldots (4)$$

From Figure 1, the following equations may also be written:

$$Q_r = Q_i + Q_{CH_4} + Q_l \ldots\ldots\ldots\ldots\ldots (5)$$

$$C_rQ_r = C_iQ_i + Q_lC_i + C_{CH_4}Q_{CH_4} \ldots\ldots\ldots (6)$$

$$Q_{tl} = Q_f + Q_{t2} \ldots\ldots\ldots\ldots\ldots (7)$$

$$Q_{t3} = Q_{t2} + Qr \ldots\ldots\ldots\ldots\ldots (8)$$

From equations (5) to (8), the following relations are obtained:

$$Cr\,(\%) = \left[\frac{Q_fC_i + C_{CH_4}Q_{CH_4}}{Q_f + Q_{CH_4}}\right] \times 100\% \ldots\ldots (9)$$

$$Q_{CH_4} = \frac{Q_f}{100}\left(\frac{C_r - C_i}{C_{CH_4} - C_r}\right) \ldots\ldots (10)$$

The main source of methane gas emissions into mine openings is from the newly exposed working face. Other minor sources of methane gas emission are from the roof, floor and side walls of the opening (particularly if there are coal seams below or above the opening), and from the gob areas (when the airway is close to worked out areas). The dilution requirements of methane gas in a given volume, G (m^3), at the face of Figure 1 (under unsteady state conditions) can be expressed by the following ordinary differential equation (Wala and Kim, 1985):

$$\frac{GdC}{dt} = Q_iC_i + Q_lC_i + Q_{CH_4}C_{CH_4} - \left[Q_{CH_4} + Q_i + q_l\right]C$$
$$= Q_fC_i + Q_{CH_4}C_{CH_4} - \left[Q_f + Q_{CH_4}\right]C \quad \ldots\ldots (11)$$

Integrating equation (11) with respect to elapsed time (t) and applying the initial conditions (at $t_o = 0$, $C = C_o$), the following relationship is obtained:

$$C = \left\{\frac{Q_fC_i + Q_{CH_4}}{Qf + Q_{CH_4}}\right\} +$$
$$\left[C_o - \left(\frac{Q_fC_i + Q_{CH_4}C_{CH_4}}{Q_f + Q_{CH_4}}\right)\right] \times e^{-\left[\left(\frac{Q_f + Q_{CH_4}}{G}\right)t\right]} \quad \ldots\ldots\ldots (12)$$

From which t can be derived as:

$$t = \left(\frac{G}{Q_f + Q_{CH_4}}\right) \ln\left[\frac{Q_fC_i + Q_{CH_4}C_{CH_4} - C_o(Q_f + Q_{CH_4})}{Q_fC_i + Q_{CH_4}C_{CH_4} - C(Q_f + Q_{CH_4})}\right] \ldots(13)$$

In the unsteady state, knowing the ventilating quantity, the gas inflow rate, the initial and final gas concentrations, the time for dilution can be determined from the following equation (Miller and Dalzell, 1982):

$$t = \frac{G}{Q_f} \ln\left(\frac{Q_{CH_4} - Q_fC_o}{Q_{CH_4} - Q_fC}\right) \ldots\ldots\ldots (14)$$

Where the leakage quantity (Q_l) through the ventilation ductings is very small compared to the inflow rate of fresh air (Q_i) into the heading (i.e., $Q_i \cong Q_f$), equation (14) becomes:

$$t = \frac{G}{Q_i} \ln\left(\frac{Q_{CH_4} - Q_iC_o}{Q_{CH_4} - Q_iC}\right) \ldots\ldots\ldots (15)$$

It is necessary to determine the rate of decrease in the concentration of methane gas (or of any contaminant) over a period of time by some fixed rate of ventilation. Assuming that the rate of emission of methane gas (or the contaminant) is very small compared to the volume flow rate of the intake air into the heading, equation (15) is further simplified to (Hemeon,1963):

$$t = \frac{G}{Q_i} \ln\left(\frac{C_o}{C}\right) \ldots\ldots\ldots\ldots (16)$$

In the steady state condition, a prevalent condition in dilution (Miller and Dalzell,1982), where thorough mixing of the gases is assumed and time for dilution is very long, ($t \rightarrow \infty$) and assuming that the leakage quantity (Q_l) is very small, equation (12) simplifies to:

$$Q_i = Q_{CH_4}\left(\frac{C_{CH_4} - C}{C - C_i}\right) \ldots\ldots\ldots\ldots (17)$$

When the concentration of methane gas emanating from the strata (C_{CH_4}) is assumed to be pure (100%), equation (17) becomes (Tsay et al.,1990):

$$Q_i = Q_{CH_4}\left(\frac{1 - C}{C - C_i}\right) \ldots\ldots\ldots\ldots (18)$$

Efficiencies of Auxiliary Ventilation Systems

In the determination of the dilution requirements for a heading or roadway, some researchers recommend that the peak values of the methane gas concentration attained during cutting be used as the initial methane gas concentration that has to be diluted to the statutory levels (Dunmore, 1981). This often leads to the design of auxiliary ventilation systems with excessive capacities culminating in unwarranted ventilation costs (since the cost of power is directly proportional to the cube of the quantity of air supplied). In this study, the following dilution scenarios were considered in assessing the efficiencies and effectiveness of the various auxiliary ventilation systems:

Condition 1 — The methane gas emanating from the strata is assumed to be pure (100%) in concentration by volume, and the concentration of methane gas in the return air is taken as that registered by the methane gas monitor in the return (outbye).

Condition 2 — The methane gas concentration at the face is taken as that registered by the monitor at the face, and the concentration of methane gas in the return air taken as that recorded by the return air monitor.

Applications in Underground Mine Development Headings

The above models were applied to underground mine development headings. The dimensions of the development headings studied varied from 4.88 m by 1.63 m through 5.18 m by 1.52 m to 6.71 m by 1.4 m. The lengths of the faces from the nearest through cross-cut varied from 73 m to 1120 m. Depths below surface of the workings varied from 420 m to 1348 m. The types of auxiliary ventilation ductings used were either rigid, flexible or flexible wire-reinforced ductings with diameters ranging from 300 mm to 1070 mm. The auxiliary forcing fans were predominantly vane axial flow Engart fans of 22.38 kW, 44.76 kW or 55.95 kW capacity while the exhaust fans were of smaller capacities ranging from 7.46 kW to 37.3 kW. The auxiliary fans were either pneumatically or electrically driven. Average mine temperatures in summer ranged from 11°C to 20°C while that in winter varied from 8°C to 16.7°C. Relative humidities of the mine air ranged from 4% to 100%. On average, the advance achieved per shift by the roadheaders varied from 2.29 m to 91.74 m depending on the rock and operational conditions at the heading.

The concentration of methane gas at the face and about 60 m in the outbye, in three development headings, were monitored continuously throughout the shift with continuous methane gas monitors (CSEs) M1 and M2 as shown in Figure 1. Equations (1) to (18) were employed in calculating the efficiencies of the auxiliary ventilation systems, methane gas emission rates and dilution times of methane gas in the development headings (Suglo, 1995). The times to dilute the concentration of the methane gas in the headings from one concentration to the other under conditions 1 and 2 were investigated. Also, the quantities of fresh air required to dilute the methane gas concentrations under conditions 1 and 2 in one minute in 11 headings studied were also calculated. Table 1 shows the average emission rates of methane gas calculated for heading Nos. 3, 5, 8, 9, 10 and 11. These emission rates of methane gas were far lower than the average methane emission rates for the top 25 coal mines in the United States but approximately the same as the emission rates of some civil engineering tunnels throughout five states in the United States (Grau, 1987).

Analysis and Discussion of Results

The efficiencies of the auxiliary ventilation systems in heading Nos. 9 to 11 were calculated. Table 2 summarizes the quantities of fresh air required to dilute the methane gas concentrations within the stated limits in conditions 1 and 2 at heading Nos. 9 to 11 while Table 3 gives the dilution times under the two conditions at the same headings. Table 4 shows the calculated values of the dilution efficiencies of the auxiliary ventilations systems in heading Nos. 9 to 11 under conditions 1 and 2. From Table 2, the actual quantities of air supplied to the headings were far lower than those required under conditions 1 and 2. This means that the methane concentrations could easily build up to explosive levels if the roadheader is to work continuously at the face. As well, the required purging quantities are much higher in condition 1 in all headings because the methane gas is being diluted over a much wider range in concentration than in condition 2. From the results in Table 3,the average dilution times in condition 1 varied from 3.84 minutes in heading No. 9 to 7.90 minutes in heading No. 10. In condition 2, the average dilution times in the headings were shorter and ranged from 1.46 minutes (heading No. 11) to 3.09 minutes (heading No. 10).

From Table 4, it is noted that the dilution efficiencies vary from 12.83% in heading No. 10 (condition 1) to 138.78% in heading No. 11 (condition 2).Heading No. 11 has an efficiency exceeding 100% indicating that the auxiliary ventilation system provides more air than required to dilute the methane gas to set levels. Thus, it will cost about 1.63 times more per cubic meter of air to ventilate heading No. 11. Generally, the efficiencies obtained may be described as ranging from poor to excellent. This means that the auxiliary ventilation systems were capable of controlling the methane gas concentrations within statutory levels but may not be able to cope with large and unusual methane gas makes in the headings.

Table 1. Average methane gas emission rates for some headings

Heading No.	Methane gas emission rate (m^3/sec.)
3	0.0072
5	0.0293
8	0.0339
9	0.0051
10	0.0067
11	0.0015

Table 2.Quantity of air required to dilute methane gas concentrations in one minute

	Air quantity (m^3/sec.)		
Heading No.	Condition 1	Condition 2	Actual amount supplied
9	26.03	7.35	6.77
10	27.97	10.95	3.54
11	19.38	5.43	4.57

Table 3.Calculated dilution times at various headings under conditions 1 and 2

	Average dilution time (min.)	
Heading No.	Condition 1	Condition 2
9	3.84	1.92
10	7.90	3.09
11	4.24	1.46

Table 4.Dilution efficiencies of auxiliary systems under conditions 1 and 2

	Average dilution efficiency (%)			
Heading No.	Condition 1	Comments	Condition 2	Comments
9	26.28	Fair	89.19	Good
10	12.83	Poor	33.43	Fair
11	24.92	Fair	138.78	Excellent

Conclusions

It can be concluded from the foregoing analysis that the average quantities of fresh air required to dilute, disperse and remove methane gas concentrations within set levels in the development headings studied varied from 5.43 m^3/sec.in heading No. 11 (condition 2) to 27.97 m^3/sec. in heading No. 10 (condition 1). This represented 12% to 26% of the quantities required to dilute methane gas concentrations to safe levels within one minute under condition 1, and 33% to 139% under condition 2.

The average dilution times in the headings studied were generally less than eight minutes and ranged from 1.46 minutes in heading No. 11 (condition 2) to 7.90 minutes in heading No. 10 (condition 1).

The efficiencies of the auxiliary systems in these headings ranged from poor to excellent which meant that the auxiliary ventilation systems were capable of controlling the methane gas concentrations within statutory levels but may not be able to cope with large and unusual methane gas makes in the headings.

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Nomenclature

- A Cross-sectional area of the roadway, m^2
- C Methane gas concentration in the heading, %
- C_{CH_4} Concentration of methane gas emanating from the strata (expressed on a fractional basis)
- C_i Methane gas concentration in normal intake air (expressed on a fractional basis)
- C_r Methane gas concentration in return air, (expressed on a fractional basis)
- G L x A volume of mine opening (the control volume), m^3
- L Distance from the face to the methane gas monitor position outbye, m
- dt Change in time, s
- dC Rate of change in methane gas concentration (C) at time t
- t time elapsed (time for dilution), s
- C_o Methane gas concentration at time t = 0 (expressed as a fraction)
- Q_{CH_4} Quantity of methane gas emanating from the strata, m^3/sec.
- Q_e Quantity of intake air reaching the face of the working, m^3/sec.
- Q_f Total quantity of air through the primary auxiliary fan at nearest through cross-cut, m^3/sec.
- Q_i Quantity of fresh air discharged at the end of the main forcing ducting, m^3/sec.
- Q_l Leakage air quantity through main forcing ducting, m^3/sec.
- Q_{t1} Total intake air quantity in main ventilation airstream, m^3/sec.
- Q_{t2} Quantity of intake air in last through cross-cut which does not pass through the heading, m^3/sec.
- Q_{t3} Total quantity of air in immediate region on the downstream side of the last through cross-cut, m^3/sec.

Q_r Return air quantity in the heading, m^3/sec.

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Appendix — Guide to Practicing Ventilation Engineers

To use the equations developed in this paper to assess the auxiliary ventilation systems in any coal mine heading, the following steps may be taken:

1. Dimensions of roadway: Measure the cross-section of the roadway (i.e., the width, W, and breadth, B, of the heading) and length of the control volume, L (i.e., distance from the face to the methane gas monitor, M2, in the return).Also, measure the distance from the discharge end of the forcing ducting to the face.

2. Volume of roadway or control volume: Calculate G = L x A = L x B x W, m^3.

3. Methane gas concentration: Measure or monitor the methane gas concentrations in the intake air, Ci, at the face of the heading (as measured by M1) and in the return air, Cr (recorded by M2).

4. Methane emission rate:Calculate Q_{CH_4} using equation (10).

5. Methane concentration in return air: Calculate Cr by applying equation (9). Cross-check this value with the average recorded by monitor M2.

6. Methane concentration within control volume or heading: Use equation (12) if high level of accuracy is required or equation (5) if only an approximate value is needed.

7. Time to dilute methane concentration: Use equation (13) if high level of accuracy is required. Assume C_{CH_4} to be equal to reading of M1.Use equation (14) if unsteady conditions prevail. If leakage quantity, Q_1, is very small (i.e., if the auxiliary ventilation ductings are new and well-installed in heading). Apply equation (16) to get an estimate of the dilution time if the methane emission rate into the heading, Q_{CH_4}, is very small compared to the fresh air inflow rate into the heading (i.e., if $Q_i >> Q_{CH_4}$).

8. Fresh air requirements (steady state): Apply equation (17) and assume C_{CH_4} to be the same as that recorded by monitor M1. Apply equation (18) if the methane concentration is assumed to be pure (100%) in concentration.

9. Efficiency of forcing system, E_f: Apply equation (1).

10. Overall efficiency of overlap system, E_o: Apply equation (3). If the distance from the discharge end of the forcing ducting is less than 3 m, assume that Qi = Qe. If the distance to the face is greater than 3 m, extrapolate and prorate for the approximate quantity of fresh air that will reach the face of the heading.

NEWS IN BRIEF

North American / 9th U.S. Mine Ventilation Symposium

The North American / 9th U.S. Mine Ventilation Symposium is to be held June 8-12, 2002, in Kingston, Ontario, Canada. Please visit the official website at: http://mine.queensu.ca/ventilation. Organized by the Underground Mine Ventilation Committee (UVC) of the Society for Mining, Metallurgy, and Exploration (SME), this is the first time that the symposium is being held outside of the United States of America.

The Symposium is geared toward engineers, technicians, supervisors, researchers, educators, students, regulators and manufacturers interested in the field of mine ventilation and environmental control. Technical workshops will be offered on the first two days of the Symposium and an active social program will complement the technical sessions where over 100 papers are expected to be presented. There will be an industrial Trade Show and a Guest Program will also be available for the guests of registered participants. Following the Symposium, industry field trips will also be available to interested participants.

Cost details for Symposium registration, guest registration, workshops and hotels are listed in the "Budget Your Trip" link. The Symposium registration is C$450 (US$296) plus 7% taxes and includes one copy of the proceedings, access to all technical sessions and trade show, coffee breaks, lunches, opening reception, dinner cruise and dinner banquet.

For further information, please contact: Euler De Souza, Chair, or David DeGagné, General Coordinator, North American & 9th U.S. Mine Ventilation Symposium, Department of Mining Engineering, Queen's University, Kingston, Ontario, Canada, K7L 3N6; Tel.: (613) 533-2199; Fax: (613) 533-6597; e-mail: souza@post.queensu.ca or degagne-d@mine.queensu.ca.