# NEW METHODS FOR MORE EFFICIENT USE OF VENTILATION AIR

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## INTRODUCTION

A sufficient and constant supply of fresh air is essential to the health and safety of underground miners. This often requires elaborate ventilation systems that are costly in terms of equipment and man-hours. Moreover, these systems are relatively inefficient since 60 to 80 pct of the fresh air entering a mine often leaks into return airways, never reaching the working sections of the mine (1-2). In addition, much of the air that reaches the working section often short circuits out and never reaches the working face. The U.S. Bureau of Mines has developed some new and unique methods to help combat this inefficient use of mine air.

Ventilation efficiency can often be improved by eliminating major leaks that occur most often at permanent stoppings. However, eliminating air leaks through every permanent stopping in a ventilation system would be an expensive and time-consuming operation. A more realistic approach is to measure the leakage across each suspected stopping and repair only those having a significant leakage.

## THE BRATTICE WINDOW METHOD FOR MEASURING STOPPING LEAKAGE

The Bureau has developed two methods for measuring mine stopping leakage. One of these is called the Brattice Window Method (BWM) (3), which requires erecting a second stopping, called a temporary test stopping (TTS), in the same entry as the leaking permanent stopping. The TTS is made of an impervious fabric, such as plastic mine brattice, and is fastened to the roof, floor, and sides of the entry with spads or similar fasteners. The TTS also will leak, as air will pass through gaps around the edges. A rectangular opening, window 1, is cut into the TTS. The cross-sectional area of this window and the velocity of air passing through it are measured and the volume flow calculated from

$$Q_1 = V_1 A_1,$$

where A = cross-sectional area of window 1, $m^2$,

V = air velocity through window 1, m/s,

and Q = air volume through window 1, $m^3/s$,

Next, a second rectangular opening, window 2, is cut into the TTS. Its area $A_2$ and the velocity of air through it $V_2'$ are measured and used to calculate the air volume $Q_2'$ through it.

$$Q_2' = V_2' A_2$$

The decreased air velocity $V_1'$ through window 1 is also measured, and a new lower air volume $Q_1'$ is calculated from

$$Q_1' = V_1' A_1$$

These values are used in the brattice-window-method equation to calculate the total volume of air $Q_\ell$ in $m^3/s$ passing through the permanent stopping as follows:

$$Q_\ell = C \left[ Q_1' + Q_2' + \frac{Q_1' + Q_2' - Q_1}{V_1/V_{1'} - 1} \right].$$

The window coefficient (C) is necessary because of the vena contracta created by the airflow through the windows.

Preliminary tests by the Bureau proved that the BWM works and is practical for in-mine use. The Bureau then contracted Goodyear Aerospace Corp. (GAC) to further examine and improve the BWM. This work was first conducted in a GAC laboratory and later in three coal mines in West Virginia.

## LAB STUDIES

GAC's laboratory efforts included building the equipment and testing the Bureau of Mines Brattice Window Method in a simulated cross-cut that is 6.4 m long with a constant cross-section of 1.5 x 5.5 m. A diffusing cloth was mounted in the cross-cut to simulate a leaking permanent stopping. Outby this, the TTS was installed. A simulated entryway was added in front of the cross-cut to determine the effect of cross flow velocity or swirl on the BWM.

The total flow quantities were obtained by calculating from pitot measurements the flow into or out of the cross-cut and considering cross-cut leakage associated with the test conditions. The corrected values gave the true flow quantities through the TTS windows. These corrected flow quantities were then used with the flow quantities determined by anemometer readings taken at the TTS windows to calculate the window coefficients.

The results from these tests indicate the window coefficients are nearly constant for a given window size for corrected anemometer air flow velocities through the window of 0.5 to 5.0 m/s. The average values for the window coefficient for a velocity range of 0.5 to 5.0 m/s measured downstream and upstream of the windows are:

| Window Geometry, meters | Average Coefficient Value (C) Measurement Position Relative to TTS Downstream | Upstream |
|---|---|---|
| 0.10 dia. | 0.62 | 0.68 |
| 0.15 x 0.30 | 0.64 | 0.67 |
| 0.30 x 0.30 | 0.66 | 0.715 |
| 0.30 x 0.60 | 0.69 | 0.78 |
| 0.60 x 0.60 | 0.73 | 0.775 |

The effect of cross flow on the total flow values calculated by the Bureau of Mines Brattice Window Method Equation was determined by tests with the TTS in three different locations relative to the entryway with and without flow in the entryway. The results indicated that small errors are present when the TTS is 3.6 m from the entryway. The maximum error approaches 10 pct as the TTS is moved closer to the entryway.

## COAL MINE STUDIES

From the consistent laboratory results, it was concluded that the Bureau of Mines method was useful over a large range of values for measuring leakage past permanent stoppings.

This method was then applied by measuring leakage past permanent stoppings in the Bethlehem Mines Corporation operating mines 105 East, 105 West, and 108 near Bridgeport, West Virginia. The sizes of the cross-cuts ranged from 1.4 m x 4.9 m to 1.8 m x 6.4 m.

The following conclusions are indicated based on the results from the tests in Bethlehem Mines Corporation mines 108, 105 East, and 105 West.

1. Lightweight equipment and erection techniques were developed so that two

miners were able to use the technique for quickly obtaining the leakage past individual permanent stoppings.

a. The total weight of the equipment carried by the two miners for these tests was 18.6 to 20.2 kg. Each miner carried approximately one-half of the total weight.

b. The average time period for the miners to erect the TTS, seal it, take flow readings, and move the TTS and equipment to the adjacent cross-cut was 18 min for the tests of the 10 consecutive individual stoppings in mine 105 W. Time periods to erect the TTS, seal it, take readings, and move the TTS and equipment to an adjacent cross-cut ranged from 20 to 45 min in the other mines where the roofs were composed of scaling rock.

2. Measurements were made in an airway before and after nine stoppings. In this airway location, the flow readings were repeatable, and three values were read by each of the three different investigators. The average value for the measured difference in airway flows was 2.02 $m^3/s$, which compares with the 2.41 $m^3/s$ value for the sum of the leakages past the mine stoppings. The apparent error of 0.39 $m^3/s$ can be compared with the magnitude of the airway flow values, which were approximately 11.18 $m^3/s$ and 13.21 $m^3/s$, respectively. A plus and minus error of less than 2 pct in calculated airway flow values can account for this apparent error. Based on this comparison, accurate measurements at spaced locations in the airway can be used to locate regions in the mine where the leakage past individual stopping should be measured using the Bureau of Mines Brattice Window Method.

## TRACER GAS METHOD FOR MEASURING LEAKAGE THROUGH STOPPINGS

The Bureau of Mines has also developed a tracer gas method for measuring mine stopping leakage (4). To use this method, a temporary brattice is hung about 3.0 m from the permanent stopping. A tracer gas is released in the air volume V between the brattice and stopping, and the tracer gas concentrations are measured periodically for 60 min. A semilog plot of the concentration measurements are then used to deduce the air leakage $Q_\ell$. If the leakage air has a contaminate gas concentration greater than that initially measured in the test volume, addition of the tracer gas can be omitted. In this case, the contaminate gas can be used as a tracer, provided that the temporary brattice is hung on the low-pressure side of the leak. Air leakage is then evaluated from concentration measurements of the contaminate gas in the test volume.

The change in the tracer gas concentration C in the fixed volume between the two stoppings as a function of time t is given by

$$VdC = Q_\ell C_\ell dt - Q_\ell C dt, \tag{1}$$

where $Q_\ell$ is the stopping air leakage, which is assumed to mix well, and $C_\ell$ is the tracer gas concentration in the leakage air. For constant $Q_\ell$ and $C_\ell$,

$$C = C_\ell - (C_\ell - C_0) e^{-Q_\ell t/V}, \tag{2}$$

where $C_0$ is the initial tracer gas concentration within the volume.

Equation 2 was used to determine the amount of air (leakage) purging a volume for the two following cases:

Case 1. -- Tracer gas is released within test volume. The initial tracer gas concentration within the volume is $C_0$, and the concentration of the purging air is zero ($C_\ell = 0$). Equation 2 then becomes

$$C = C_0 e^{-Q_\ell t/V}. \tag{3}$$

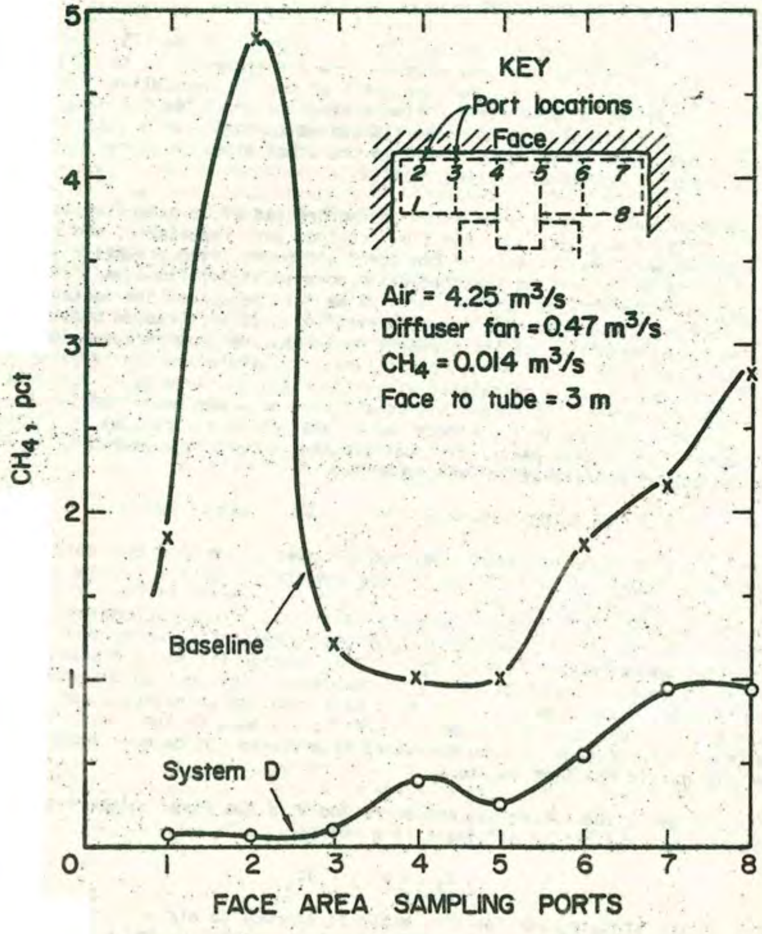


FIGURE 1. Face area methane concentrations for the baseline and the best diffuser fan system.

A semilog plot of the concentration versus time yields a straight line with a slope of $-Q_\ell/V$. The leakage is computed by multiplying the negative of the slope by the volume. Since the slope is $(\ln C_2 - \ln C_1)/(t_2 - t_1)$,

$$Q_\ell = -\left[\frac{\ln C_2 - \ln C_1}{t_2 - t_1}\right] V, \tag{4}$$

$$= -2.303\left[\frac{\log C_2 - \log C_1}{t_2 - t_1}\right] V. \tag{5}$$

Case 2.--A contaminate gas is used instead of a tracer gas. If the concentration of the contaminate gas in the leakage air is greater than its initial concentration in the enclosed volume ($C_\ell > C_0$), then this gas can be used as a tracer. In this case, the temporary brattice must be hung on the low-pressure side of the permanent stopping. Equation 2 can be rewritten as

$$C_\ell - C = (C_\ell - C_0)\, e^{-Q_\ell\, t/V}. \tag{6}$$

A semilog plot of $C_\ell - C$ versus time yields a straight line with a slope of $-Q_\ell/V$. The stopping leakage is obtained as before by multiplying the negative of the slope by the volume.

A series of underground leakage tests were conducted in the Bureau's Experimental Mine at Bruceton, Pa. A small fan conducted air through a 0.15-m-diameter pipe to the face of a dead heading to simulate a known leak. A damper and pitot tube were used to regulate and measure the leak. The air volume blown into the heading ranged from 0.007 to 0.125 $m^3/s$ as determined by pitot tube measurements. To disperse the leakage, a section of the burlap was fastened to the coal face over the exhaust end of the pipe. The temporary brattice was hung about 3.0 m outby the face of the dead heading. It was hung so that no observable gaps existed. The enclosed volume was about 22.1 $m^3$. A section of tygon tubing was suspended from the roof for sampling. One end of the tube hung in the center of the enclosed volume; the other was connected to a hand-held pump outside of the test volume.

The tests showed that it is possible to measure air leakage through stoppings as low as 0.007 $m^3/s$, using a tracer gas. Calculated values are within 10 pct of the actual leakage. If the leakage air has a constant contaminate gas content greater than that initially measured in the test volume, the addition of a tracer gas is not needed because the contaminate gas can serve as a tracer.

## OPTIMIZED DIFFUSER AND SPRAY FAN SYSTEMS FOR FACE VENTILATION

Research sponsored by the Bureau of Mines has shown that working face ventilation can be greatly improved by devices that create a strong sweep of air across the face in the same direction as the normal airflow (5).

The diffuser fan is a proven method for reducing methane at coal mine working faces. However, no systematic investigation of diffuser fan performance had been made previous to the study described here. The general approach has been to direct fresh air in the general direction of the face, and to leave it at that. In 1973, the Bureau of Mines awarded Foster-Miller Associates a contract to optimize the diffuser fan. The initial objective was to determine the best airflow and best location for the fan outlet, using full-scale plywood models of a mine passageway and a mining machine with a rotating ripper-type head and water sprays. However, as the investigation proceeded, it developed that other, more novel approaches to face ventilation could perform as well as, or better, than the diffuser fan.

| | | $m^3/s$ | 7.0 $kg/cm^2$ water, ℓ/s |
|---|---|---|---|
| Spraying system BD5-5 | Hollow cone | 0.061 | 0.082 |
| Bete WL 1/4 80 | Solid cone | 0.033 | 0.025 |
| Conflow No. 1 nozzle | Venturi | 0.057 | 0.088 |

FIGURE 2. Airflow induced by three typical water sprays.

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## BASELINE TESTING

The initial problem in the contract was to evaluate current face ventilation and establish how much room for improvement existed. For this purpose, Foster-Miller conducted an extensive series of baseline tests using different machine configurations, and varying airflows, methane flows, and brattice-to-face distances; tubing was used in addition to brattice.

Methane was released through manifolds that stretched across the face and back along the first few meters of the rib. Measurements were made 0.3 m below the roof, down the sides and in some tests, directly under the cutter head. Locations around the front of the mining machine represent FVE (face ventilation effectiveness) measurements points, which we are using as our indicator of how well the face is being ventilated.

The FVE is the methane concentration in the return divided by the average concentration at the face, as measured at the FVE points. Since the FVE measurements are 0.3 m from the face, the FVE provides an index of what fraction of the total air is getting up to the last 0.3 m.

How much air does actually reach the last 0.3 m? Under most conditions the percentage is not high. With 1.42 $m^3/s$ of air exhausting through a vent tube located 6 m from the face, the FVE is 16 pct. Thus only 16 pct of 1.42 $m^3/s$ reaches the last 0.3 m. A large area of over 2 pct methane results with a methane flow of 0.014 $m^3/s$. With 4.25 $m^3/s$ exhausting from a tube located at 3.0 m. the FVE is still not good and some regions still have over 2 pct methane.

## DIFFUSER FAN TESTING

The actual diffuser fan study considered a number of different designs.

Figure 1 is an indication of what the best design, called system D, will do. The total air is 4.25 $m^3/s$ exhausting through a tube at 3.0 m. The methane emission is 0.014 $m^3/s$. With no diffuser fan (the baseline) every point 0.3 m from the face is 1 pct methane or greater. With system D diffuser fan every point is below 1 pct and the average methane concentration was considerably reduced.

The same kinds of results were obtained with other machine positions, other tube-to-face distances, other methane flow rates, and with brattice. We felt that our diffuser fan studies had been quite successful.

## WATER SPRAY TESTING

During the course of the diffuser fan program, the test plan called for turning the water sprays on and off to see how they were affecting the face ventilation. In some machine positions, particularly at the beginning of the slab cut, activating the sprays substantially reduced the methane. Air was being entrained in the sprays and this was bringing fresh air, carrying away the methane.

So, we added a new series of tests to systematically investigate the air-moving ability of water sprays and the improvement in face ventilation that might result from some modified placement of the sprays.

We started with water sprays in a test chamber. Dozens of different nozzles were tested. The results from three representative nozzles are shown in figure 2. Note the venturi does not move more air than an ordinary spray. It is not necessary to confine the spray in a small tube to make it an effective air mover. It is important to recognize that the space between the mining machine and the roof or rib can also act as a tube.

In designing the water spray system, we were able to take advantage of what we learned from the diffuser fan. First, we had to entrain fresh air; second, we had to sweep it across the face.

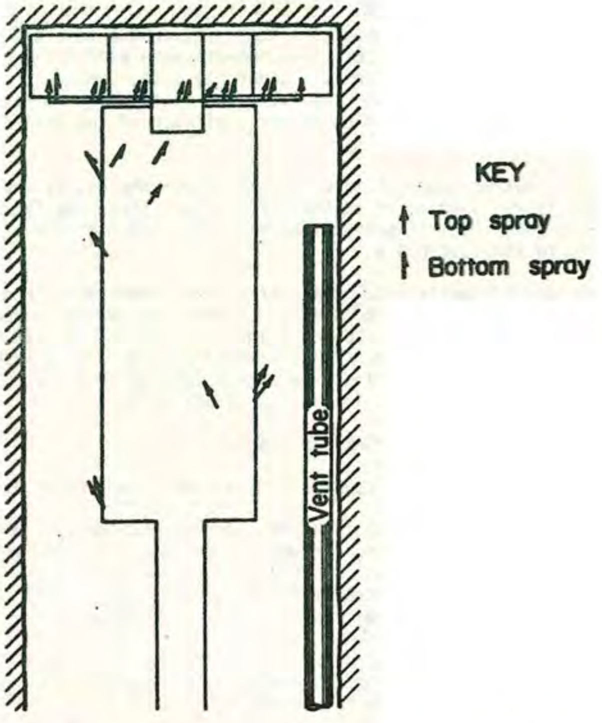


FIGURE 3. The best modified water spray system.

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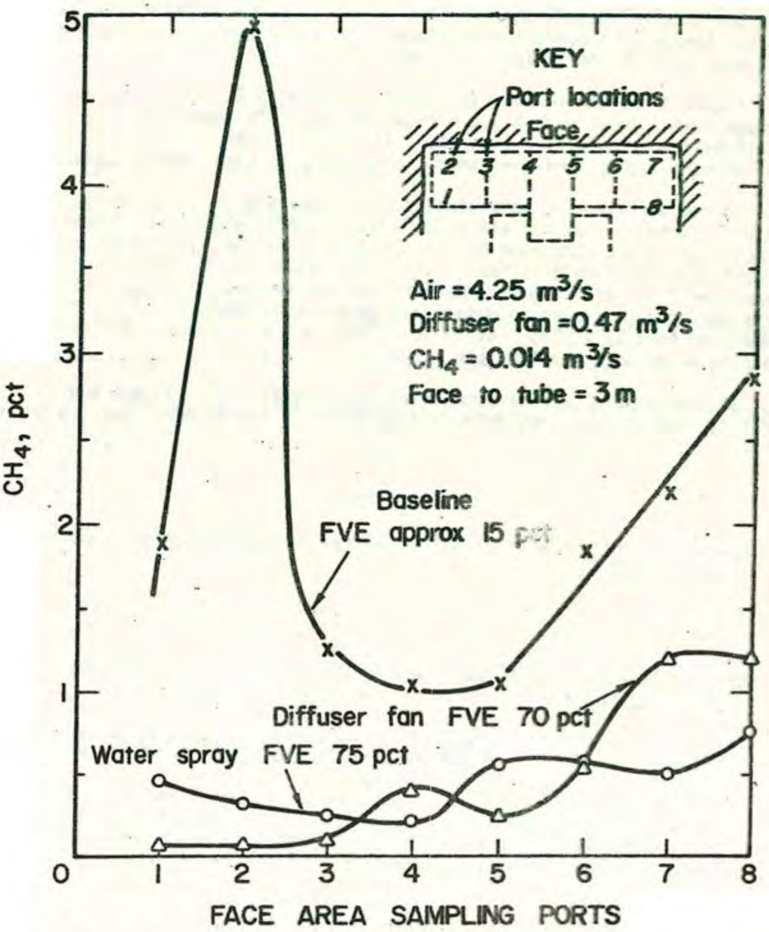


FIGURE 4. Effectiveness of the diffuser fan and modified water spray systems as compared with the baseline under good ventilation conditions -9,000 cfm exhausting through a tube at 10 feet.

The final design (fig. 3) has three key features: First, the sprays at the cutter head are turned 10° to the right. Second, sprays are mounted on the left side of the machine to catch the fresh air at the back of the machine and move it up along the left side to the face. Third, a few sprays mounted on the right side keep the swirl of dust-laden air ahead of the operator. Water consumption is no greater than with a conventional system, and the system still functions reasonably well when some of the sprays are clogged.

Figure 4 represents one case investigated with 4.25 $m^3/s$ air exhausting through a vent tube located 3.0 m from the fans. The modified water spray system is slightly more effective than the 0.47-$m^3/s$ diffuser fan and this also holds for the two-step ripper configurations. Both are five times better than the baseline.

In conjunction with both studies, one important question we had to answer was how well these systems operate under adverse conditions. What decrease in efficiency takes place when some of the sprays clog or the water pressure drops? What happens when the line brattice is at 9.1 m, instead of 3 or 6 m? Or, if the region around the front of the mining machine becomes abnormally clogged with broken coal? Tests were run to answer in detail all of these questions, and the results can be summarized here by saying that it was always better to have a diffuser fan or modified water spray system than to have nothing at all.

In conclusion, we have found that both the diffuser fan and modified water spray systems produce considerable reductions in the methane level at the working face.

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# REFERATI,ДОКЛАДИ, PROCEEDINGS,

# REFERATS, BERICHTE

# C, D, E, F, G

SFR JUGOSLAVIJA
Cavtat (Dubrovnik)
7—14. oktobra 1979.