MECHANISM OF REGRINDING IN ROTARY CUTTING MACHINES

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ABSTRACT

This paper presents an analysis of mechanism of secondary fragmentation and respirable dust generation due to regrinding. In continuous miners, the recovered coal and uncleaned coal from the previous cutting cycle have to be transported through the circumference of the cutting drum, subjecting the coal to regrinding. Such regrinding comminutes the coal material utilizing kinetic energy, thus creating a great amount of respirable dust. The mechanism of a single particle subjected to regrinding is analyzed. A mathematical model is developed to describe regrinding in field conditions. The important factors influencing the comminution process are discussed in detail. Utilizing three types of coal seams, laboratory experiments are being carried out with an Automated Rotary Coal Cutting Simulator (ARCCS) to quantitatively evaluate the effects of some particular parameters, such as cutting velocity, depth of sump and properties of coal material, on breakage and on the amount of respirable dust generated. An orthogonal fractional factorial experimental arrangement is used to optimize experimental design. Preliminary experimental results are presented.

INTRODUCTION

It is well recognized that reducing the amount of dust generated at the source is the most authentic approach to respirable dust control. Coal cutting is a major dust source in coal mines. During cutting a small percentage of the generated dust becomes airborne, while most of the generated dust remains in a setting state or adheres to the large particles. This adhered dust becomes airborne during loading and transportation. Reducing the amount of respirable dust generated at the source not only decreases the dust concentration during cutting but also lessens the amount of

dust liberated during loading and transportation. In recent years, a number of researchers have studied the mechanism of coal fragmentation and dust generation. Studies have shown that under the action of the cutting tool, three damage zones, namely, crushing zone, chipping zone and fracturing zone, will be created in the coal seam. Majority of the dust is from the crushing zone, which is right around the bit tip. Many parameters, such as cutting depth, cutting velocity, bit geometry and bit-mounting configuration, properties of coal material and in-situ condition, will significantly affect the characteristics of the damage zones. A great achievement on improving cutting head and bit design and selecting proper operating parameters to reduce the dust generation has been obtained. Many of researchers also found that all the broken coal and part of the remaining coal from previous cutting cycles must be transported through part or entire circumference of the cutting drum. This transportation of already recovered coal through the cutting zone of rotary head makes it a rotary grinder. Some of the coal particles will be subjected to secondary fragmentation or regrinding, The potential effect on dust generation by regrinding may be at least as much as that produced by primary fragmentation. The objective of this study is to analyze the mechanism of secondary fragmentation, quantify the amount of dust generated by regrinding, evaluate the effects of some major parameters, such as cutting velocity, depth of sump and coal properties, on dust generation due to regrinding.

SINGLE PARTICLE SUBJECTED TO REGRINDING

The feature of regrinding is that the moving bits and bit blocks on the cutting drum impact coal particles and transfer some of their kinetic energy to the particles. Then, moving coal particles are free to impact one

another and they also impact the intact coal seam or drum body, so that some of kinetic energy of the cutting system is transformed into strain energy. Internal stresses are created in the particles. When the internal stresses reach failure limit, fractures will initiate and develop and then the particles break. The stored strain energy is released. Therefore, a great amount of respirable dust will be generated.

If energy transformation were perfect, the relationship between kinetic energy and strain energy for a single particle would be

$$\frac{mv^2}{2} = \iiint_Q \frac{\sigma^2}{2E} dQ, \qquad (1)$$

where m is mass of the particle, v is moving velocity, E is Young's modulus, Q is volume of the particle and σ is internal stress.

Required kinetic energy to break the particle is

$$\frac{mv^2}{2} = \iiint_Q \frac{\sigma_*^2}{2E} dQ, \qquad (2)$$

where σ_* is internal stress distribution under critical condition. Let allowable strain energy be

$$W_a = \iiint_Q \frac{\sigma_*^2}{2E} dQ, \qquad (3)$$

Then, critical velocity to break a particle can be obtained

$$v_* = \sqrt{\frac{2W_a}{m}}, \qquad (4)$$

It is obvious that whether or not a particle will be broken depends on allowable strain energy and kinetic energy obtained by the particle. Allowable strain energy is a function of mechanical properties of the particle and influenced by the size and geometry of the particle. Kinetic energy is determined by mass and moving velocity. For a given particle, moving velocity is the key parameter. Critical velocity v_* or allowable strain energy W_a can be determined in laboratory. Figure 1 illustrates a simple experiment. When two irregular coal
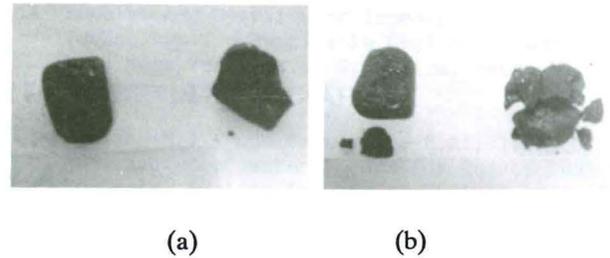

(a)   (b)

Figure 1 Coal particles impacting concrete floor
(a, v=4.32 m/s; b, v=4.71 m/s)

particles, Waynesburg coal seam, impacted concrete floor at a moving velocity of 4.32 m/s, neither particles were broken in spite of little chipping on the contact area. However, when the velocity increased to 4.71 m/s, both particles were broken into small particles.

Critical velocity can also be determined by internal stress distribution analysis. First, the magnitude and distribution of impact-induced surface pressure on contact area are determined. Then, analytical or finite element techniques are used to analyze internal stresses. Selecting proper failure criteria, critical velocity can be calculated. Due to space limitation, this work is not included in this paper.

REGRINDING IN FIELD CONDITION

In the field, regrinding problem is very complicated because of the following facts: First, the particles to be ground are the result of primary cutting. In other words, the feed is a combination of particles with widely varied size. Second, not all particles are subjected to secondary fragmentation during grinding. One particle may obtain different moving velocity from another. Some of them are broken while the others just pass through the cutting zone without breakage. Therefore, breakage for any particle is only possible. Third, the product from a particle may fall into any size range. Figure 2a demonstrates feed size distribution and product size distribution. In order to quantify the relationship between distributions, the feed and product size are divided into a number of subranges Δx_1, Δx_2, ..., Δx_n. assume that product size distribution and feeding size distribution in weight are $p(\Delta x_1)$, $p(\Delta x_2)$, ..., $p(\Delta x_n)$ and $f(\Delta x_1)$, $f(\Delta x_2)$, ..., $f(\Delta x_n)$ respectively, simply called p_1, p_2, ..., p_n and f_1, f_2, ..., f_n as shown in Figure 2b. For a narrow size range of feed (f_j), the probability of being broken is s_j. That means that the fraction ($s_j f_j$) of the
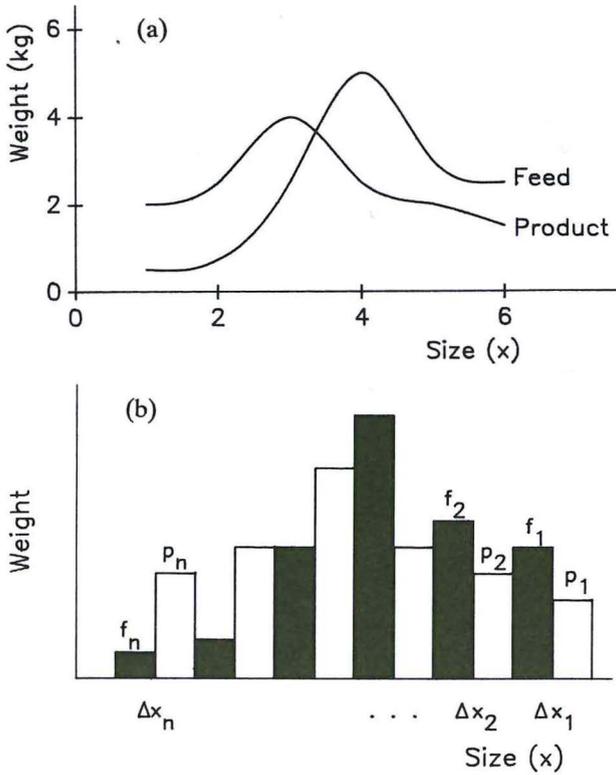
Figure 2 (a and b) Feed and product size distributions in weight

feed (f_j) will be broken and the others (($1-s_j$)f_j) will pass through cutting path without breakage. The product from $s_j f_j$ has some probability (b_{ij}) of falling into the (Δx_i) size range, making the contribution of p_{ij} to p_i.

Therefore, total amount of broken material P_b is expressed as Equation (5)

$$P_b = \sum_{j-1}^{n} s_j f_j . \tag{5}$$

The total product is

$$P_t = \sum_{j-1}^{n} s_j f_j + \sum_{j-1}^{n} (1-s_j) f_j . \tag{6}$$

And the product distribution is expressed in Equation (7)

$$\{P\} = [B]*[S]*\{F\} + ([I]-[S])*\{F\}, \tag{7}$$

where $\{P\} = \{p_1, p_2, ... , p_n\}^T$,

$\{F\} = \{f_1, f_2, ... , f_n\}^T$,

$[B] = b_{ij}$, (i,j=1, 2, ... , n) is called breakage function,

$[S] = s_{ij}$, (i,j=1, 2, ... , n), $s_{ij} = 0$ when $i \neq j$, and is select function,

$[I]$ is unit matrix.

Let $[C] = [B]*[S] + [I]-[S]$

$$\{P\} = [C]*\{F\}. \tag{8}$$

Equation (8) relates the product size distribution to feed size distribution In matrix form. In the equation, $\{F\}$ is feed distribution and can be determined by studying the results from primary cutting. $[C]$ is correlation matrix. It depends on selection function $[S]$ and breakage function $[B]$ and is influenced by a number of parameters:

Physical and mechanical properties of coal material. Equation 2 shows that the properties of coal material determine the critical condition, thus influencing the selection function, and also effecting product or breakage function. Figure 3 shows that impacted Lower Kittanning coal particles were broken into a number of small and fine products, while Waynesburg coal particles were broken into large size and a little fine products. The properties of three types of coal seam are listed in Table 1 and Table 2.
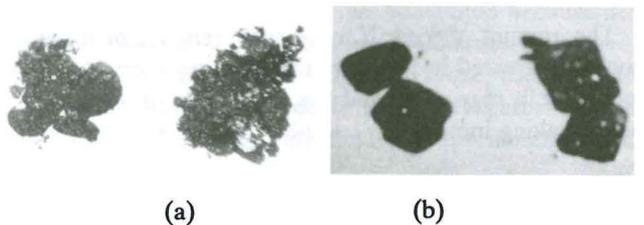

(a) (b)

Figure 3 Lower Kittanning and Waynesburg coal particles impact concrete floor (a, Lower Kittanning coal; b, Waynesburg coal)

Cutting velocity. Higher cutting velocity transforms more kinetic energy into strain energy, enabling more particles to build up high internal stresses so that

TABLE 1 Physical properties of coal seams

Type of coal	Moisture %	Ash %	Sulfur %	Rank	Hardgrove grindability	Sp.Gr.
Waynesburg	0.0-1.5	13.5-13.7	2.03-2.06	hvBb*	50.43	1.42
Pittsburgh	0.0-1.1	9.70-9.80	3.44-3.48	hvAb**	60.38	1.30
Lower Kittanning	0.0-1.0	8.40-8.50	0.93-0.94	hvAb	69.09	1.31

* High volatile B bituminous, ** High volatile A bituminous.

TABLE 2 Mechanical properties of coal seams

Type of coal	Orientation	Compressive strength (kg/cm^2)	Young's modulus (kg/cm^2)	Poisson's ratio	Indirect tensile Strength (kg/cm^2)	Direct shear strength (kg/cm^2)
Waynesburg	Face cleat	329.57	6,186.40	0.33	10.83	14.34
	Butt cleat	227.35	8,154.80	0.28	14.41	12.65
	Bed plane	520.22	3,515.00	0.47	10.26	5.62
Pittsburgh	Face cleat	181.66	2,179.30	0.26	8.65	6.82
	Butt cleat	198.04	3,655.60	0.29	12.79	9.98
	Bed plane	280.08	4,288.30	0.27	5.41	8.93
Lower Kittanning	Face cleat	236.56	3,725.90	0.22	7.73	5.55
	Butt cleat	143.27	4,007.10	0.28	8.93	6.19
	Bed plane	191.78	2,741.70	0.31	6.26	8.93

selection function [S] increases and finer particles are produced.

Size of feed. Although large size particles can store more strain energy, they have a lower allowable unit strain energy (W_a/V) because of size effect. Also, in the cutting system, large particles have a higher opportunity to contact with bit than small ones.

The amount of feed. If more coal remains or more coal is recovered in primary cutting, more particles are including in regrinding and the probability of impact and breaking increases.

Depth of sump. Depth of sump determines the length of arc cut. A longer arc cut makes coal particles grind for a longer time, so that there is more breakage and more dust is generated.

Since secondary fragmentation is a probability, correlation matrix [C] can only be determined statistically through laboratory experiment or field investigation. In order to estimate the effect of every factor on correlation matrix in detail, a laboratory experiment program is carried out in this study.

LABORATORY EXPERIMENT ON REGRINDING

Experimental facilities

An Automated Rotary Coal Cutting Simulator (ARC-CS) is used to simulate experiment of regrinding in laboratory. It was originally designed with the capability to study different machines and in-situ parameters that influence on fragmentation of coal and resulting dust generation. In order to meet the need of regrinding simulation, the ARCCS is modified with: 1) A cowl to hold a coal block at the bottom of the confining chamber. This cowl has built in hydraulic cylinders to advance the coal block upwards to simulate the downward movement of cutting head; 2) A hopper-like feeder to be used in feeding the predetermined size and amount of coal particles on to the entire width of cutting drum to simulate the recovered coal from primary cutting; 3) An enclose to completely cover up the cutting head which enable investigator to collect airborne dust. Figure 4 shows modified ARCCS.
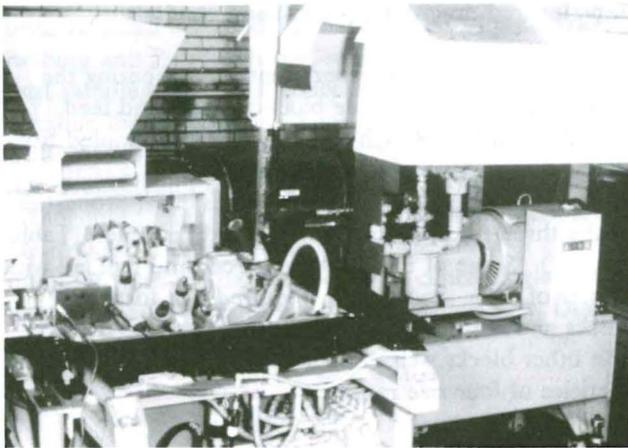
Figure 4 Modified ARCCS

Experimental design

As mentioned before, the feed is the material from primary cutting. If the feed distribution $(f_1, f_2, ... , f_n)$ is known, in order to determined correlation matrix $[C]$, only one size range f_j is fed into the cutting system at a time. If the product from f_j is p_{ij} (i=1,2,...,n), Then, c_{ij} can be determined by taking the ratio of p_{ij} to f_j. For example, as shown on Table 3, $c_{21} = p_{21} / f_1$.

TABLE 3 Product yielded by feed f_j

Feed	f_1	f_2	...	f_n
Product	p_{11} p_{12} ... p_{1n}	0 p_{22} ... p_{2n}	... ...	0 0 ... p_{nn}

However, c_{ij} will vary with any change in coal properties, depth of sump, cutting velocity or rpm (revolution per minute) and the amount of feed. In order to relate c_{ij} to those influencing factors, a factorial experiment has been conducted. Three levels for each factor were taken and codes 0,1,2 were used to represent the first level, the second level and the third level respectively. The corresponding value of all factors are listed in Table 4.

TABLE 4 Level values of influencing factors

Level code	0	1	2
Type of coal	W	P	K
Depth of sump (cm)	5	10	15
Cutting velocity (v/s)	0.64	0.90	1.15
The amount of feed (kg/min)	13.6	22.7	31.8

W-Waynesburg coal, P-Pittsburgh coal, K-Lower Kittanning coal

If all possible parametric combinations are considered, the total number of combinations is 81. Under each combination, four size ranges are used, yielding 324 experiment runs. Since so many experiment runs will be time consumption, in this study, an orthogonal fractional factorial experimental design is used. This kind of design permits the estimation of all relevant effects with zero correlation. In the design, only a fraction of all possible combinations are taken. Table 5 shows the parametric layout. The number of parametric combinations is 18. Four size ranges (3.81-3.18cm, 3.18-2.54cm, 2.54-1.91cm, 1.91-1.27cm) of particles were fed under each combination. The amount of feed is f_1, f_2, f_3, f_4 respectively, thus yielding 72 experiment runs.

In the experimental design, the levels of a factor are arranged in one row. Any two rows are orthogonal to each other. There are two main reasons for considering orthogonal fractional factorial design. First, the experimental data are extremely simple to analyze because the level distribution is diagonal, easily invertible. Second, the design is optimal in a very wide sense, especially in statistical justification since it lets investigators obtain most useful information with the least experimental work.

TABLE 5 Experimental design

Parameter	Parametric combination																	
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Type of coal	0	0	0	1	1	1	2	2	2	0	0	0	1	1	1	2	2	2
Depth of sump	0	1	2	0	1	2	0	1	2	0	1	2	0	1	2	0	1	2
Cutting velocity	0	1	2	1	2	0	2	0	1	2	0	1	0	1	2	1	2	0
Amount of fee	0	2	1	1	0	2	2	1	0	1	0	2	2	1	0	0	2	1

These features can be seen in experimental data analysis. Take any c_{ij}, for instance, c_{11}. 18 c_{11}s will be obtained under 18 parametric combinations. That is $c_{11}{}^1$, $c_{11}{}^2$, ..., $c_{11}{}^{18}$. Table 6 shows how to estimate the effect of cutting velocity on c_{11}.

TABLE 6　Data analysis to relate the dependent to independents

Cutting velocity level	Sum of $c_{11}{}^i$ yielded by the combinations with cutting velocity level 0, 1, 2 respectively	Mean $c_{11}{}^{(l)}$
0	$S_{11}{}^{(0)} =$ $c_{11}{}^1 + c_{11}{}^6 + c_{11}{}^8 + c_{11}{}^{11} + c_{11}{}^{13} + c_{11}{}^{18}$	$c_{11}{}^{(0)} =$ $S_{11}{}^{(0)}/6$
1	$S_{11}{}^{(1)} =$ $c_{11}{}^2 + c_{11}{}^4 + c_{11}{}^9 + c_{11}{}^{12} + c_{11}{}^{14} + c_{11}{}^{16}$	$c_{11}{}^{(1)} =$ $S_{11}{}^{(1)}/6$
2	$S_{11}{}^{(2)} =$ $c_{11}{}^3 + c_{11}{}^5 + c_{11}{}^7 + c_{11}{}^{10} + c_{11}{}^{15} + c_{11}{}^{17}$	$c_{11}{}^{(2)} =$ $S_{11}{}^{(2)}/6$

The significance of the factor can be judged by variance analysis. The variability of cutting velocity is

$$SS_v = \frac{1}{6}\sum_{l=0}^{2}(c_{11}^{(l)} - \frac{1}{3}\sum_{l=0}^{2}c_{11}^{(l)})^2. \qquad (9)$$

Total variability is

$$SS_t = \sum_{i=1}^{18}(c_{11}^i)^2 - \frac{1}{18}(\sum_{i=1}^{18}c_{11}^i)^2. \qquad (10)$$

And the residual is

$$SS_e = S_t - \sum_{j=1}^{4}S_j. \qquad (11)$$

Table 7 presents the information to judge significance of any factor.

TABLE 7　Variance analysis

SS	df	MS	F
SS_v	2	$MS_v = SS_v/2$	MS_v/MS_e
SS_t	17		
SS_e	9	$MS_e = SS_e/9$	

df-Degree of freedom, MS-Mean squares, F-statistic.

Give significance level α. If $F > F_\alpha$ the factor is significant. Otherwise, it is not significant.

Experimental procedure, preliminary results

The first task of the experiment was preparing the coal samples which include block samples and feed particle samples. Rough coal blocks were sampled from Waynesburg, Pittsburgh, and lower Kittanning coal seams respectively. Physical and mechanical properties of the three types of coal seam were determined (Table 1 and 2). Some of the blocks were cut and molded with plaster of Paris according to required size for front blocks and bottom blocks to be set up in the ARCCS. The other blocks were broken and sieved to obtain feed particles of four size ranges.

After sample preparation had been completed, the front and bottom blocks were set up as shown in Figure 5. The blocks were cut to create certain depth of sump. A predetermined rpm and a feed velocity were set according to experimental design. The machine was run and then the particles of f_1, f_2, f_3, f_4 were fed separately. In each parametric combination the amount of feed was predetermined. Therefore, $f_1 = f_2 = f_3 = f_4$. For each feed, three cascade impactors were positioned at top, middle and bottom of cutting head horizons to sample the dust. The product was collected for size distribution analysis. Then, another parametric combination was taken and another group of feeds was run. Experiments on Waynesburg and Pittsburgh coal seams have been completed. Experiments on lower Kittanning coal seam are in progress at the time of preparation of this paper.

Due to the nature of orthogonal experimental design, a complete parametric analysis can be done only after all the experiments on three types of coal seam are
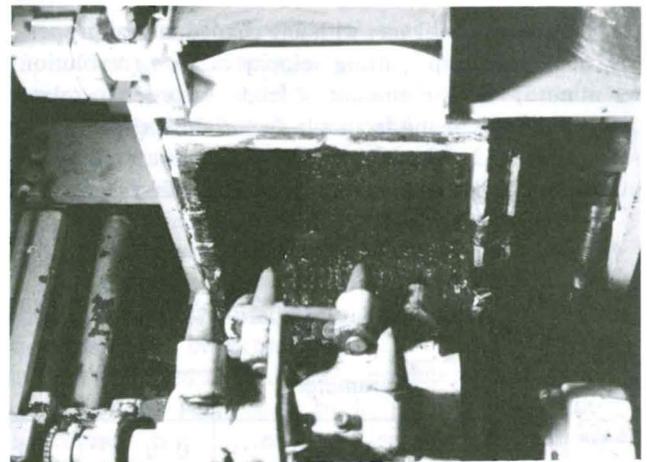

Figure 5　Experimental set up

conducted. However, some preliminary evidence has been revealed based on the experiments of both Waynesburg and Pittsburgh coal seams. Table 8 presents dust sample data. Dust concentration D_{cj} is caused by grinding the particles, f_j. Though the 12 combinations which have been conducted is not a complete orthogonal arrangement, the row representing type of coals is orthogonal to any of the others. Therefore, it is possible to make a direct estimation of the effect of the different types of coal seam on dust concentration. Let us take middle cascade sample data to analyze. Dust concentration, D_c, caused by regrinding {F} can be expressed as

$$D_c = c_{d1} f_1 + c_{d2} f_2 + c_{d3} f_3 + c_{d4} f_4$$
$$= \{C_d\} \{F\}^T \tag{12}$$

where $\{C_d\} = \{c_{d1}, c_{d2}, c_{d3}, c_{d4}\}$ is coefficient vector, c_{dj} presents dust concentration caused by grinding unit weight of particles in the jth size range. Therefore,

$$c_{dj} = \frac{D_{cj}}{f_j}. \tag{13}$$

Table 9 shows c_{dj} values in the middle of the cutting head horizon under the 12 parametric combinations. A direct estimation of the effect of the Waynesburg coal and the Pittsburgh coal is made as shown in Table 10. If unit weight of large particles are ground. The Pittsburgh coal caused higher dust concentration than the Waynesburg coal. However, this evidence is not found in grinding the small coal particles. Figure 6 illustrates the relationship between dust concentration coefficient and feed size for two types of coal.

Equation 14 is a typical formula of the relationship between product and feed size distributions obtained under a particular parametric combination of Waynesburg coal, depth of sump = 2.5 cm, v = 1.15 m/sec and the amount of feed = 22.7 kg/min. It seams that regrinding smaller size material produces more dust and fine particles. Figure 7 shows the relationship between fine product index (c_{sj}) and feed size. This analysis is based on single parametric combination, hence, it can not be justified statistically.

TABLE 8 Dust sample data (mg/L)

Combi-nation	Feed range	Top cascade	Middle cascade	Bottom cascade
0000	1	0.083	0.183	0.300
	2	0.133	0.283	0.500
	3	0.183	0.300	0.667
	4	0.367	0.533	1.067
0021	1	0.217	0.367	0.600
	2	0.333	0.533	0.667
	3	0.450	0.617	0.833
	4	0.566	0.667	0.933
0112	1	0.350	0.433	0.733
	2	0.533	0.567	1.133
	3	0.550	0.893	1.533
	4	1.033	1.150	2.033
0100	1	0.150	0.267	0.300
	2	0.183	0.333	0.400
	3	0.267	0.450	0.700
	4	0.367	0.583	0.967
0221	1	0.516	0.533	0.633
	2	0.517	0.567	0.800
	3	0.567	0.583	0.867
	4	1.067	1.067	1.800
0212	1	0.417	0.600	0.967
	2	0.600	0.783	1.167
	3	0.700	0.883	1.667
	4	1.117	1.317	2.067
1011	1	0.316	0.567	0.667
	2	0.316	0.617	0.733
	3	0.533	0.783	1.300
	4	0.683	1.000	1.633
1002	1	0.167	0.467	0.800
	2	0.433	0.567	1.000
	3	0.467	0.700	1.567
	4	0.683	1.017	1.733
1120	1	0.117	0.233	0.400
	2	0.183	0.300	0.433
	3	0.225	0.350	0.500
	4	0.333	0.500	0.700
1111	1	0.417	0.433	0.733
	2	0.433	0.500	0.833
	3	0.383	0.550	0.933
	4	0.567	0.817	1.367
1202	1	0.550	0.650	1.067
	2	0.883	0.883	1.733
	3	0.917	1.017	1.567
	4	0.967	1.350	2.600
1220	1	0.233	0.283	0.467
	2	0.267	0.333	0.500
	3	0.367	0.533	0.667
	4	0.483	0.567	0.867

TABLE 9 Dust concentration coefficients

Parametric combination	Dust concentration constant (mg/L/kg)			
	c_{d1}	c_{d2}	c_{d3}	c_{d4}
0000	0.0135	0.0208	0.0221	0.0392
0021	0.0162	0.0235	0.0272	0.0294
0112	0.0136	0.0178	0.0281	0.0362
0100	0.0196	0.0245	0.0331	0.0429
0221	0.0235	0.0250	0.0257	0.0470
0212	0.0189	0.0246	0.0278	0.0414
1011	0.0249	0.0272	0.0345	0.0441
1002	0.0147	0.0178	0.0220	0.0320
1120	0.0171	0.0221	0.0257	0.0368
1111	0.0191	0.0220	0.0242	0.0360
1202	0.0204	0.0278	0.0320	0.0425
1220	0.0208	0.0245	0.0392	0.0417

TABLE 10 Relationship between c_{dl} and type of coal

Type of coal	Mean (mg/L/kg)			
	c_{d1}	c_{d2}	c_{d3}	c_{d4}
W	0.0176	0.0227	0.0273	0.0394
P	0.0195	0.0236	0.0296	0.0389
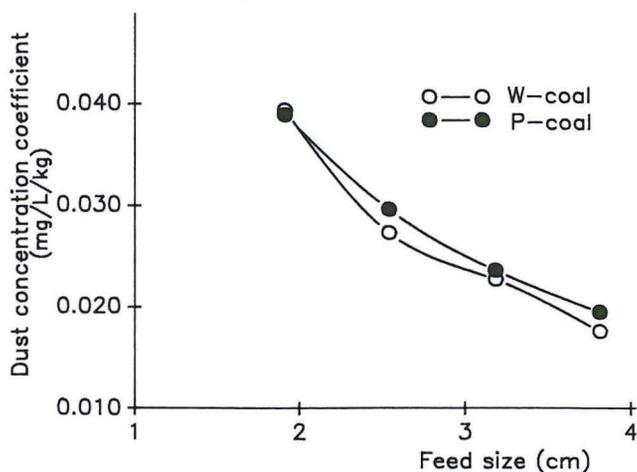
Figure 6 Relationship between dust concentration coefficient and feed size

$$\begin{Bmatrix} p_1 \\ p_2 \\ p_3 \\ p_4 \\ p_5^* \end{Bmatrix} = \begin{bmatrix} 0.887 & 0 & 0 & 0 \\ 0.072 & 0.920 & 0 & 0 \\ 0.013 & 0.040 & 0.913 & 0 \\ 0.004 & 0.010 & 0.038 & 0.946 \\ 0.023 & 0.030 & 0.049 & 0.054 \end{bmatrix} \begin{Bmatrix} f_1 \\ f_2 \\ f_3 \\ f_4 \end{Bmatrix} \qquad (14)$$

* p_5-the product particles which are smaller than 1.27 cm.
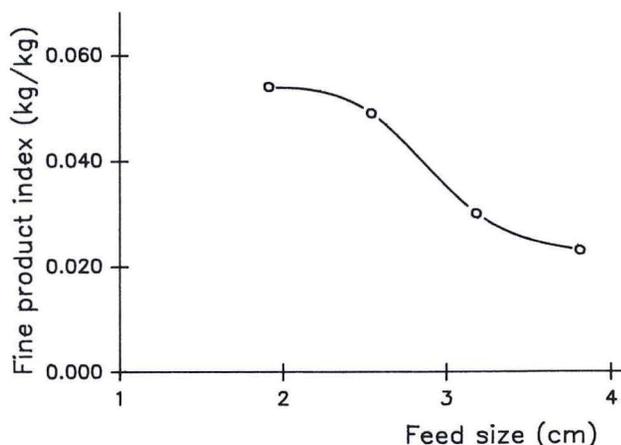
Figure 7 Relationship between fine product index and feed size

CONCLUSION

In rotary cutting machines, regrinding means that cutting head uses kinetic energy to break the recovered coal particles. The fragmentation and dust generation mainly depend on cutting velocity and properties of coal material. Selecting rpm properly is very important to control secondary fragmentation and dust generation due to regrinding. Depth of sump is also an important factor so that should be determined carefully in account of regrinding. Loading the entire recovered coal will reduce the amount of coal to be ground thus reducing the secondary fragmentation and dust generation. Because the experiments are still in progress, quantitative analysis of these relevant effects could not be presented in this paper.

ACKNOWLEDGEMENT

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MEASUREMENT AND ANALYSIS OF RESPIRABLE DUST
AND GASEOUS POLLUTANT CONCENTRATIONS
IN THREE U.S. UNDERGROUND COAL MINES

by

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ABSTRACT

The air in three U.S. underground coal mines has been sampled. The data developed show the relative importance of the various pollutants and quantify the need for control. Total respirable particulate matter (TRPM) was found to be the critical pollutant (the one requiring the most dilution for control). TRPM is considered to be "respirable coal dust" by procedures approved for sampling in U.S. underground coal mines. However, this "respirable coal dust" also includes diesel particulate matter (DPM) and particulate from other sources. While "respirable coal dust" was the critical pollutant, Raman spectroscopic analysis showed that more than half of this "respirable coal dust" was DPM. The DPM in the samples ranged from 22 to 152% of the 2 mg/m³ limit for respirable coal dust. Thus DPM was the critical pollutant using the 2 mg/m³ respirable coal dust limit. SO_2 gas produced by the combustion of diesel fuel containing sulfur was the second-most critical pollutant.

Characteristic curve slope (CCS) values were calculated to determine the quantitites of the various pollutants generated per unit of diesel haulage activity. Mine 1 had the highest TRPM CCS values with 29 mg/m³/%CO_2 for the Ramcar and 57 mg/m³/%CO_2 for the Haulageway location. The highest coal particulate CCS measured was 30 mg/m³/%CO_2 and the highest diesel particulate CCS measured was 27 mg/m³/%CO_2. Both of these high values were measured in the Mine 1 haulageway location.

Calculations were made to determine the controlling CO_2 concentrations (CCC) at which all other measured pollutants would be within their limits. These CCC values, which, for the mines sampled, were based on TRPM, varied from a low of about 0.07% for the Mine 1 haulageway location, to a high of about 0.15% for the Mine 2 Ramcar and haulageway locations. Weekly average CO_2 concentrations measured in these mines ranged from 0.09 to 0.11% for the Ramcar, from 0.05 to 0.09% for the haulageway, and from 0.08 to 0.11% for the return. The concentrations of TRPM ranged from 1 and 2 mg/m³ in mine areas frequented by personnel, and to greater than 4 mg/m³ in returns where personnel are not exposed.

INTRODUCTION

This paper presents an analysis of particulate and gaseous pollutant concentrations made over a period of 4 to 5 days each in three diesel underground coal mines. Mine air particulate matter samples were analyzed for their diesel, coal, and sulfate contents. The concentrations of CO, CO_2, NO, and NO_2 were measured and the SO_2 concentration was determined by calculation. DPM/coal samples were analyzed in the laboratory. This research was conducted as part of a joint project between MTU and other Universities in the Generic Center for Respirable dust (Pennsylvania State and the University of Minnesota) in cooperation with the Bureau of Mines Twin Cities Research Center.

Background

The use of diesel equipment in underground coal mines is on the increase (Watts, W.F. Jr., 1987). As a result, diesel exhaust and its effect on the health of underground coal miners has become an important concern. This concern has now increased even more due to the recent NIOSH decision to regard "whole diesel exhaust" as a potential occupational carcinogen (NIOSH, 1988).

Watts (Watts, W.F. Jr., 1987) presents a brief summary of diesel particulate matter health concerns as follows:

"Diesel particulate matter is of particular concern because it is almost entirely respirable in size, with 95 pct of the particles by mass having a mass median diameter less than 1.0 micrometer. This means that the particles can penetrate to the deepest regions of the lungs and, if retained, cause or contribute to the development of restrictive lung disease. Of even greater concern is the ability of the particulate matter

to adsorb other chemical substances such as potentially carcinogenic polynuclear aromatic hydrocarbons, and gases such as SO_2 and NO_2, and acids such as H_2SO_4, and HNO_3. The particulate matter acts as a carrier to bring these substances into the lung where they reach to other regions of the body and cause damage to other target organs besides the lung. Animal studies suggest chronic exposure to diesel particulate matter can cause impaired pulmonary function, reduced growth rate, increased susceptibility to lung infections and decreased clearance of lung particulate matter."

Coal mines are required by law to maintain miner exposure to airborne diesel exhaust pollutant concentrations to within limits recommended in the TLVs. Among the pollutants listed in the TLVs are CO, CO_2, NO, NO_2, SO_2, and some hydrocarbons. Figure 1 is an illustration listing the primary sources of particulate matter found in a diesel underground coal mine.
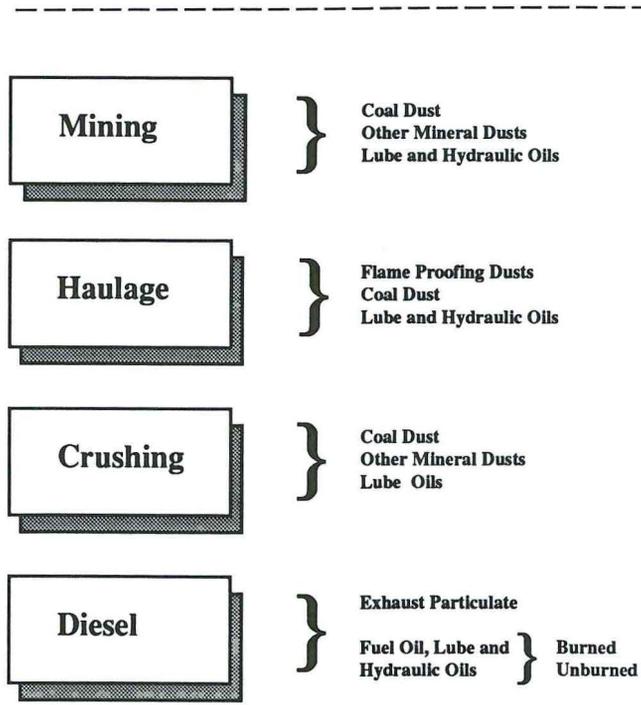
FIGURE 1. PRIMARY SOURCES OF COAL MINE PARTICULATE MATTER.

Diesel particulate matter (DPM), while shown by chemical and biological characterization to contain potential carcinogens, (Dainty, E.D., et al., 1986 and French, I.W. and Mildon, M.A., 1984), is not yet listed in the TLVs. It appears, however, that a standard will be adopted soon that will result in the need to control DPM concentrations to values that are even lower than the 2 mg/m³ 8-hr TWA respirable coal dust exposure limit. The coal dust standard also limits the DPM

concentration because there are, as yet, no fully-proven and accepted methods by which to distinguish it from coal dust.

Coal dust sampling methods as presently required by Federal law collect DPM along with coal dust and treat the entire sample as coal dust. Because the air in a diesel underground coal mine usually contains more DPM than coal dust, the measured coal dust concentration is usually at least double the actual value (Cantrell, B.K., et al., 1987 and Cornilsen, B.C., et al., 1988). As a result, distinguishing coal from diesel particulate is the subject of a number of research projects in the Generic Center for Respirable Dust.

OVERVIEW AND PROCEDURES

Mine Conditions

The three mines studied extracted coal using electrically-powered milling-type continuous miners. These machines break the coal loose and load it into Jeffrey Ramcars (diesel-powered shuttle cars). The Ramcars haul the coal to a feeder breaker — a chain conveyor that passes the coal under a spiked drum which breaks up the larger pieces and transfers the broken coal to a conveyor belt.

The model 4110 Jeffrey Ramcars from Mines 1 and 3 used naturally aspirated MWM 916.6 diesel engines. The model 4114 Ramcars from Mine 2 used naturally aspirated Caterpillar 3306 diesel engines. In each mine, the sampling took place in one section. A section usually includes all the operations needed to mine coal including a continuous miner, 2-3 Jeffrey Ramcars, a feeder breaker and a roof bolter. All 3 mines were of the room and pillar type and were laid out similar to Figure 2.
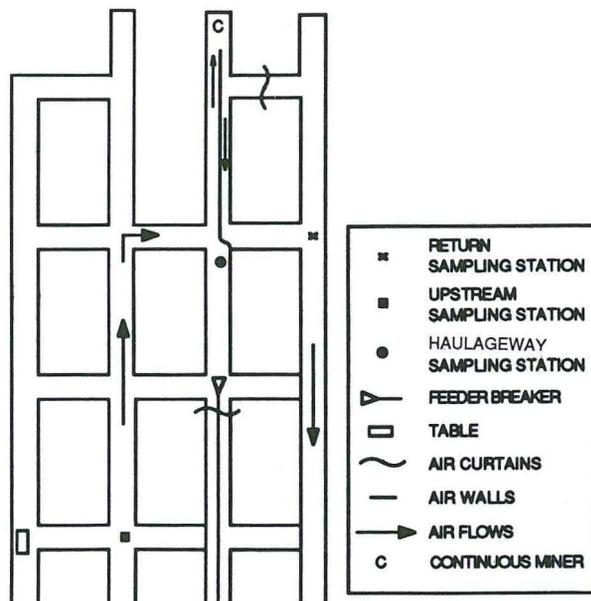

Figure 2. MINE PLAN VIEW

Samples were collected throughout the work shift at three primary locations. These were in the haulageway and return and on the Ramcar. Samples of fresh air to the section (intake) were usually collected also. Intake air contains little or no coal dust, but may contain low concentrations of diesel pollutants from outby vehicles (personnel carriers and supply vehicles).

The 'haulageway' represents the section drifts through which Ramcars travel while hauling coal from the well-ventilated mining face to the feeder breaker. Line brattice (tarpaulin-like cloth) partitions the drifts to direct ventilation air along the perimeters of the mined-out areas and into the drifts where mining is in progress. Coal dust generated by mining is, thereby, swept away from mine personnel and into the 'return' (drifts located downstream from the continuous miner in this directed air stream).

Mine Ambient Air Measurements

Rationale for Measurements. The concentration of a diesel exhaust pollutant in a mine airway is determined primarily by the emission rate upstream and the flow rate of dilution air. The section ventilation system must be properly sized to handle the anticipated emission rate. Ventilation is a significant cost item in coal mines; therefore, the mine operator must determine the minimum air flow rates needed to maintain the concentrations of the various pollutants within limits.

Because diesel engines emit a large array of pollutants, each with an exposure limit, in areas where personnel are stationed ventilation must reduce the concentration of the critical pollutant (pollutant requiring the most dilution) below its 8-hr TWA exposure limit. This critical pollutant differs for different diesel engines, different types and levels of control and for radically-different operating conditions.

Improper engine maintenance may markedly increase the emission rate for a particular pollutant making it the critical one. Our experience in a number of underground mines has, however, indicated that DPM at the 2.0 mg/m³ limit for "respirable coal dust" is usually the critical diesel pollutant.

Because control devices usually only reduce the concentration of one type of pollutant, it is not usually economical to attempt to completely eliminate this pollutant. If, for example, DPM is the critical pollutant, the wise use of particulate traps will only aim to reduce the DPM concentration to where an uncontrolled pollutant becomes the critical one. Additional DPM control will not result in further reductions in ventilation requirements without first controlling the next-most-critical pollutant that is uncontrolled by the particular device.

It becomes apparent that large numbers of measurements are required to assure adequate control of diesel emissions underground. However, the use of the systematic characteris-tic curve approach to monitoring and data analysis can greatly reduce the numbers of measurements and still provide the information needed to determine the level of control (Johnson, J. H., Carlson, D. H. and Schimmelman, M. K., 1984).

The characteristic curve approach quantifies the extent of mine air pollution and provides some of the data needed to evaluate control. The basis for this monitoring approach is that all diesel engines emit CO_2 in direct proportion to the quantity of fuel burned. Thus when the flow rate of dilution air through a mine airway remains constant, the concentration of CO_2 in the airway is directly proportional to the quantity of fuel burned and, for a cyclic operation, to the diesel horse-power used. For this reason, equation 1 can be used to calculate the rate of dilution air per brake horsepower hr of diesel work performed in the particular mine airway.

$$\frac{CFM}{BHP} = \frac{45.18 \times BSFC}{\text{Diesel-produced } CO_2, \% \text{ by Vol.}} \qquad \text{(Eq. 1)}$$

where:
BFSC = Average brake specific fuel consumption, lb/bhp- hr (can vary, 0.45 lb/bhp-hr is a fairly typical value)
CFM/BHP = Volume flow rate of dilution air, cfm per brake horsepower produced, bhp

Typically the mine air concentrations of CO_2 and of other diesel exhaust pollutants (DPM, CO, NO, NO_2, SO_2, SO_4^{--} and others) build up in a mine airway at the start of the work shift and remain relatively constant as long as the diesel equipment continue to be used in the same cyclical operation (such as hauling coal with a shuttle car). Thus, if the amount of dilution air is doubled, this also reduces the concentrations of CO_2 and of each other pollutant by half. Likewise, if the amount of diesel activity is cut in half, it reduces the concentrations of CO_2 and of each other pollutant by half. Increases in diesel activity and decreases in ventilation have the opposite effect.

Because the other pollutants and CO_2 are affected similarly by these changes, the ratio obtained by dividing the diesel exhaust pollutant concentration by the CO_2 concentration remains unchanged. Scattergrams with the particular pollutant's concentration on the ordinate and the CO_2 concentration on the abscissa are plotted and the slope of the least squares regression line through the plot is referred to as a "characteristic curve slope (CCS)".

The CCS is essentially an average pollutant/CO_2 ratio for that pollutant. Thus the CCS is independent of changes in either the amount of diesel activity or in the air flow rate, and has proven to be an excellent parameter for identifying vehicles that emit unusual quantities of the various pollutants. If, for example, the diesels in a section were emitting double the usual CO per unit volume of exhaust, the CCS would double.

Measurements Made

TWA concentrations of the various pollutants and CO_2 were measured simultaneously at each underground location. These measurements were made using the following procedures.

CO & CO_2 One Dupont p-125 pump drew mine air through a filter and into a 22-L Calibrated Instruments 5-Layer bag at approximately 50 cm^3/min. The CO and CO_2 concentrations were analyzed at the end of the sampling period by attaching the bag to an Ecolyzer 2600 CO instrument and to a Fuji ZFP5 CO_2 instrument.

NO_2 and NO Triplicate NO_2 and NO_x measurements were made using Palmes passive samplers or diffusion tubes (Palmes, E.D., et al., 1976). These tubes are open at one end allowing gases to diffuse in. The concentration at the opposite end is maintained at zero. The NO_2, therefore, diffuses through the tube at a rate that is dependent upon its concentration in the mine air and reacts with triethanolamine, an alkaline adsorbent material coating a screen that is located near the closed end of the tube. Upon adsorption, the NO_2 is converted into the nitrite ion. After sampling, the adsorbed nitrite ion is dissolved and treated with a chemical reagent to form a deep red color. The concentration of the nitrite ion is determined by measuring the absorbance of light by the solution using a colorimeter and this equals the moles of NO_2. The average NO_2 concentration in the mine air is calculated using the number of moles of NO_2 collected and the equation for Fick's first diffusion law (Hirshfelder, J.O., Curtis, C.F., and Bird, R.B., 1954).

When NO_x is sampled, a chromic-acid-coated glass fiber disk is inserted behind the triethanolamine-coated screen. Upon contact with chromic acid, NO is oxidized to NO_2. This NO_2 is adsorbed on the triethanolamine-coated screen along with the NO_2 from the mine air. Thus the combined concentration of NO and NO_2 or NO_x is measured. When this NO_x sampler is used alongside one which measures NO_2 only, the NO concentration can be calculated as the difference between the two.

Respirable Coal Dust Mine air at 2 L/min was drawn through a 10 mm nylon cyclone preseparator followed by a preconditioned/preweighed clean 25 mm Gelman A/E glass fiber filter. The "respirable particulate matter" deposited on the filter is determined by conditioning and re-weighing the loaded filter and subtracting the weight of the clean filter. The concentration is calculated using the known sampling rate and sampling time. The concentration is corrected to an "MRE-equivalent" concentration, by multiplying by an accommodation factor of 1.38.

Table 1 compares the procedure used to measure the respirable particulate matter concentration to the procedure specified for compliance sampling to determine miner exposure to respirable coal dust.

Table No. 1
Comparison of Dust Sampling Procedure
Used with the Standard Respirable Coal Dust Procedure
Used in US Underground Coal Mines.

	Procedure	
	Std.	Used
Sampling time, hr	8	4-8
Flow rate, min^{-1}	2	2
Preseparator nylon cyclone	10 mm nylon cyclone	10 mm
Filter type	Gelman A/E glass fiber	Gelman GLA 5000
Filter size	37 mm	25 mm
Filter approved	yes	no
Preweighed cass.	yes	no *
System approved	yes	no
Accommodation factor of 1.38	yes	yes

* The filter is weighed alone without the cassette.

Based on Table 1, we think our respirable particulate matter concentrations are approximately equal to respirable coal dust (coal) concentrations measured at the same locations over the same time periods by the approved procedure. However, our samplers monitored fixed locations while the Federal procedure samples a person's breathing zone. Furthermore, our samples were collected during periods of intense mining activity only, while the Federal procedure samples the entire 8-hr work shift. Thus our concentrations are probably some what higher than concentrations measured in the same mine areas using the approved method.

Fractions Diesel and Coal in "Respirable Coal Dust" Samples
The percentage of a respirable coal dust sample that consists of diesel particulate matter (DPM) was determined by the Raman method, which has been discussed in detail in other publications (Johnson, J.H., et al., 1982, and Cornilsen, B.C., et al., 1989).

This method requires collection of "coal-only" and "diesel-only" filter samples for calibration. The analysis involves the following procedures that are designed to provide high precision spectra and to allow detection of sampling irregularities when they occur. The 25 mm Gelman A/E glass fiber particulate collection filter is mounted on a sample spinner and spun to prevent sample heating and decomposition in the laser beam. A variation in laser intensity and/or background fluorescence while the Raman spectrum is being collected can change the spectrum "shape" and greatly reduce precision. Therefore, a "spectrum" summed from 5 scans is more precise than 1 spectrum collected over the same total time period.

High precision is obtained by summing a series of spectral scans to provide a high signal-to-noise ratio (S/N). We have determined that a twenty-scan sum provides a sufficiently-large S/N ratio within a reasonable time period. The data for a twenty-scan sum is collected in a series of four five-scan runs, two at one radius (inner) and two at a second radius (outer). A five-scan sum provides a sufficiently-large S/N ratio to check for reproducibility over time in the laser beam and to check for filter inhomogeneity.

Statistical calculations for data at the two radii and for the repeat scans allow evaluation of reproducibility. Comparison of intensity ratios and % DPM values for the two or three filters collected simultaneously (in the mine) provides a measure of experimental precision and accuracy.

% DPM is calculated using equation 2.

$$\text{\% DPM} = 100 / [(g'/g) \times (r'-M)/(M-r) + 1] \qquad \text{(Eq. 2)}$$

The slope (g'/g), diesel-only ratio (r'), and coal-only ratio (r) are used with an empirical intensity ratio for the mixture sample (M), to calculate %DPM. A baseline is drawn under each Raman band, tangent to the baseline minima. The intensity for each band (I_D and I_G) is then measured, and the ratio $(I_D/I_G = M)$ is used to calculate % DPM (Johnson, J.H., et al., 1982).

<u>SO_2 and SO_4^{--}</u> The following equations derived by the authors from stoichiometric relationships in the combustion of diesel fuel are used to calculate the concentrations of SO_2 and SO_4^{--} in the mine air from: 1) the fuel sulfur content, 2) the mine air CO_2 concentration, 3) the assumed percent conversion of fuel sulfur to SO_4^{--} by combustion, and 4) the ratio of the number of atoms of H to atoms of C in the fuel (obtained by analysis of the fuel);

$$SO_2(\text{ppm}) = 0.443 \times \text{pct S in fuel} \times \text{pct } CO_2^* \text{ in mine air} \times (100\text{-}n)$$
$$\text{(Eq. 3)}$$

$$SO_4^{--}(\text{mg/std m}^3) = .1267 \times (\text{pct } CO_2\text{-}.035) \times (12 + H/C) \times \text{pct S} \times n$$
$$\text{(Eq. 4)}$$

where:
n = percent conversion of fuel sulfur to sulfate,
*pct CO_2= diesel-produced CO_2 (subtract 0.035 from mine air CO_2 concentration to correct for CO_2, % in clean air)

The percentage of fuel sulfur converted to sulfate (n) can be determined as follows. The concentration of SO_4^{--} in the respirable particulate sample collected on the filter is first determined by chemical methods. The analytical procedure involves dissolution of the SO_4^{--} in water and analysis by ion chromatography. Once the weight of SO_4^{--} in the sample is determined, the mine air SO_4^{--} concentration is calculated in the same way as the respirable coal dust concentration. We refer to this measured sulfate concentration as the "actual sulfate" concentration. We also calculate the sulfate concentration using equation 4 with the assumption that 100 % of the fuel sulfur is converted to sulfate. The percentage conversion is then determined by equation 5 which expresses the "actual sulfate concentration" as a percentage of the "concentration calculated assuming 100 % conversion to sulfate."

$$\text{\% conv. to } SO_4^{--} = \frac{100 \times \text{Actual Meas. } SO_4^{--}}{\text{Calc. } SO_4^{--} \text{ assuming 100\% conv.}}$$
$$\text{(Eq. 5)}$$

The remaining percentage of the fuel sulfur is assumed to be combusted to the SO_2 form. When no catalyst is used in the exhaust, system, about 97-98 % of the fuel sulfur is combusted to SO_2 and 2-3% to SO_4^{--}. Note that it is also necessary to multiply the sulfate concentration by 1.38 to arrive at an "MRE-equivalent" value where such data are desired.

MINE AIR CONCENTRATIONS

Concentration Limits for Pollutants Measured

TWA-TLVs are available for most of the pollutants measured. The limits presented in Table 2 have been taken from the 1986-1987 issue of the TLVs (other than those for TRPM and diesel particulate). The limit used for TRPM is the TLV limit for respirable coal dust which contains no silica. The silica content was not measured. The limit used for sulfate (SO_4^{--}) is the TLV limit for sulfuric acid. Our experience indicates that this SO_4^{--} limit is rarely, if ever, exceeded, and that the main concern with SO_4^{--} is that it contributes to the already uncomfortably high respirable coal dust concentration. If SO_4^{--} makes a significant contribution to the TRPM, the mine can easily control it by purchasing a lower sulfur fuel.

As mentioned earlier, there is, as yet, no fully accepted standard method for distinguishing coal from DPM, meaning that in actual practice DPM including SO_4^{--} is considered to be respirable coal. Thus the limit for DPM is assumed here to be the same as the TLV limit for coal.

Table 2.
Pollutant Concentration Limits Used In Data Analysis.

Particulate Matter	8-hr TWA Limit	Gas	8-hr TWA Limit
TRPM	2.0 mg/m³	CO	50.0 ppm
Coal	2.0 mg/m³	CO_2	0.50 %
DPM	2.0 mg/m³	NO	25.0 ppm
Sulfate	1.0 mg/m³	NO_2	3.0 ppm
		SO_2	2.0 ppm

Determination of the Critical Pollutant and the Effect of Controls.

Diesel Pollutant Concentrations. When the measured concentrations are presented as percentages of their TLVs or limits, a higher percentage means that more dilution air is required to maintain the particular pollutant at or below its TLV or limit. Figures 3 through 11 present 4-5 shift average pollutant concentrations expressed as percentages of the limits. The TRPM values are highest in all 3 mines. Thus TRPM is the critical pollutant.
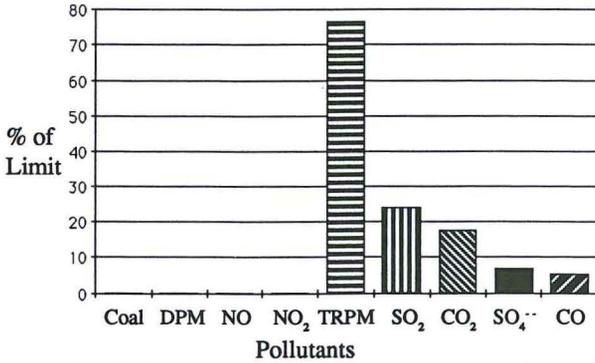
FIGURE 3. Mine 1 Ramcar Pollutant Concentrations* Expressed as a Percentage of TLV or Assumed Limit. For Particulate Data, Concentration Used Equals Measured X 1.38.

* - In Mine 1, NO and NO_2 were not sampled and the Ramcar particulate samples were not analyzed to determine the fractions diesel and coal.
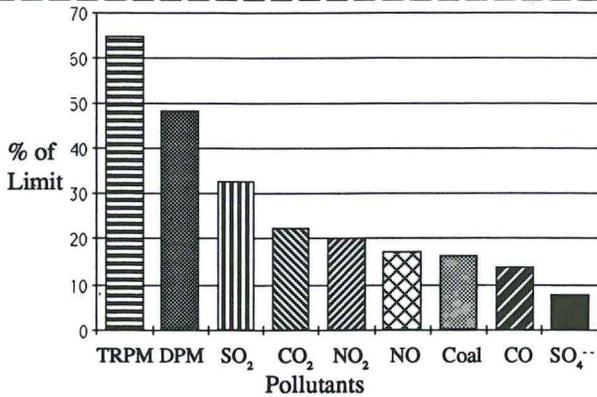
FIGURE 4. Mine 2 Ramcar Pollutant Concentrations Expressed as a Percentage of TLV or Assumed Limit. For Particulate Data, Concentration Used Equals Measured X 1.38
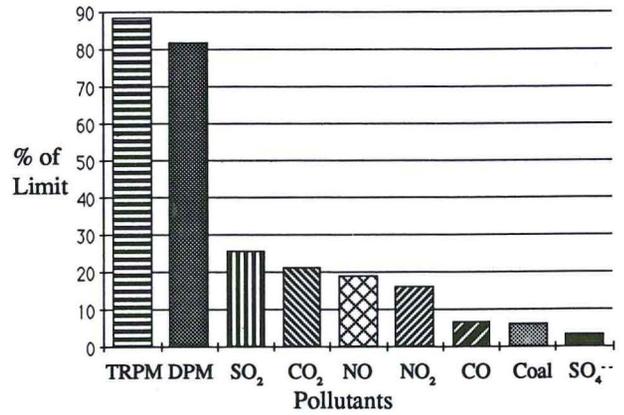
FIGURE 5. Mine 3 Ramcar Pollutant Concentration Expressed as a Percentage of TLV or Assumed Limit. For Particulate Data, Concentration Used Equals Measured X 1.38.
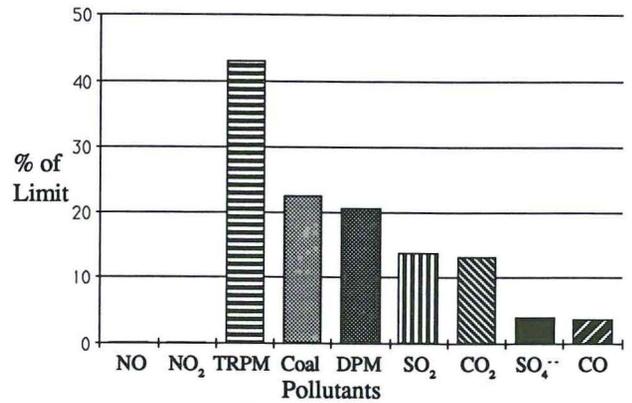
FIGURE 6. Mine 1 Haulageway Pollutant Concentrations Expressed as a Percentage of TLV or Assumed Limit, For Particulate Data, Concentration Used Equals Measured X 1.38*
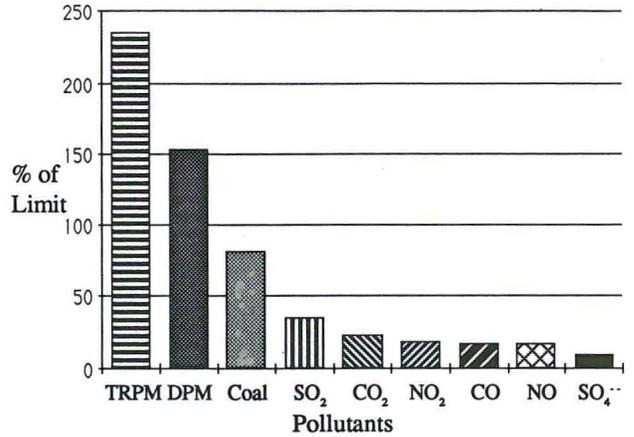
* - NO_2 and NO were not sampled.
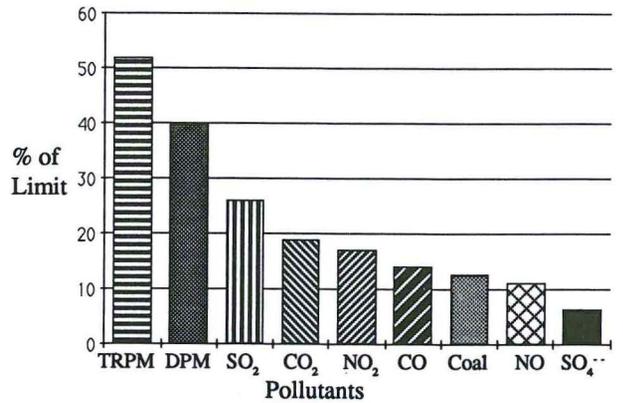
FIGURE 7. Mine 2 Haulageway Pollutant Concentrations Expressed as a Percentage of TLV or Assumed Limit, For Particulate Data, Concentration Used Equals Measured X 1.38
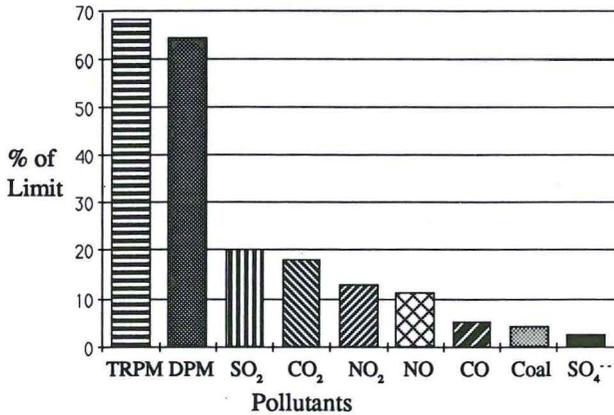
FIGURE 8. Mine 3 Haulageway Pollutant Concentrations Expressed as a Percentage of TLV or Assumed Limit, For Particulate Data, Concentration Used Equals Measured X 1.38.
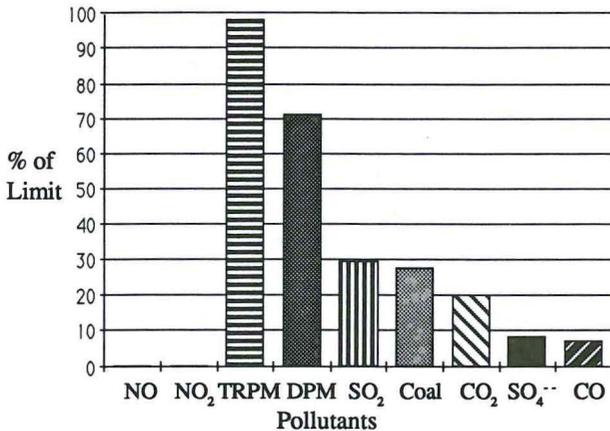
FIGURE 9. Mine 1 Return Pollutant Concentrations Expressed as a Percentage of TLV or Assumed Limit, * For Particulate Data, Concentration Used Equals Measured X 1.38.
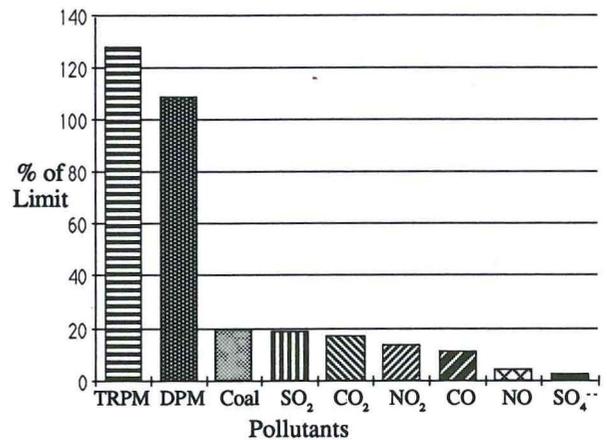
* - NO and NO_2 not sampled in this mine.

FIGURE 10. Mine 2 Return Pollutant Concentrations Expressed as a Percentage of TLV or Assumed Limit, For Particulate Data, Concentration Used Equals Measured X 1.38.

FIGURE 11. Mine 2 Return Pollutant Concentrations Expressed as a Percentage of TLV or Assumed Limit, For Particulate Data, Concentration Used Equals Measured X 1.38.

Since Raman analysis showed over half of the TRPM to be DPM in all but the Mine 1 Haulageway, DPM is the second-most critical pollutant using the 2 mg/m³ limit for respirable coal dust. From this finding, it is obvious that a functional particulate trap would reduce the measured "coal dust concentrations" to one-half or less.

Fuel Sulfur Effect. SO_2 is the third-ranked pollutant and would become the critical pollutant if significant reductions in DPM were achieved. SO_2 can also be controlled by reducing the fuel sulfur content, but in contrast to SO_4^{--}, SO_2 cannot be controlled by an ordinary particulate trap.

Fuel sulfur contents and percentages converted to SO_4^{--} were determined by the procedures discussed, and these are listed in Table 3.

Table 3.
Measured Fuel Sulfur Contents and Estimated Percentages Converted to Sulfate by Diesel Engine Combustion.

	Est. Fuel Sulfur Content	% conv. to SO_4^{--}
Mine 1 -	0.208	2.5*
Mine 2 -	0.195	2.0
Mine 3 -	0.168	1.0

* The % conversion of fuel sulfur to SO_4^{--} was not measured for Mine 1 and was assumed to be 2.5 in the calculations.

Figure 12 illustrates the effect of the fuel sulfur content on the mine air sulfate concentration where 2.5 % of the fuel sulfur is combusted to sulfate. The three SO_4^{--} concentration columns at each fuel sulfur content are for three different CO_2 concentrations: 0.05, 0.10 and 0.15 % (a typical range of CO_2 concentrations). A high 0.40 % fuel sulfur increases the SO_4^{--} concentration to greater than 0.20 mg/m^3.
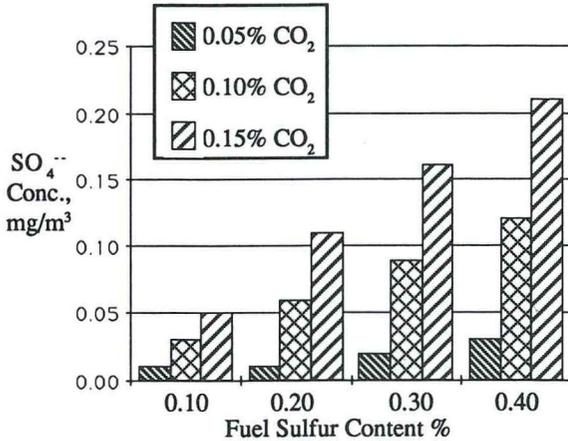

FIGURE 12. Mine Air Sulfate Concentration Vs. Fuel Sulfur Content for Mine Air CO_2 Concentrations of 0.05, 0.10, and 0.15% by Volume ---2.5% Conversion of Fuel Sulfur to Sulfate Assumed.

Figure 13 is a similar plot for 10 % conversion. Such a high percentage conversion of fuel sulfur to sulfate can occur when a catalytic converter is used on the tailpipe. The sulfate content (SO_4^{--}) exceeds 0.60 mg/m^3 for a 0.30 % sulfur fuel and 0.80 mg/m^3 for a 0.40 % sulfur fuel. These data illustrate the importance of fuel sulfur in the maintenance of coal dust concentrations within the standards. The need to purchase low sulfur fuel is obviously more critical when catalytic oxidation occurs in the exhaust system.
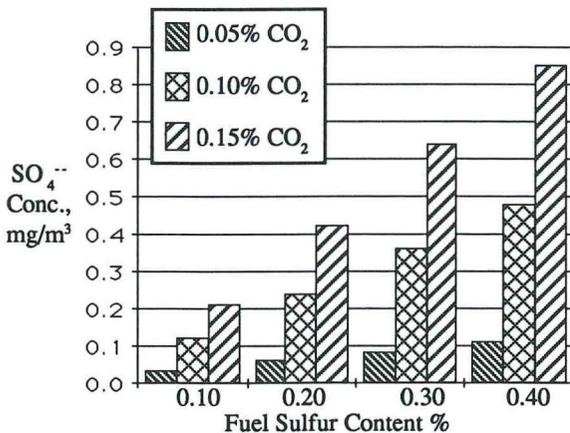

FIGURE 13. Mine Air Sulfate Concentration Vs. Fuel Sulfur Content for Mine Air CO_2 Concentrations of 0.05, 0.10, and 0.15% by Volume ---10% Conversion of Fuel Sulfur to Sulfate Assumed.

Figure 14 shows the effect of fuel sulfur on the SO_2 concentration. The graph assumes 97.5 % conversion of fuel sulfur to SO_2. The overall effect of typical changes in the fuel sulfur conversion level on SO_2 is relatively small because SO_2 is so high to begin with (usually about 97.5 %). Figure 14 illustrates that at the 97.5 % typical noncatalyzed conversion level, with 0.40 % sulfur fuel, SO_2 in the mine air reaches the 2.0 ppm limit when the CO_2 concentration reaches 0.15 %.
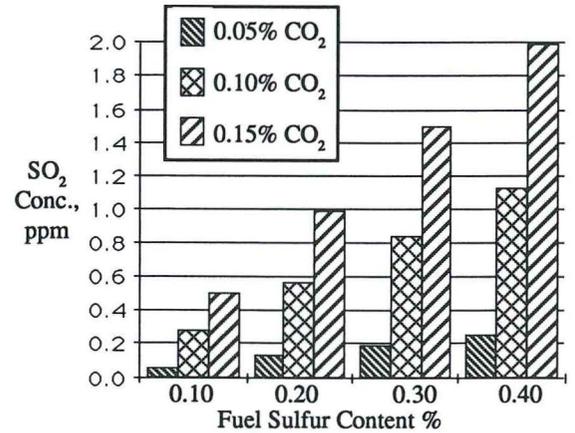

FIGURE 14. Mine Air SO_2 Concentration Vs. Fuel Sulfur Content for Mine Air CO_2 Concentrations of 0.05, 0.10, and 0.15% by Volume --- 2.5% Conversion of Fuel Sulfur to Sulfate Assumed.

In conclusion, as long as the mine is ventilated such that the CO_2 concentration remains below 0.15 %, the SO_2 concentration will not exceed its TLV at fuel sulfur contents below 0.40 % by weight. SO_4^{--} will be important whenever the cumulative respirable particulate matter from other sources (coal and other DPM) exceeds 1.5 mg/m^3 or so.

CO$_2$ Concentration Limit for Control

Both the critical pollutant and controlling CO_2 concentration (CCC) may differ for different mine locations. As discussed above, total respirable particulate matter (TRPM) which is considered to be "respirable coal dust," was the limiting pollutant for all three locations in the three mines tested.

Figure 15 illustrates these CCC values for each location. The lowest CCC was 0.07 % for the Mine 1 haulageway. This means that maintenance of the mine air CO_2 concentrations below 0.07 % for all the locations monitored, would maintain the other measured pollutants below their limits. However, CCC values range all the way up to 0.15 %, the value found for the mine 2 Ramcar and Haulageway locations. Thus it would not be economical to ventilate to the 0.07 % CO_2 level at these locations. Actual average CO_2 concentrations for these mines are illustrated in Figure 16. Figure 17 shows that TRPM or "respirable coal dust" (the critical pollutant) ranged from 1 to 2 mg/m^3 in those areas that are frequented by personnel and to over 4 mg/m^3 in the return where personnel are not affected.
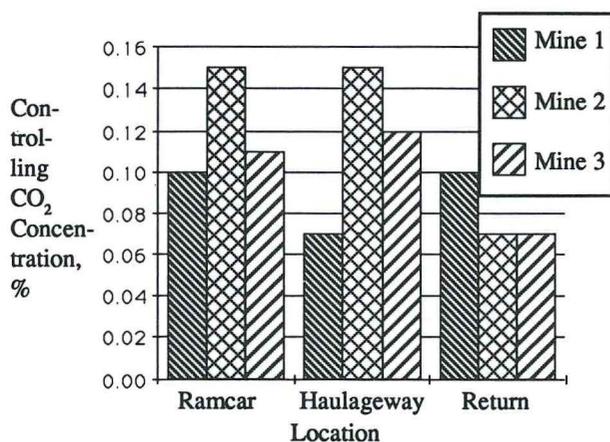
FIGURE 15. Controlling CO_2 Concentrations For Ramcar, Haulageway and Return Locations at Each of 3 Underground Coal Mines --- All Controlled by TRPM Concentration.
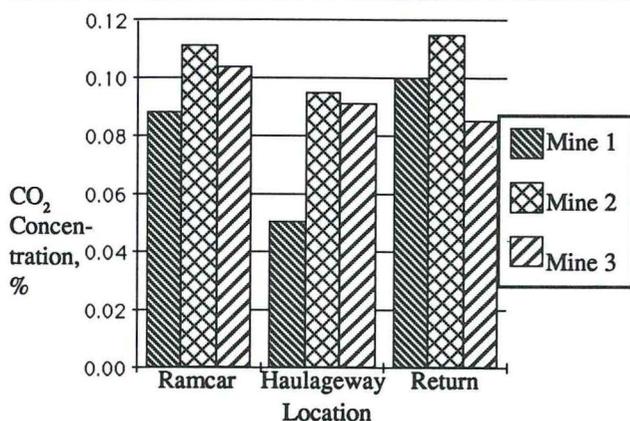

FIGURE 16. CO_2 Concentrations Averaged over One Week of Measurements in 3 Underground Coal Mines.
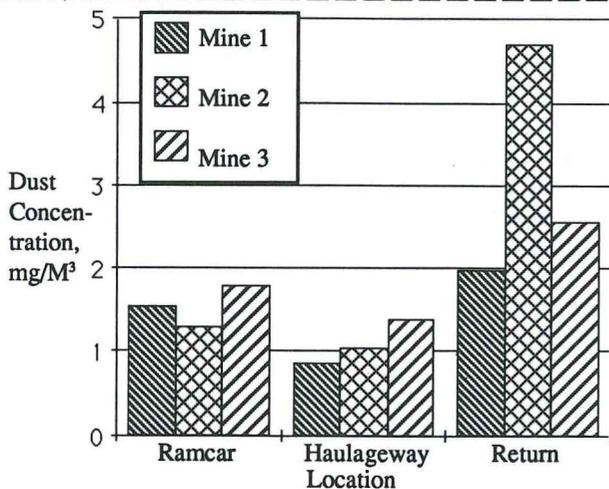

FIGURE 17. Respirable Dust Concentrations (Meas. X 1.38) Averaged over One Week of Measurements in 3 Underground Coal Mines.

SUMMARY AND RECOMMENDATIONS

1. Total respirable particulate matter (TRPM), which is considered to be "respirable coal dust," is the critical pollutant or the one requiring the most dilution for control in each of the three locations monitored in all three mines. This is based on the time-weighted average (TWA) exposure limit for respirable coal dust containing no quartz of 2 mg/m³.

2. Raman analysis showed that over half the TRPM is diesel particulate matter (DPM). For all mines and locations with one exception (Mine 1 haulageway), DPM which is considered to be "respirable coal dust" is the second-most critical pollutant. This finding suggests that functional particulate traps on the diesel haulage vehicles would reduce the "respirable coal dust concentrations" to one-half or less as measured by the mines using the procedures that are approved at the present time.

3. The SO_4^{--} concentration will be of critical importance whenever the cumulative respirable particulate matter from other sources (other DPM and coal) exceeds about 1.5 mg/m³.

4. If mine ventilation is sufficient to maintain the CO_2 concentration below 0.15 %, it will also maintain the SO_2 concentration below its TLV at fuel sulfur contents below 0.40 % by weight. At a typical 97.5 % conversion of fuel sulfur to SO_2 during diesel mine vehicle operation, 0.40 % sulfur fuel produces SO_2 in the mine air equal to the 2.0 ppm SO_2 TLV when the diesel- produced CO_2 concentration reaches 0.15 %. SO_2, like sulfate can be controlled by reducing the fuel sulfur content. However, unlike SO_4^{--}, SO_2 cannot be controlled by an ordinary particulate trap.

5. Mine 1 had the highest TRPM characteristic curve slope (CCS) with a CCS of 29 mg/m³/%CO_2 for the Ramcar and 57 for the Haulageway locations. The highest coal particulate CCS was 30 mg/m³/%CO_2 while the highest diesel particulate CCS was 27 mg/m³/%CO_2. Both of these very high CCS values were measured in the Mine 1 haulageway location indicating that Mine 1 diesels emitted more particulate and the vehicles generated more coal dust per unit of diesel haulage than engines and vehicles in either Mine 2 or Mine 3.

6. Controlling CO_2 concentrations (CCCs) varied from a low of about 0.07 % for the Mine 1 haulageway location to a high of about 0.15 % for the mine 2 Ramcar and Haulageway locations. Because TRPM was the critical pollutant, the CCC value at each location was based on the associated TRPM concentration. The TRPM concentration ranged between 1 and 2 mg/m³ in mine areas occupied by personnel, and to greater than 4 mg/m³ in returns where personnel are not exposed. Weekly average CO_2 concentrations measured in these mines ranged from 0.09-0.11 for the Ramcar, from 0.05 to 0.09 for the haulageway and from 0.08 to 0.11 for the return.

Recommended Monitoring Procedures for Underground Coal Mines

The mines themselves could greatly simplify the task of assuring that diesel emissions are properly controlled by adopting the characteristic curve approach to monitoring. Implementation in the mines would involve the following phases:

1. The first phase would involve tailpipe measurements on the various mine vehicles in use to make sure none was an unusually large polluter.

2. The second phase would be simultaneous measurement of the concentrations of the diesel exhaust pollutants of concern and of CO_2 using vehicles determined to be normal emitters in the tailpipe tests. The data would be analyzed and characteristic curves plotted (pollutant concentration on the ordinate plotted against the CO_2 concentration on the abscissa). The slope of the characteristic curve is essentially an average pollutant/CO_2 ratio. From the characteristic curves, the CO_2 concentration below which none of the pollutants would exceed their TLVs would be calculated. This CO_2 concentration would be the maximum CO_2 concentration that the mine operator would allow in similar diesel operations. The mine operator would monitor the CO_2 concentration in all airways that use diesel equipment on a regular basis. When the mine air CO_2 concentration, approached this maximum value, the mine operator would take action to either decrease diesel activity or increase the air flow.

3. The third phase would be to repeat tailpipe measurements as often as experience dictates as necessary to assure that the diesel engines in the airway are not excessive polluters.

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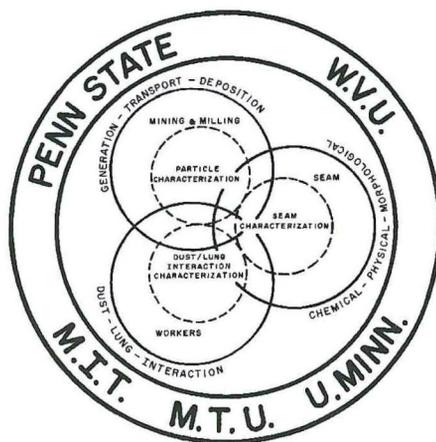
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