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The Explosion Hazard in Mining

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NOTE TO READERS

The information in this report generally is current through 1977. Since the report has been in preparation, 10 miners have been killed in two especially serious explosions at underground mines. On June 8, 1979, five miners at Cargill, Inc.'s Belle Isle salt mine, St. Mary Parish, La., died in an explosion which occurred when methane liberated from a pocket in the salt was ignited by burning insulation or arcing wires. On Nov. 7, 1980, an explosion claimed five lives at Westmoreland Coal Co.'s Ferrell No. 17 mine, Boone County, W. Va.; as of the time this report goes to press, the cause of the explosion has not been officially determined. The occurrence of these disasters demonstrates that, although fatalities from mine ignitions and explosions have been on the decrease, the explosion hazard in mining remains a serious concern. It is hoped that the information in this report will be of value in helping to prevent such disasters in the future.

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THE EXPLOSION HAZARD IN MINING

by

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ABSTRACT

This semi-technical overview of surface- and underground-mine ignitions and explosions summarizes information from a variety of sources, including inspectors', disaster, and research reports, often not readily available to the general public. It is meant to assist mine operators, miners, inspectors, and others in combating these hazards by presenting lessons learned from past disasters and from research, especially the extensive laboratory and large-scale tests made since 1910 by the Bureau of Mines. With emphasis on the causes of ignitions and explosions in underground coal mines, the report presents statistics on accidents and fatalities; discusses ignition sources, flame propagation, pressure development, and explosion control; and compares methane and coal dust explosions.

INTRODUCTION

Since 1880 about 25 billion tons of coal have been mined in the United States. During this time about 500 major gas and dust explosions and several thousand minor explosions and ignitions have occurred in U.S. coal mines. The number of fatalities from these occurrences exceeds 15,000. Explosions have also occurred in metal and nonmetal mines, and the estimated number of such explosions and resulting fatalities each exceed 100. Because an explosion is a more significant hazard, documentation of coal mine explosions is better, and more research has been conducted on coal mine explosions than for similar occurrences in other mines, emphasis is given in this paper to the coal mine environment. Explosions in coal mines have been fairly well-documented since 1880, whereas similar documentation in other mines was poor until 1975.

The frequency and magnitude of mine explosions and the number of related fatalities have been decreasing since 1910. Nevertheless, explosions still occur, partly because of the lack of appreciation of the causes and partly because of the lack of understanding of explosion phenomena. Although sufficient information is available to make an explosion a rare event, much of the information is in detailed research papers and inspectors' reports with restricted circulation. This paper presents an overview of the subject to focus on those factors which the mining public can use to further minimize

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[1] Physical scientist (now retired), Technical Support.

the mine explosion hazard. Although emphasis is given to ignitions and explosions in underground coal mines, the potential hazards in surface mines, in surface facilities of underground mines, and in metal and nonmetallic mines are also noted.

A summary of coal mine explosion research was recently published by Cybulski (6),[2] an outstanding Polish authority. Results of studies on explosion propagation and control are given in the Proceedings of the International Conference of Directors of Safety in Mines Research. This organization meets biannually. All major coal-producing countries perform research on mine safety and publish their findings. Although mining systems and practices vary in the different countries, mining hazards, particularly those relating to explosions, are universal.

At present, the general public and mining engineers accept that dispersed coal dust can explode. Historically this was not so; conclusive evidence was not provided until 1894, when tests were made in Great Britain. Even with the results of this and other studies, the mining public was slow to accept the fact. Initial tests in the Bureau of Mines experimental coal mine at Bruceton, Pa., in 1910, were designed to show the explosibility of coal dust. As late as 1928 an article was published in an attempt to show that coal dust alone would not explode. The reluctance of the mining public to accept the explosibility of coal dust accounts in a large measure for the high death toll from mine explosions between 1890 and 1920. Rice (35) published an excellent review of the early research and the stories of those pioneers who demonstrated the explosion hazards of coal dust.

EXPLOSIONS AND IGNITIONS

Underground Coal Mines

The number of fatalities per year from 1880 through 1977 from underground coal mine ignitions and explosions is shown by figure 1. The total number of fatalities since 1880 is about 15,000. The highest number (960) in any year occurred in 1907. The peaks on the curve in figure 1 are for the most part caused by one or more disasters involving a large number of miners. The data shown in figure 1 are approximate because recording by any agency was not mandatory. Much of the information was taken from Humphrey's (21) detailed description of coal mine explosions prior to 1958.

By averaging the data on the number of fatalities during 15-year periods, the smooth curve shown in figure 2 is obtained. To obtain the data for figure 2 the total number of explosions in a 15-year period, beginning with the period 1885-1899, was obtained. This total was then divided by 15 to obtain the average number of fatalities for a given period. The average number, thus obtained, was plotted at the median date for each of the time periods. Thus, figure 2 shows the trend of occurrence of underground coal mine explosions. Since 1910 the average number of fatalities per year has been decreasing.

[2]Underlined numbers in parentheses refer to references at the end of this report.

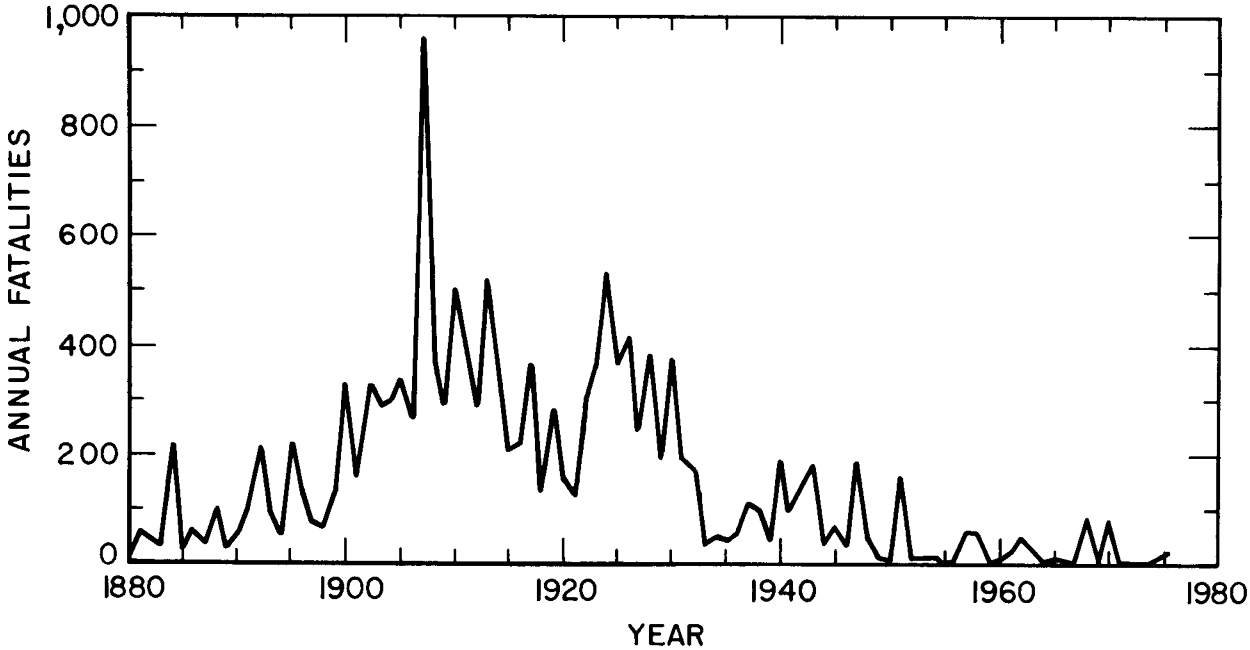

FIGURE 1. - Fatalities from coal mine explosions, 1880-1977.

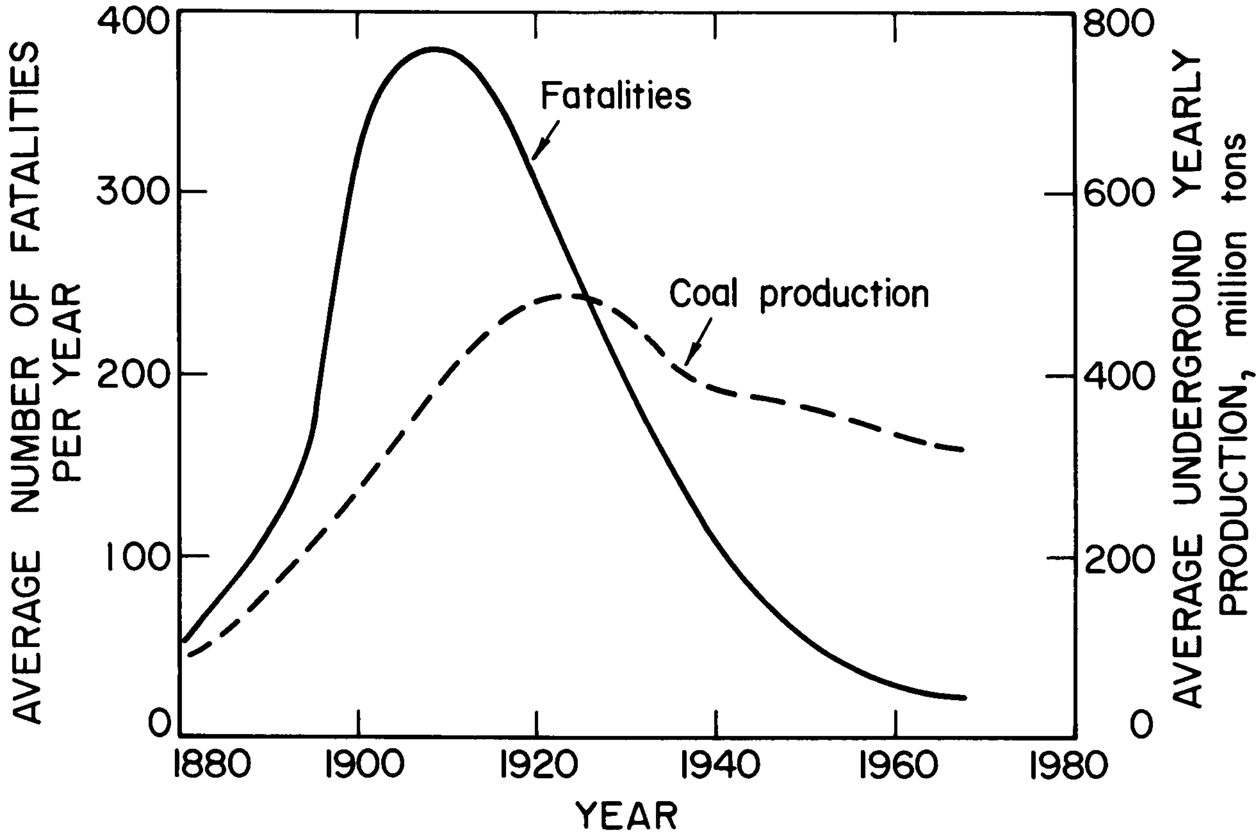

FIGURE 2. - Average yearly fatalities from coal mine explosions (averaged over 15-year periods).

For the 15-year period 1960 to 1974, the average number of fatalities per year was 21.

Data for underground coal production are also shown in figure 2. Comparison of the number of fatalities with coal production shows the number of fatalities is not directly related to coal production.

Prior to the passage of the Coal Mine Health and Safety Act of 1969 (53), a "major explosion" was defined as one which caused five or more fatalities. A "minor explosion" was defined as one which caused one to four fatalities or produced physical damage in the mine workings. The classification of explosions as major and minor was made because of the large number of such accidents. An "ignition" was understood to be the burning of methane or dust without causing a fatality or physical damage to the mine workings. However, even though these definitions were generally used, the distinction made in inspectors' reports between an ignition and a minor explosion is poor.

Table 1 shows the number of minor explosions and ignitions, major explosions and related fatalities from 1941 to 1969. The data on minor explosions and ignitions are approximate as some, but not all, ignitions are included. Evidence indicates that many unreported ignitions occurred. In fact, before 1920 it was common practice in coal mines to purposely ignite and burn methane accumulations to remove the methane from the working place. The data in table 1 show that although the number of fatalities from explosions decreased from 1941 to 1969; the number of minor explosions and ignitions per year did not materially change. The fewest fatalities (three) from explosions occurred in 1969, the last year of this period.

TABLE 1. - Coal mine explosions, 1941-69

Year	Minor explosions and ignitions		Major explosions		Year	Minor explosions and ignitions		Major explosions	
	Number	Fatalities	Number	Fatalities		Number	Fatalities	Number	Fatalities
1941	20	24	7	66	1956	42	11	-	-
1942	19	20	6	127	1957	23	7	4	59
1943	30	24	7	159	1958	22	6	2	36
1944	26	17	2	22	1959	19	1	1	9
1945	24	17	4	53	1960	10	4	-	-
1946	15	5	2	27	1961	16	4	1	22
1947	17	11	6	179	1962	16	4	2	48
1948	26	16	3	32	1963	32	5	2	31
1949	14	11	-	-	1964	34	6	-	-
1950	17	8	-	-	1965	33	5	2	14
1951	18	6	5	157	1966	24	3	1	7
1952	26	9	1	6	1967	29	12	-	-
1953	26	9	1	5	1968	23	1	2	87
1954	24	3	1	16	1969	32	3	-	-
1955	21	4	-	-					

The major coal mine explosions from 1958 through 1977 are shown in table 2, which lists pertinent information on the individual explosions and supplements those given in a previous publication (21). An average of one major explosion occurred each year during this period. Four major explosions occurred during the period 1970 through 1977. Explosions occurred in each of the eight major coal-producing States; West Virginia had the largest number (seven), followed by Kentucky (four).

TABLE 2. - Major coal mine explosions, 1958-77

Date	Mine	Company	State	Type	Ignition source	Killed
10-28-58	Burton..........	Oglebey......	WV	Methane....	Arc..............	14
10-27-58	Bishop No. 34....	Consolidation	WV	....do.....	Blasting..........	22
03-23-59	Phillips No. 1...	Phillips.....	TN	....do.....	Trolley or smoking	9
03-02-61	Viking..........	Viking.......	IN	....do.....	Arc or flame......	22
10-02-62	Robena No. 3.....	U.S. Steel...	PA	....do.....	Friction or arc...	37
01-10-62	Blue Blaze.......	Carbon.......	IL	....do.....	Arc..............	11
04-25-63	Compass No. 2....	Clinchfield..	WV	....do.....	Arc..............	22
12-16-63	No. 2...........	Carbon Fuel..	UT	....do.....	Friction..........	9
05-24-65	No. 2A..........	Kline........	TN	....do.....	Smoking..........	5
10-16-65	Mars No. 2.......	Clinchfield..	WV	....do.....	Fire.............	7
12-28-65	Dutch Creek No. 2	Mid-Continent	CO	....do.....	Trailing cable....	9
07-23-66	Siltex..........	New River....	WV	....do.....	Shuttle car-arc...	7
08-07-68	River Queen No. 1	Peabody......	KY	Coal dust..	Explosives........	9
11-20-68	No. 9...........	Consolidation	WV	NA	NA	78
12-30-70	Nos. 15 and 16...	Finley.......	KY	Coal dust..	Explosives........	38
12-16-72	Itmann No. 3.....	Itmann.......	WV	Methane....	Trolley..........	5
03-09-76	Scotia..........	Scotia.......	KY	....do.....	NA	15
03-11-76	Scotia..........	Scotia.......	KY	Methane and dust.	NA	11

NA - Not available.

Since 1970, no distinction has been made between minor and major explosions. The Federal Coal Mine Health and Safety Act of 1969 defines accidents in Section 3(b). An accident "... includes a mine explosion, mine ignition, mine fire, or mine inundation, or injury to, or death of, any person." Neither the Act nor the regulations further define explosion or ignition. For MSHA's purposes, an ignition is now defined as the burning of gas or dust without injury, fatality to miners or damage to the mine workings. An explosion is defined as the burning of gas or dust, or the sudden release of pressure which causes injury or fatality to one or more miners or causes physical damage to the mine workings. This definition of explosion includes accidents such as the bursting of a steam boiler, air receiver, or compressed gas cylinder, or a steam explosion in a refuse bank, as well as the more common exothermic combustion reaction between a fuel (gas or dust) and oxygen (air). All of these types of explosions have occurred in and about mines; the majority of explosions are of the combustion reaction type.

In accordance with the above definition, a listing of underground coal mine explosions between 1970 and 1977 is given in table 3. A total of 54 explosions occurred, causing 78 fatalities and 91 injuries. About 90 percent of the explosions involved methane. Four of the explosions involved coal dust alone and four involved both gas and dust.

TABLE 3. - Underground coal mine explosions, 1970-77

Date	Mine	Company	State	Type	Ignition source	Killed	Injured
01-06-70	Moss No. 2..........	Clinchfield........	VA	Methane.....	Friction........	-	1
01-23-70	Eagle No. 2.........	Peabody............	IL	.....do.....	.......do.......	-	2
02-06-70	Guyan No. 1.........	Island Creek.......	WV	Coal dust...	Welding.........	-	2
02-27-70	No. 18..............	Wanomie............	PA	Methane.....	Safety lamp.....	-	3
03-05-70	Forge Slope.........	Glen Nan...........	PA	.....do.....	.......do.......	-	4
04-10-70	Homer City..........	Helen..............	PA	.....do.....	Friction........	1	3
04-20-70	Compass No. 2.......	Clinchfield........	WV	.....do.....	Trolley or motor	1	-
06-02-70	No. 3...............	Kem................	VA	.....do.....	Arc.............	-	1
06-15-70	No. 28..............	(Abandoned)........	WV	.....do.....	Smoking.........	-	2
08-28-70	Lambert Fork........	Clinchfield........	VA	.....do.....	Friction........	-	1
09-02-70	.........do.........	........do.........	VA	.....do.....	.......do.......	-	2
09-10-70	Cedar Grove No. 1...	Consolidation......	WV	.....do.....	.......do.......	-	1
09-30-70	Blacksville No. 1...	........do.........	WV	.....do.....	.......do.......	-	2
10-26-70	St. Charles.........	A&T................	KY	.....do.....	Smoking.........	-	2
10-28-70	Va. Pocahontas No. 2	Island Creek.......	VA	.....do.....	Friction........	-	1
11-30-70	Pyro No. 2..........	Pyro...............	KY	.....do.....	Smoking.........	-	1
12-30-70	Nos. 15 and 16......	Finley.............	KY	Coal dust...	Explosives......	38	1
02-01-71	Maitland............	Consolidation......	WV	Methane.....	Friction........	-	1
03-15-71	Forge Slope.........	Glen Nan...........	PA	.....do.....	Explosives......	-	2
03-16-71	Va. Pocahontas No. 4	Island Creek.......	VA	.....do.....	Smoking.........	-	2
05-21-71	Eagle No. 1.........	Peabody............	IL	.....do.....	Friction........	-	1
05-24-71	.........do.........	........do.........	IL	.....do.....	.......do.......	-	2
05-27-71	Florence No. 2......	Florence...........	PA	.....do.....	.......do.......	-	3
06-01-71	Humphrey No. 7......	Christopher........	WV	.....do.....	.......do.......	-	1
06-05-71	.........do.........	........do.........	WV	Coal dust...	Explosives......	-	1
01-04-72	No. 3...............	Itmann.............	WV	.....do.....	Cutting machine.	-	8
03-01-72	Kepler..............	Consolidation......	WV	Methane.....	Friction........	-	1
05-20-72	Moss No. 2..........	Clinchfield........	VA	Coal dust and methane	Power cable.....	-	-
06-13-72 06-15-72	Concord No. 1.......	U.S. Steel.........	AL	MAPP gas and coal dust.	Friction........	-	7
06-15-72	Zeigler No. 9.......	Zeigler............	KY	Methane.....	.......do.......	-	2

08-07-72	Maple Creek No. 2...	U.S. Steel.........	PA	Methane.....	Friction........	–	1
12-07-72	Marianna No. 58.....	Bethlehem..........	PA	.....do.....	.......do.......	–	2
12-16-72	No. 3...............	Itmann.............	WV	.....do.....	Trolley.........	5	3
03-10-73	Va. Pocahontas No. 1	Island Creek.......	VA	.....do.....	Friction........	–	2
09-25-72	No. 4...............	Red Ash Oakwood....	VA	.....do.....	Battery.........	2	–
11-09-73	Va. Pocahontas No. 2	Island Creek.......	VA	.....do.....	Friction........	–	1
01-10-74	Pinnacle Creek No.50	U.S. Steel.........	WV	.....do.....	.......do.......	–	1
02-04-74	Va. Pocahontas No. 2	Island Creek.......	VA	.....do.....	.......do.......	–	1
04-01-74	Va. Pocahontas No. 1	........do.........	VA	.....do.....	.......do.......	–	1
07-15-74	Bishop..............	Consolidation......	WV	.....do.....	.......do.......	–	1
10-22-74	Beatrice............	Beatrice Pocahontas	VA	.....do.....	Power line......	–	–
01-28-75	Bird No. 3..........	Bird...............	PA	.....do.....	Friction........	–	1
08-02-75	Va. Pocahontas No. 4	Island Creek.......	VA	.....do.....	.......do.......	–	2
10-11-75	Bessie..............	U.S. Pipe and Foundry.	AL	Methane and coal dust.	Explosives......	–	1
11-12-75	Beckley No. 1.......	Ranger Fuel........	WV	Methane.....	Friction........	–	1
02-10-76	Wolf Creek No. 4....	A&T Massey.........	KY	.....do.....	Power line......	1	–
02-17-76	.........do.........	........do.........	KY	.....do.....	Friction........	–	1
03-09-76	Scotia..............	Scotia.............	KY	.....do.....	NA	15	–
03-11-76	.........do.........	........do.........	KY	Methane and coal dust.	NA	11	–
03-16-76	Kermit No. 1........	Kermit.............	WV	Methane.....	.......do.......	–	1
05-13-76	Mulga...............	Mead...............	AL	Oxyacetylene	Welding.........	–	1
05-18-76	J. Walters No. 3....	J. Walters.........	AL	Methane.....	Friction........	–	2
07-30-76	Braztah No. 5.......	Braztah............	UT	.....do.....	.......do.......	–	2
04-18-77	No. 2...............	Ron................	WV	Oxyacetylene	Welding.........	–	1
07-07-77	St. Charles.........	P & P..............	VA	Methane.....	Smoking.........	4	–
07-22-77	Terry Glenn.........	Terry Glenn........	KY	.....do.....	.......do.......	–	5

During this same period (1970 through 1977) 335 ignitions of methane (or methane and coal dust) occurred in underground coal mines. These data are shown in table 4. Eighty-five percent of these ignitions were caused by frictional sparks generated by cutting machines or continuous mining machines when the bits struck hard materials at the working face. Each ignition has the potential to become an explosion if sufficient force develops to cause damage. The number of ignitions per year is now about double the number of minor explosions reported prior to 1970.

TABLE 4. - Methane ignitions in underground coal mines, 1970-77

Year	Number of ignitions						
	Frictional sparks	Welding	Arc	Battery	Explosives	Lightning	Smoking
1970........	30	-	-	-	1	-	-
1971........	29	1	-	-	-	-	-
1972........	19	-	-	-	-	-	-
1973........	21	3	1	-	-	-	1
1974........	41	5	2	[1]1	1	-	-
1975........	53	4	5	-	4	-	-
1976........	61	7	2	-	-	2	1
1977........	31	4	3	-	2	-	-
Total..	285	24	13	1	8	2	2

[1]Hydrogen ignition.

The trend in methane ignitions in British coal mines is approximately the same as in American coal mines. Although they report a lesser number of ignitions, the frequency is not materially decreasing with time (9).

Surface Coal Mines

Data on ignitions and explosions at surface coal mines and surface facilities of underground coal mines between 1958 and 1977 are given in table 5. In this 20-year period, there occurred 100 ignitions and explosions (an average of five per year) and 20 fatalities (an average of one per year). Almost all of the accidents listed conform to the definition of explosion rather than ignition. The data represent those accidents reported to the Bureau of Mines or to the Mining Enforcement and Safety Administration (MESA), MSHA's predecessor agency. It can be assumed that prior to 1970 additional explosions occurred which were not reported. For example, Schrecengost and Childers made a survey and study in 1964 of 29 fluidized-bed thermal coal dryers. They report, "Forceful explosions which produced external effects had occurred in 11 (39 percent) and internal 'puffs' or 'bumps' had occurred in at least 24 (79 percent) of the plants examined..." (49).

TABLE 5. - Surface coal mine ignitions and explosions, 1958-77

Date	Mine	Company	State	Facility	Type	Ignition source	Killed	Injured
03-29-58	No. 3..............	Gary............	MD	Hoist.....	NA	NA	-	2
06-25-58	Jane Ann No. 1......	Powellton........	WV	Tipple....	Dust........	NA	-	2
05-07-59	O'Donnel No. 1......	R&P.............	WV	Prep plant	.....do.....	NA	-	3
07-24-59	Dekoven............	Pittsburg & Midway.	KY	Tunnel....	Methane.....	Electric switch.	1	2
08-12-59	Montour No. 4......	Consolidation....	PA	Prep plant	Dust........	Fire......	-	1
10-06-59	Bishop.............	.......do........	WV	Prep plant	.....do.....	Sparks....	-	3
12-29-59	O'Donnel No. 1......	R&P.............	WV	....do....	.....do.....	Gas heater	-	1
02-01-60	Central............	Omar............	WV	....do....	.....do.....	Open flame	-	1
04-27-60	Itmann.............	Consolidation....	WV	....do....	.....do.....	....do....	1	2
06-22-60	Black Eagle.........	Amigo............	WV	Oxygen cylinder.	Acetylene...	NA	1	1
11-01-60	No. 3..............	Clells...........	VA	Battery...	Hydrogen....	Arc.......	-	1
11-21-60	Central............	Amherst..........	WV	Chute.....	Dust........	Welding...	-	1
11-25-60	Ruby No. 2..........	Robey Run........	WV	Dryer.....	.....do.....	Sparks....	-	2
12-09-60	Kano No. 6..........	Pecks Run........	WV	Prep plant	.....do.....	Gas heater	-	4
01-09-61	Lucerne No. 3......	R&P.............	PA	Prep plant	.....do.....	Sparks....	-	-
08-02-61	Joanne.............	Joanne...........	WV	Dryer.....	.....do.....	....do....	-	1
08-09-61	Saxsewell No. 1.....	Gauley...........	WV	Prep plant	.....do.....	NA	1	-
02-05-62	Federal No. 1.......	EAC..............	WV	Dryer.....	.....do.....	Fire......	-	-
03-08-62	.........do.........	.......do........	WV	....do....	.....do.....	....do....	-	-
03-19-62	.........do.........	.......do........	WV	....do....	.....do.....	Sparks....	-	-
03-23-62	.........do.........	.......do........	WV	....do....	.....do.....	....do....	-	-
03-23-62	Nemacolin...........	Buckeye..........	PA	Refuse dump.	.....do.....	Fire......	-	-
03-28-62	Nelms No. 2.........	Youghigheny......	OH	Silo......	Methane and coal.	Welding...	-	3
05-07-62	Federal No. 1.......	EAC..............	WV	Dryer.....	Dust........	Sparks....	-	-
10-26-62	No. 6..............	Carbon Fuel......	WV	....do....	.....do.....	....do....	-	4
11-01-62	Kopperston..........	EAC..............	WV	....do....	.....do.....	....do....	-	-
12-05-62	Redwing............	Colowyo..........	CO	Prep plant	.....do.....	Friction..	-	1
01-05-63	Mary Sara No. 1.....	Chapel...........	WV	....do....	.....do.....	Welding	-	3
01-18-63	Bird No. 3..........	Bird.............	PA	Silo......	.....do.....	NA	3	-
02-02-63	No. 4..............	Ky. Mt...........	KY	Stove.....	Diesel fuel.	Fire......	-	2
05-17-63	Teramana...........	Teramana.........	OH	Strip.....	NA	NA	-	-
10-11-63	Moss No. 3..........	Clinchfield......	VA	Prep plant	Dust........	Sparks....	-	4
11-13-63	Federal No. 1.......	EAC..............	WV	Dryer.....	.....do.....	....do....	-	-
12-26-63	Grays Creek.........	Grundy...........	TN	NA	Acetylene...	NA	-	3
01-02-64	NA	EAC..............	WV	Pumproom..	Methane.....	Smoking...	-	1
02-03-64	Compass No. 3.......	Clinchfield......	WV	Dryer.....	Dust........	Sparks....	-	-
03-28-64	Norma..............	M&P.............	OH	Tipple....	.....do.....	NA	-	-
05-15-64	Maiden No. 3.......	Valley Camp......	PA	Dryer.....	.....do.....	Sparks....	-	-
06-17-64	Saxsewell..........	Gauley...........	WV	....do....	.....do.....	....do....	1	3
11-11-64	No. 22.............	Bethlehem........	KY	....do....	.....do.....	....do....	-	-
12-31-64	Lynch..............	U.S. Steel.......	KY	....do....	.....do.....	NA	-	-
02-11-65	Moss No. 2..........	Clinchfield......	VA	....do....	.....do.....	NA	-	-
02-15-65	Keystone...........	EAC..............	WV	Trucks....	NA	NA	-	-
03-18-65	King...............	U.S. Fuel........	UT	NA	NA	NA	-	1
08-07-65	Moss No. 3..........	Clinchfield......	VA	Prep plant	Dust........	Wind storm	-	-
01-12-66	No. 3..............	Valley Camp......	WV	Dryer.....	.....do.....	Sparks....	-	-
02-02-67	Nos. 7 and 8........	Spring Canyon....	UT	NA	.....do.....	NA	-	1
06-05-67	No. 17.............	Island Creek.....	WV	NA	NA	NA	1	-
01-22-68	Loveridge..........	Mountaineer......	WV	Dryer.....	Dust........	Sparks....	-	3
04-23-68	Scotia.............	Scotia...........	KY	....do....	.....do.....	NA	-	-
05-17-68	Nos. 7 and 8........	Spring Canyon....	UT	Prep plant	.....do.....	Welding...	-	-
08-15-68	Orient No. 5........	Freeman..........	IL	Tunnel....	Methane.....	NA	4	-
09-19-68	B&B................	B&B.............	WV	Auger.....	.....do.....	NA	1	1
10-08-68	River Dock..........	Weirton..........	OH	Conveyor..	Dust........	NA	-	-
10-21-68	Elkhorn No. 3.......	Evanston.........	KY	NA	Methane.....	NA	-	1
10-31-68	Kentland...........	Clinchfield......	KY	Dryer.....	Dust........	Sparks....	-	1
02-06-69	Robinson Run No. 95.	Consolidation....	WV	....do....	.....do.....	....do....	-	-
05-14-69	Federal No. 2.......	EAC..............	WV	Pumphouse.	Methane.....	NA	-	-

NA - Not available.

TABLE 5. - Surface coal mine ignitions and explosions, 1958-77--Continued

Date	Mine	Company	State	Facility	Type	Ignition source	Killed	Injured
07-29-69	No. 9..............	Consolidation....	WV	NA	Methane.....	NA	-	1
09-18-69	No. 2..............	Sinclair.........	OK	Tunnel....	Dust........	Explosives	-	3
01-31-70	Beatrice...........	Beatrice Pocahontas.	WV	Silo......	Methane.....	NA	-	-
02-14-70	Humphrey No. 7......	Christopher......	WV	NA	Dust........	Welding...	-	1
06-12-70	Lambert Fork........	Clinchfield......	VA	Tunnel....	Methane.....	Arc.......	-	-
06-30-70	River King..........	Peabody..........	IL	Strip shovel.	Hydraulic oil.	Light bulk	1	-
03-16-71	Va. Pocahontas No. 4	Island Creek.....	VA	Shaft.....	Methane.....	Smoking...	-	2
04-20-71	Jenkins Jones.......	Pocahontas.......	WV	Bin.......	Dust........	Sparks....	-	4
07-06-71	No. 4 Slope.........	MGS..............	PA	Hoist.....	Gasoline....	NA	-	-
07-12-71	Inland..............	Inland Steel.....	IL	Prep plant	Dust........	Welding...	-	1
08-15-71	Coal Loading........	Indian Pocahontas	WV	Plant.....	.....do.....	NA	-	1
03-07-72	Burning Star No. 3..	Consolidation....	IL	Chute.....	.....do.....	Explosives	-	5
08-14-72	Kopperston Nos. 1 and 2.	EAC..............	WV	Dryer.....	Methane.....	Arc.......	-	-
08-15-72	.........do.........	.......do........	WV	....do....	Dust........	Open flame	-	-
09-19-73	No. 32..............	Bethlehem........	PA	Shaft.....	Methane.....	Welding...	-	-
01-30-74	Guyan No. 5.........	Island Creek.....	PA	Refuse pile.	Steam.......	NA	2	-
09-19-74	Kitt No. 1..........	Ohio Atlas.......	WV	Shaft.....	Methane.....	Welding...	-	-
10-21-74	Captain.............	Southwestern Ill.	IL	Shop......	Tire........	....do....	1	-
10-22-74	Beatrice............	Beatrice Pocahontas.	VA	Shaft.....	Methane.....	Arc.......	-	-
01-05-75	Bee Hive............	American.........	UT	Tipple....	Dust........	....do....	-	-
03-20-75	Va. Pocahontas No. 5	Island Creek.....	VA	Shaft.....	Methane.....	Welding...	-	-
06-05-75	.........do.........	.......do........	VA	A Shaft...	.....do.....	....do....	-	-
09-22-75	McKinley............	Pittsburg and Midway.	NM	Prep plant	Dust........	....do....	-	2
11-10-75	No. 5..............	Island Creek.....	VA	Shaft.....	Methane.....	....do....	-	-
01-21-76	Federal No. 2.......	EAC..............	VA	H.R. Shaft	.....do.....	Lightning.	-	-
08-12-76	No. 131............	Bethlehem........	WV	Prep plant	Oxyacetylene	Welding...	-	-
08-23-76	No. 50.............	U.S. Steel.......	WV	No. 3 shaft.	Methane.....	....do....	-	-
10-12-76	No. 131............	Bethlehem........	WV	Pump......	Steam.......	....do....	-	-
11-09-76	Wharton No. 4.......	EAC..............	WV	Regulator.	Oxyacetylene	....do....	-	1
11-11-76	No. 1..............	Permac...........	VA	Well house	Methane.....	Arc.......	-	1
01-18-77	Bottom Creek No. 2..	Hawley...........	WV	Pumphouse.	.....do.....	Smoking...	-	6
03-01-77	Welch No. 1.........	Bills............	OK	Shop......	Acetylene...	Bomb......	-	1
05-05-77	Revloc No. 32.......	Bethlehem........	PA	Shaft.....	Methane.....	Welding...	1	-
06-01-77	Robinson Run No. 95.	Consolidation....	WV	16-inch borehole.	.....do.....	Arc.......	-	1
06-10-77	Lancashire No. 25...	Barnes Tucker....	PA	Silo......	Methane and dust.	Welding...	-	3
07-29-77	Florence No. 2......	Florence.........	PA	14-inch borehole.	Methane.....	....do....	-	-
12-06-77	Revloc No. 32.......	Bethlehem........	PA	Borehole..	.....do.....	Arc.......	-	-

NA - Not available.

Almost half of the accidents (41) were coal dust explosions in preparation plants and dryers. About fifteen percent (13) were in shafts or boreholes; 4 explosions each occurred in silos and pumps. In two accidents oxygen-acetylene cylinders blew up. The trend over time indicates a decrease in explosions in preparation plants and an increase in explosions in shafts and boreholes.

Fifty-two of the surface explosions involved coal dust and 30 involved methane. Single explosions occurred with hydrogen and acetylene gases, and single explosions involved the liquid fuels: gasoline, diesel fuel, or hydraulic fluid. Two steam explosions occurred, and in one instance a vehicle tire blew up when welding was being performed on a truck tire rim. The heat from the welding operation generated combustible gas within the partially deflated tire; ignition was from the red hot rim.

An unusual coal dust explosion occurred in a surface refuse pile on Jan. 30, 1974, causing two fatalities. During loading of red dog from a 50-foot-high burning refuse pile, a major slide occurred. Coal dust in the pile was dispersed during the slide and was ignited by the burning refuse. Flame projected 400 feet outward, blowing one victim 900 feet. In another unusual accident, on Aug. 24, 1975, a steam explosion occurred within a refuse bank. About 80,000 cubic feet of refuse was forcefully ejected 500 feet. Rain water had apparently seeped into the burning pile; disturbance from a red dog loading operation at the base caused the sudden inrush of the entrapped water. A literature search (13, 15) showed that prior to these events at least three other serious dust explosions and four steam explosions had occurred in refuse piles.

Metal and Nonmetallic Mines

Prior to 1966 accidents in metal and nonmetallic mines were not required to be reported to the Federal government. Hence complete historical statistics on explosions in these mines are not available. The following brief summary is given in Miners Circular No. 55R (4):

> Numerous ignitions of methane have occurred in certain types of metal mines and tunnels....
>
> The best and surest precaution against accidents from ignition of explosive mixtures of methane and air is to prevent the accumulation of such concentrations by ventilation that will remove the gas. When it is found that methane may be encountered in tunnels or mine workings, regular checks should be made with a flame safety lamp or preferably with one of the more sensitive detectors, and any gas so found should be removed before any other work is done in that locality. To prevent ignitions of methane in workings where dangerous quantities of gas may be encountered, 'No Smoking' rules should be strictly observed, all open flames should be excluded, only permissible types of electrical equipment should be used, and blasting practices should be controlled very carefully.

A methane explosion in a metal-mine development shaft in Alpena County, Mich., killed five men and injured the sixth in 1952. The ignition occurred during unwatering operations, and the force of the explosion was confined to the shaft and the area immediately surrounding the collar of the shaft. The explosion was probably initiated by a hand-held electric drill operated by 1 of 2 men taking rock samples 180 feet below the collar of the shaft.

In the metal and nonmetallic mines of California there is a record of 20 explosions during the period 1926-53. These explosions resulted in 5 fatal and 34 nonfatal injuries. The cause of the ignitions are as follows: Open lights (carbide) 13, shorted electric light wires 1, smoking 5, and cause unknown 1....

Sulfide dust has exploded in certain mines [(10)]. Such explosions usually are attributed to the ignition of fine sulfide dust in suspension by the flame from a blast; they have damaged timber and in some instances caused the death of men who inhaled or were enveloped in the burning sulfide dust. A serious explosion occurred in 1943, resulting in the death of 8 men and injury to 17 others from the fumes following an explosion of sulfide dust. The sulfide-dust cloud was created, raised into suspension, and ignited by blasting a number of shots in a stope in heavy sulfide ore. The concussion from the explosion threw a belt from the fan pulley on the surface, the air current reversed, and the fumes were swept back over men going off shift through a haulageway normally on fresh air. The following precautions were adopted to prevent future disaster of this type:

Remove all men from the vicinity when blasting in stopes.

Place an attendant at the fan at these times.

Blast from a control switch at a safe location.

Wet down stopes before blasting.

Limit the number of holes to be blasted in one place.

Dangerous quantities of sulfur dioxide and hydrogen sulfide plus some carbon monoxide have been found after blasting in heavy sulfides. To prevent sulfide-dust explosions, settled dust should not be allowed to accumulate, and all working places should be thoroughly wetted by sprinkling before blasting....

A widespread gilsonite-dust explosion, which occurred in the No. 1 Incline mine (opencut) November 5, 1953, killed 8 mine employees, the only men in the mine at the time of the explosion, and slightly injured 3 men working on the surface; 5 of the 8 bodies were recovered 15 days after the explosion, and the other 3 bodies were not recovered until March 12-14, 1954.

Owing to the large, open, unsupported wall area and the quantity of debris that fell into the inclined shaft area, recovery of the last 3 bodies was delayed because a shaft had to be sunk 131 feet and a drift driven 126 feet to reach the inclined shaft where the 3 men were working at the time of the explosion and where the bodies were found.

The explosion probably originated on the surface near the bucket elevator when a cloud of gilsonite dust, produced by dumping a car of ore, was ignited probably by electric arcing at the bucket-elevator enclosure or from static discharge. The explosion was propagated throughout the length and depth of the mine.

The No. 1 Incline mine was opened by an inclined shaft, which was extended about 660 feet on a dip of 60° in the top part and 57° in the bottom part. The mine was equipped with a steel headframe, a single-drum hoist geared to an electric motor, and a steel car of 2-ton capacity. The inclined shaft was used for transporting ore, supplies, and men.

Further information on ignitions and explosions in metal and nonmetallic mines can be obtained from other U.S. Bureau of Mines and MESA publications, and inspectors' reports. The data, summarized in table 6, shows 94 fatalities and 99 injuries since 1884. Generally only serious accidents or those with multiple fatalities were reported. It can be inferred from the 16 explosions reported in 1976 and 1977, when mine inspectors were asked to be more careful in reporting explosions in metal and nonmetal mines, that numerous other ignitions and explosions occurred in previous years. Most of the methane explosions were caused by smoking, explosives or electric arcs.

Potential Explosion Hazards in Oil Shale Mines

Oil shales, particularly those shales with high kerogen content, are combustible and may present a fire and an explosion hazard. In 1935 (1) exploratory trials, made with oil shale dust in an 8-inch-diameter gallery 17 feet long, showed that an explosion would propagate when the shale contained 20 or more gallons of oil per ton. These trials were followed by two large-scale tests in the Experimental Mine (14) which verified the gallery results. In one test, a loading of 0.9 pound of oil shale dust per linear foot of entry propagated flame 800 feet beyond the 200-foot length containing the oil shale dust. The flame velocity was 1,000 ft/sec. Based on limited test data, an estimated 23 to 65 percent incombustible (rock dust) would be needed to arrest flame propagation.

In 1974 five carefully chosen oil shale dusts were studied in the laboratory. The oil contents were 14.1, 20.1, 30.9, 33.9 and 64.8 gallons per ton. The study showed that dust explosibility increased with oil content, but the minimum quantity for explosibility could not be established by laboratory tests. The ignition temperatures were similar to those for bituminous coal and the explosion pressures and rates of pressure rise were lower. Where sufficient oil shale dust is present underground and where dusts are handled in surface operations, proper precautions against explosion should be taken.

TABLE 6. - Ignitions and explosions in metal and nonmetallic mines

Date	Mine	Ore	State	Type	Ignition source	Killed	Injured
1884	NA	Marble..........	NY	Boiler.....	NA	6	-
1910	Jumbo.............	NA	OK	Methane....	NA	13	-
1920	Jefferson..........	Salt.............	LA	Solvent....	Match.......	6	-
1927	NA	NA	MI	Methane....	Open light..	2	-
1929	Stone Mt.[1].........	Granite..........	GA	Air........	Receiver....	7	-
1932	(3 ignitions)......	NA	AL	Methane....	Open light..	-	-
1943	Boyd..............	Copper..........	TN	Dust.......	Explosives..	8	17
1926-1943[2]	NA	NA	CA	Methane....	Smoking, open light.	5	34
1952	NA	NA	MI	....do.....	Arc.........	5	1
1953	No. 1..............	Gilsonite........	NA	Dust.......	.....do.....	8	3
1960	NA	NA	NA	NA	NA	3	-
1960	West Va. Co........	NA	WY	Methane....	Explosives..	-	-
1962	Dudley[1]...........	Iron.............	AL	Diesel oil.	Welding.....	1	1
1962	White Pine.........	Copper..........	MI	Tank.......	Flame.......	1	1
1963	Belle..............	Limestone........	PA	Methane....	NA	-	2
1963	Cane Creek.........	Potash	UT	....do.....	Arc or flame	18	2
1964	Mineral Kine.......	NA	MO	....do.....	NA	2	-
1964	NA	NA	KY	NA	NA	1	-
1964	Climax.............	Copper..........	CO	NA	NA	1	-
1965	Phelps Dodge.......	.......do........	AR	Natural gas	Flame.......	1	4
1965	Coalton Stone......	NA	KY	Methane....	NA	-	-
1967	Gresham[1]..........	Stone............	NA	....do.....	Welding.....	1	1
1968	Saunders...........	NA	NY	....do.....	NA	-	-
1970	Dead Horse.........	Mercury..........	CA	....do.....	Match.......	-	2
1973	Pacific[1]..........	Asbestos.........	CA	Propane....	Welding.....	-	1
1973	Gall Silica[1]......	Sand.............	FL	....do.....	Smoking.....	-	3
1974	Freeport Kaolin[1]...	Clay.............	GA	Vapor......	Heat........	-	-
1974	Kennecott..........	Copper..........	UT	Acetylene..	Match.......	1	-
1975	Retsof.............	Salt.............	NY	Hydrocarbon	Light bulb..	4	4
1975	Reiland[1]..........	Sand.............	IL	Hydrogen...	Battery.....	-	1
1975	Bossardville[1].....	Limestone........	PA	Oil vapor..	Welding.....	-	1
1975	Mayo Shell[1].......	Oyster shell.....	TX	Natural gas	Match.......	-	1
1976	Marshall[1]..........	Sand.............	MI	Propane....	Smoking.....	-	1
1976	Catnip[1]...........	Limestone........	KY	Oil vapor..	Welding.....	-	1
1976	Millertown A.......	Shale............	TN	Propane....	Flint.......	-	1
1976	Valley Road[1]......	Stone............	NJ	Compressor.	Air.........	-	1
1976	Cleveland Cliff....	Iron.............	MI	Acetylene..	Torch.......	-	2
1976	Belle..............	Limestone........	PA	Methane....	Smoking.....	-	2
1976	Penn Dixie[1].......	Cement...........	IA	....do.....	Kiln........	-	-
1976	Mobile[1]...........	Aluminum refining	AL	Natural gas	Torch.......	-	-
1977	Am. Colloid[1]......	Bentonite........	SD	....do.....	NA	-	1
1977	U.S. Lime Products[1]	Limestone........	AR	Coal dust..	Furnace.....	-	-
1977	Crystal Silica[1]....	Sand.............	GA	Natural gas	NA	-	2
1977	Mobile[1]...........	Aluminum refining	AL	....do.....	Heater......	-	2
1977	Page...............	Barite...........	GA	Ether......	Smoking.....	-	1
1977	Am. Colloid[1]......	Bentonite........	SD	Coal dust..	Flame.......	-	1
1977	Alchem.............	Trona............	WY	Methane....	Arc.........	-	4
1977	Timberline[1].......	Gold.............	CO	Oil vapor..	Welding.....	-	1

NA - Not available.
[1]Surface installation.
[2]Explosions occurred between 1926 and 1943.

The limited data available at this time show that methane may be contained in some of the oil shale and adjacent strata. The quantity depends partly on depth and past fracturing of the beds. Preliminary dust sampling in oil shale mines indicates that the quantity of dust present in passageways may be insufficient for explosion propagation. The effects of size of entry and type of mining have yet to be investigated.

IGNITION SOURCES

Underground Explosions and Ignitions

A principal means for minimizing the explosion hazard in and about mines is the control or elimination of sources of ignition. Data on the ignition sources of recent underground coal mine ignitions and explosions and of major explosions, 1958-69, are summarized in table 7.

TABLE 7. - Ignition sources of recent underground coal mine explosions and ignitions

Ignition source	Major explosions, 1958-69	Ignitions and explosions, 1970-78		
		Ignitions	Explosions	Total ignitions and explosions
Friction	3	285	32	317
Electric arc	8	13	7	20
Welding	-	24	3	27
Explosives	3	8	4	12
Smoking	1	2	6	8
Flame	1	-	-	-
Battery	1	1	2	3
Lightning	1	2	-	2
Safety lamp	-	-	2	2
Unknown	1	-	-	-
Total	18	335	56	391

Of the total of 391 ignitions and explosions which occurred in underground coal mines between 1970 and 1977, 81 percent were caused by friction, 7 percent by welding, 5 percent by electric arc, 3 percent by explosives, and 2 percent by smoking. Lightning and safety lamps each ignited methane in two instances.

The need for improved ventilation practices, degasification, and a means to minimize the frictional sparking from cutting bits at the face is obvious. That relatively few (20) ignitions were caused by electric arcs is gratifying considering the wide extent to which electricity is used in face areas. The 27 ignitions from welding and the 9 ignitions from smoking are troubling, as ignition by these sources should be easily controlled. The ignitions from welding and smoking are identified in tables 8 and 9.

TABLE 8. - Ignitions of methane in underground coal mines by welding, 1970-77

Mine	Company	State	Date
Guyan No. 1	Island Creek	WV	02-06-70
Vesta No. 5	J & L	PA	04-19-71
Itmann No. 3	Itmann	WV	01-04-72
Wabash	AMAX	IL	02-28-73
Cambria No. 33	Bethlehem	PA	05-10-73
No. 32	do	PA	09-19-73
Cambria No. 33	do	PA	01-14-74
Keystone No. 1	EAC	WV	04-12-74
Wente No. 1	Westmoreland	VA	04-29-74
No. 20	Stirrat	WV	09-09-74
Kitt No. 1	Republic	WV	09-19-74
Va. Pocahontas No. 5	Island Creek	VA	03-20-75
Do	do	VA	06-05-75
McKinley	Pittsburg & Midway	NM	09-22-75
C-Shaft No. 5	Island Creek	VA	10-23-75
Keystone No. 1	EAC	WV	02-27-76
Federal No. 2	do	WV	04-23-76
Mulga	Meade Coal	AL	05-13-76
Lady Dunn No. 105	Cannelton	WV	06-29-76
Urling No. 1	R & P	PA	08-23-76
No. 50	U.S. Steel	WV	08-23-76
Kitt No. 1	Republic	WV	08-27-76
Emerald No. 1	Emerald	PA	12-16-76
RonCoal No. 2	Ron	WV	04-18-77
Revloc No. 32	Bethlehem	PA	05-05-77
Beckley No. 1	Ranger Fuel	WV	05-18-77
National Pocahontas	National Mines	WV	05-30-77
Lick Run	Beckley Lick Run	WV	09-23-77

NA - Not available.

TABLE 9. - Ignitions of methane in underground coal mines from smoking, 1970-77

Mine	Company	State	Date
Wanamie	Wanamie	PA	02-27-70
No. 28	(Abandoned)	WV	07-15-70
St. Charles	A&T	KY	10-26-70
Pyro No. 2	Pyro	KY	11-30-70
Dixianne	Coal Processing	VA	02-08-71
Va. Pocahontas No. 4	Island Creek	VA	03-16-71
No. 28	(Abandoned)	WV	06-15-77
St. Charles No. 2	P&P	VA	07-07-77
Terry Glenn No. 1	Terry Glenn	KY	07-22-77

Included in table 7 are the ignition sources of the major explosions between 1958 and 1969. Almost half of these explosions were ignited by electric arc; about 20 percent were ignited by friction and 20 percent by explosives. Despite the fact that continuous mining machines are used extensively underground, a relatively high percentage of the ignitions and explosions are caused by the misuse of explosives.

A further breakdown and description of the common ignition sources for underground explosions and ignitions is given below.

I. Friction:
   A. Frictional sparks from machine bits striking roof, floor, or hard inclusion in coal.
   B. Frictional sparks from drill bit striking hard material.
   C. Drill steel striking iron frame.
   D. Frictional sparks during roof fall from sandstone striking sandstone and other hard rock.

II. Electrical:
   A. Sparking in nonpermissible electric equipment, such as mining machines, pumps, and personnel carriers.
   B. Arcs from battery-operated equipment.
   C. Broken light bulb.
   D. Trailing cable being pulled in two.
   E. Faulty splice in trailing cable.
   F. Arc at trolley.
   G. Intermachine arcing.
   H. Power line arc from roof fall or haulage wreck.
   I. Arc at wheel of vehicle.

III. Flame:
   A. Welding and cutting torch.
   B. Propane torch.
   C. Fire.
   D. Blow torch.
   E. Carbide lamp.
   F. Match (smoking or otherwise).
   G. Cigarette lighter.

IV. Explosives:
   A. Nonpermissible explosives.
   B. Misuse of permissible explosives: overcharged hole, blown out shot, mud-cap shot.
   C. Detonators.
   D. Nonpermissible blasting unit.
   E. Long-delay blasting.

V. Miscellaneous:
   A. Red hot drill bit.
   B. Glowing particles from cutting and welding.
   C. Safety lamp: defective, opening and striking key, or purging with compressed air.
   D. Lightning.

Surface Explosions and Ignitions

A listing of the ignition sources for surface explosions and ignitions is given in table 10 for the periods 1958-69 and 1970-77. In the former period, about one-third of the ignitions were caused by sparks in thermal dryers and preparation plants. These sparks were formed in open-type fires used to heat the drying air. It is very surprising that in the latter period only one out of the 40 ignitions was caused by sparks from these open-flame-type air heaters. One can presume the safety controls on the dryers imposed by the 1969 Act effectively minimize the ignition hazard from this source.

TABLE 10. - Ignition sources of recent surface coal mine explosions and ignitions

Ignition source	Number of explosions and ignitions		
	1958-69	1970-77	Total
Welding	4	18	22
Sparks	19	1	20
Electric arc	2	10	12
Flame	9	1	10
Smoking	1	3	4
Explosives	1	2	3
Lightning	-	1	1
Friction	1	-	1
Unknown	21	4	25
Total	59	40	99

Most alarming is that nearly 50 percent of the ignitions between 1970 and 1977 were caused by welding and cutting. About 25 percent of the ignitions in this period were caused by electric arcs, whereas in the former period ignition by electric arc was negligible. Only 4 of the total of 99 surface ignitions were caused by smoking. Lightning caused one surface ignition.

By reference to table 5, it can be seen that two-thirds of the surface ignitions involved coal dust and one-third methane; whereas in the underground accidents, 95 percent involved methane.

Potential Ignition Sources

Methane-air mixtures require so little energy for ignition that sources such as a miner's pick striking a rock (3, 5), a buffing wheel (46), a pinhole leak in a compressed air line (23), or nails in a miner's boot (5), can all ignite the mixture under some conditions.

The ignition by a pinhole air leak is rather unusual; the jet of air vibrates and heats adjacent material. Ignition of methane was caused by a leak in an air line where a piece of conveyor belting was clamped over the leak. The heat generated by the vibratory motion actually set fire to the belting.

A lighted cigarette might not ignite methane as the temperature of burning tobacco is too low; however, a fragment of the paper wrapper, if turned up and ignited, will ignite the methane. Sparks from a cigarette lighter will readily ignite methane.

Ignitions of methane can also be caused by compressed air tools. Incendive particles may be discharged from the compressed air outlet, the tool may generate excessive frictional heat, the drill bit may become red hot, the air compressor may become red hot, or an explosion of combustible oil vapors may occur in the compressed air line. A major disaster occurred from this latter cause in France (12).

Frictional sparks generated by contact of aluminum and rusty iron are highly incendive, much more so than frictional sparks from metal or stone. Two serious coal mine explosions occurred in Great Britain from this cause. British research showed that incendive sparks could be generated when a car wheel (rusty iron) ran over a candy wrapper (aluminum foil). Aluminum foil wrappers are prohibited in British mines.

The minimum electrical energy to ignite a coal dust cloud is about 70 times greater than the 0.3 millijoules required to ignite methane (8). Experiment showed a coal dust cloud can be ignited directly by frictional sparks (52) in the absence of methane, but the energy required is much higher than that for methane.

Coal dust can be ignited by explosives in the absence of methane. The most recent example of a serious mine explosion from direct ignition by explosives was one at the Nos. 15 and 16 Mines of the Finley Coal Co. in Kentucky on Dec. 30, 1970, in which 38 miners were killed. The explosion was caused by either the detonating cord or dynamite explosives (both nonpermissible). Ignition of methane or coal dust by explosives, including permissible explosives, has been shown by experiments (16) to occur in several ways, including an exposed charge, blown-out shot, blown-through shot of light burden, a crevice in the burden, excessive charge weight, exposure of subsequent charge by the firing of a preceding charge in a round, or the deflagration of a charge.

Statements on Ignitions From Coal Mine Inspectors' Reports

The following excerpts from inspectors' reports are typical statements on the suspected igniting sources of explosions and ignitions in coal mines.

1. Dutch Creek, Mid-Continent Coal & Coke, Colo., Dec. 28, 1956. "Bureau of Mines investigators believe that the explosion originated ...when an explosive mixture of methane and air was ignited by an electric arc from a blown-out temporary splice in a trailing cable, or by sparks from a poorly insulated splice in a trailing cable."

2. Evans Jones Slope, Evans Jones, Ala., Jan. 18, 1957. "The Bureau of Mines investigator believes that the initial explosion originated in a pillar pocket ... when blasting with permissible-type explosives in a nonpermissible manner."

3. No. 34, Pocahontas Fuel Company, Va., Feb. 4, 1957. "Bureau of Mines investigators believe that the explosion originated ... when an explosive mixture of methane-air was ignited by an electric arc or spark from face equipment or a power conductor."

4. Marianna No. 58, Bethlehem Mines Corp., Pa., Apr. 23, 1957. "Bureau of Mines investigators believe that the explosion originated ... when an explosive mixture of methane and air was ignited by arc or spark from the trolley pole of a jeep or when the trolley wire cut-out was opened."

5. Gregory No. 3 Auger, Wolf Summit, W. Va., June 12, 1957. "Reportedly, Grady C. McNabb, employee, entered the borehole [of an auger mine] with a lighted cigarette or it is believed that he lit a match for illumination or to light a cigarette and ignited an explosive methane air mixture."

6. No. 31, Pocahontas Fuel, Va., Dec. 27, 1957. "The explosion originated when an explosive mixture of methane air was ignited by an electric arc or spark from the face electrical equipment or a power conductor."

7. Bishop, Pocahontas Fuel, Va., Oct. 27, 1958. "Bureau of Mines investigators believe that the explosion originated ... where an explosive mixture of methane-air was ignited when shots fired at the face of No. 6 place blasted through to No. 5 place."

8. Burton, Olgebay Norton, W. Va., Oct. 28, 1958. "The gas was ignited by an electric arc or spark initiated when a roof fall caused the power wires in No. 2 entry to contact the return conductor in the frame structure of the belt conveyor."

9. Bird No. 3, Bird Coal, Pa., Feb. 19, 1959. "It is believed that the explosion occurred when methane accumulated in the crosscut was ignited by an arc caused when a shuttle car, energized by an accidental ground, contacted the bottom of the rear conveyor or structure of a continuous mining machine."

10. No. 5, Pitfair, W. Va., Mar. 2, 1959. "The explosion occurred when coal dust in a shear was ignited by improperly confined permissible explosives fired in a borehole which interconnected with a shear at the back of the cut."

11. No. 1, Phillips & West, Tenn., Mar. 23, 1959. "Bureau of Mines investigators believe that the explosion originated ... when an explosive mixture of methane and air was ignited by an electric arc or spark from the trolley wheel of the locomotive or from the lighting of matches or cigarettes."

12. Dutch Creek, Mid-Continent Coal & Coke Company, Colo., Aug. 15, 1959. "The explosion originated in the fan motor room where an explosive mixture of methane-air was ignited by the oxygen-acetylene torch used by the men repairing the fan-motor drive coupling."

13. No. 64 Slope, S&MC company, Pa., Oct. 8, 1959. "This explosion occurred when an explosive mixture of methane at the face ... was moved over a nonpermissible flame safety lamp hanging on a pump a short distance outby the face."

14. Olga No. 1, W. Va., Sept. 11, 1960. "Bureau of Mines investigators believe the explosion originated in and about 1,000 feet outby the face ... when an explosive-air mixture was ignited by an electric arc from the trolley pole of a porta bus."

15. No. 6, Frank Kite, Va., Jan. 16, 1962. "The explosion was caused by a blown-out shot or shots which ignited coal dust thrown into suspension by the preceding shots."

16. Lancashire No. 15, Barnes & Tucker, Pa., Dec. 14, 1962. "It is believed the explosion occurred when gas in a pillared and partially caved area was ignited by friction sparks from a fall of sandstone roof."

17. Cane Creek Mine, Texas Gulf Sulfur, Utah, Aug. 27, 1963. "Bureau of Mines investigators believe the explosion originated in the shop area where an explosive mixture of combustible gases was ignited by electric arcs or sparks, open flame, or heated metal surfaces."

18. Diamond Slope, Mickey Coal Partnership, Pa., June 11, 1964. "An explosive mixture of methane and air ... was ignited by the flame of a propane torch and/or by burning fuse."

19. No. 2A, C. L. Kline, Tenn., June 24, 1965. "Bureau of Mines investigators believe that an explosive mixture of methane and air was ignited by a cigarette lighter at the face of the left air course which was driven 300 feet inby the last open crosscut."

20. Burnwell No. 1, O. A. Pilcher, Colo., Mar. 2, 1966. "An explosive mixture of methane and air was ignited by one of the following: A miner striking a match to light a cigarette; an electric arc or spark in the controller of the storage battery locomotive; a bare spot in the battery cable contacting battery cell terminals of the locomotive; or an arc or spark from an unknown fault in a power conductor."

21. Robena No. 3, U.S. Steel, Pa., June 23, 1966. "The explosion originated in a recently pillared area and was initiated when methane was ignited by frictional sparks created during a pillar fall."

22. No. 7, Horn and Whited, Va., Sept. 6, 1967. "An explosive methane-air mixture was ignited by an electric arc in the control box of a rubber-tired, battery-powered tractor."

23. Lancashire No. 15, Barnes & Tucker, Pa., Sept. 27, 1967. "The explosion was initiated by frictional sparks created when the bits of a continuous mining machine struck pyritic material in the immediate roof."

24. Dekoven No. 6, Pittsburg & Midway, Ky., Nov. 17, 1967. "An accumulation of methane in a large roof cavity over the belt feeders ... was ignited by flame or acetylene torch."

25. West Gulf No. 5, Winding Gulf Coals, W. Va., Jan. 22, 1968. "The explosion was initiated by an electric arc that was created when the trolley-pole slide of a haulage locomotive was removed from the trolley wire in the presence of an explosive mixture of methane and air."

26. Dekoven No. 6, Pittsburg & Midway, Ky., Mar. 11, 1968. "A gas ignition occurred ... when a methane-air mixture was ignited by a blown-out shot of permissible-type explosives."

27. Slab Fork No. 8, Slab Fork, W. Va., July 24, 1968. "The explosion occurred as a result of an explosive mixture of methane-air being ignited by frictional sparks created by carbide-tipped bits of Lee-Norse Miner striking fossiliferous rock in the stratum overlying the coal bed."

28. River Queen No. 1, Peabody, Ky., Aug. 7, 1968. "The evidence indicated that an undetermined, but appreciable, quantity of permissible explosives on a coal drill parked near an entry face was detonated, probably by a fragment projected from the face when the coal was blasted, and that this detonation ignited coal dust...."

29. No. 1, Princess Coals, Ky., July 28, 1969. "Bureau of Mines investigators are of the opinion that the explosion originated ... where an explosive mixture of methane and air was ignited either when a workman attempted to light a cigarette with a cigarette lighter or from an electric arc or spark."

30. Forge Slope, Glen Nan Coal, Pa., Mar. 5, 1970. "Testimony by all four injured men was to the effect that the explosion occurred when the foreman tried to relight his permissible-type flame-safety lamp."

31. Compass No. 2, Clinchfield Coal, W. Va., Apr. 2, 1970. "The explosion was caused when an undetected accumulation of explosive methane-air mixture was ignited by an electric arc or spark created by either an energized dust tight belt conveyor drive motor or the trolley slide of a personnel carrier."

32. River King (surface), Peabody, Ill., June 13, 1970. "A loose connection in a hydraulic oil line caused oil to be expelled in a mist from under the high pressure of the system, and the oil-air mixture was ignited, presumably by the breaking of an incandescent lamp struck by the escaping oil."

33. St. Charles, A & T, Ky., Oct. 26, 1970. "The mixture apparently was ignited by the spark or flame created by a cigarette lighter."

34. Pyro No. 2, Pyro Mining, Ky., Nov. 30, 1970. "Scott sat down and Osburn in a slightly standing position attempted to light a cigarette."

35. Nos. 15 and 16, Finley, Ky., Dec. 30, 1970. "...the explosion occurred when coal dust was thrown into suspension and ignited by primacord or by permissible explosives used in a nonpermissible manner or by use of nonpermissible explosives."

36. Dixianne, Coal Processing Corp., Va., Feb. 8, 1971. "Methane emitted from caved pillared areas ... was ignited by a defective headlight on a continuous mining machine or by open flame from smoking articles."

37. Forge Slope Mine, Glen Nan Coal Co., Pa., Mar. 15, 1971. "A small quantity of methane, generated when the cut of coal was initially blasted, was probably ignited upon reestablishment of the face ventilation by either explosives burning in the blast hole, or by burning explosives detonating and producing a flash that ignited the methane and air mixture."

38. Va. Pocahontas No. 4, Centennial Development, Va., Mar. 16, 1971. "The ignition was caused by smoking."

39. Humphrey No. 7 Mine, Christopher Coal Co., W. Va., June 5, 1971. "Coal dust which was thrown into suspension from blasting stumps and a subsequent roof fall, was ignited when a partial or unconfined shot was fired in the remaining two stumps."

40. Itmann No. 3 (longwall), Itmann Coal, W. Va., Jan. 4, 1972. "The explosion occurred when a massive roof fall forced a dense cloud of coal dust into suspension which was ignited by hot metal from a welding and cutting operation."

41. Mars No. 2, Clinchfield Coal, Va., May 20, 1972. "A coal bump released great quantities of methane and forced coal dust into suspension which was ignited by an electric arc or spark from a power cable to the impedance box [trolley phone signal booster]."

42. Concord No. 1, U.S. Steel Corp., Ala., June 13, 1972. "The explosion occurred when frictional sparks from a crusher ignited gases released from ruptured compressed gas cylinders [MAPP and oxygen] in an area where a small amount of float coal was present."

43. Itmann No. 3, Itmann Coal Co., W. Va., Dec. 16, 1972. "The methane was ignited by an electric arc from a porta bus."

44. Oakwood Red Ash, Va., Sept. 25, 1973. "The methane was ignited by an electric arc from one of several components of a nonpermissible personnel carrier."

45. Guyan No. 5 (surface), Island Creek, W. Va., Jan. 30, 1974. "A surface refuse-pile slide and explosion type accident ... there was a heavy cloud of dust and flames in combination with the slide action."

46. Revloc No. 32 (surface), Bethlehem Mines Corp., Pa., May 5, 1974. "The accident occurred when metal was being cut with a torch at the top of the mine shaft."

47. Cedar Grove No. 1 (abandoned), Zapata Coal Co., W. Va., May 31, 1974. "According to an eyewitness the explosion occurred during a severe electrical storm when lightning struck the belt conveyor structure which was connected to the metal tunnel lining."

48. Captain, Ill., Oct. 21, 1974. "The explosive vapor-air mixture [inside a tire] was ignited by the interior area at the point of the weld."

49. Beatrice Pocahontas (E shaft), VA., Oct. 22, 1974. "Methane accumulated [in the shaft] and was ignited when a splice in the power cable leads heated or arced."

50. Beehive, American Coal Co., Utah, Oct. 30, 1974. "A 100-volt light fixture containing a 300-watt incandescent bulb was smashed down onto the metal frame of a belt conveyor. It is theorized that when the roof collapsed, coal dust was thrown into suspension and was ignited by an electric arc from the damaged light fixture."

51. No. 3, U.S. Pipe & Foundry, Ala., Mar. 6, 1975. "The ignition occurred when a feeder in the right corner of the face was ignited by falling sparks when the rotating head of the Galis roof drill struck the steel channel that was being installed for roof support."

52. Beckley, Beckley Co., W. Va., Aug. 5, 1975. "The ignition occurred when an overheated roof drill was being withdrawn from a drill hole."

53. Wharton No. 4 (surface), Eastern Associates, W. Va., Nov. 9, 1976. "The accident was a result of a back charge of acetylene in the oxygen regulator."

54. No. 131 Preparation Plant (surface), Bethlehem Mines Corp., W. Va., Oct. 12, 1976. "The explosion was caused by steam pressure when the pump was allowed to operate for a period of time while the pump was plugged."

55. No. 2, Ron Coal, W. Va., Apr. 18, 1977. "The accident and resultant injuries occurred because untrained men used defective oxyacetylene equipment."

56. Jenkin Jones Coal Mine (surface), Pocahontas Fuel Co., W. Va., Apr. 20, 1978. "The explosion occurred when fine-coal dust was ignited by either smoldering coal or sparks present in the cyclone collectors or associated duct at or near collectors."

FACTORS AFFECTING PROPAGATION OF EXPLOSION FLAME

Criteria of Propagation

Factors affecting coal dust explosion flame propagation are evaluated primarily by research involving large scale tests in surface galleries or in underground facilities such as the Experimental Mine at Bruceton. Although underground facilities are used in Poland, Germany, England, Russia, and Japan, the following discussion primarily relates to studies by the U.S.

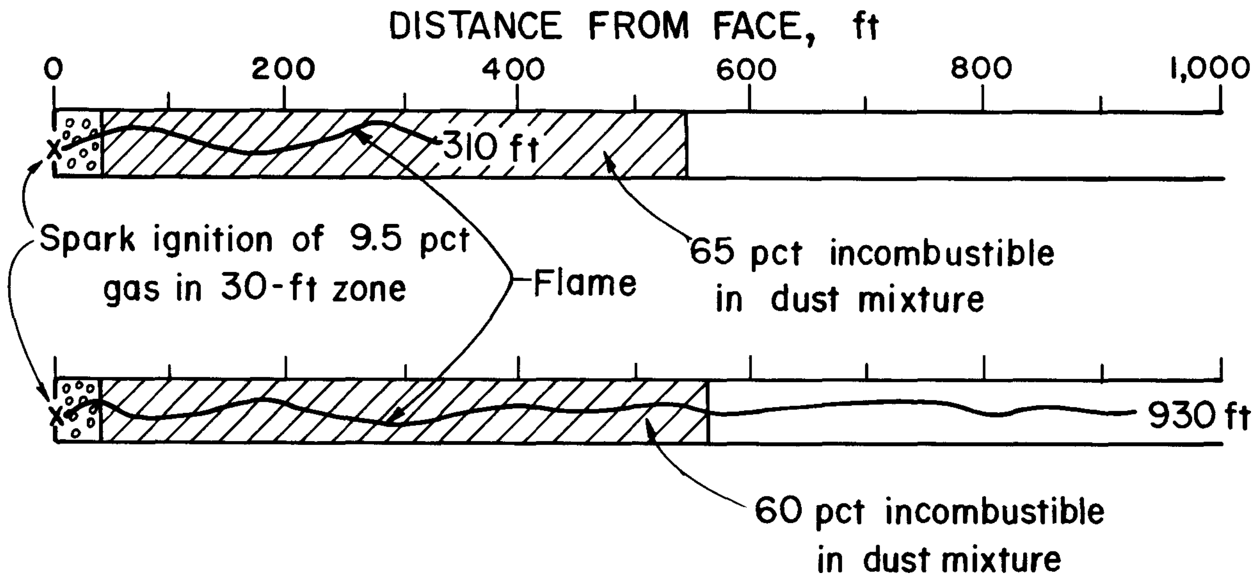

FIGURE 3. - Explosion flame extension in single-entry tests of Pittsburgh coal-limestone dust mixtures.

Bureau of Mines in the Experimental Mine at Bruceton, Pa. A special sense is given the word "propagation" in mine explosion research; a propagating explosion is defined as one in which the flame travels beyond a specific and somewhat arbitrary distance.[3] In an operating mine the word "propagation" is used to characterize the flame travel regardless of the total distance it extends from the point of origin. Hence, all explosions in an operating mine "propagate" whereas in research some explosions "propagate" and some do not.

In the research, the flame may extend several hundred feet into a dust loading but die out before reaching the end of the test zone. Such an explosion is considered not to propagate. Figure 3 is a schematic diagram showing a nonpropagating and a propagating explosion. Note that the flame traveled 310 feet in the nonpropagating explosion whereas it traveled 930 feet in the propagating explosion. The dust loading in both tests extended 550 feet from the face; in most tests this was the chosen distance for establishing propagation.

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[3]A new criterion has been proposed in current studies for "propagation" and "nonpropagation": For a propagating explosion the average gradient of the static explosion pressure and the flame speed shall be zero or positive throughout the length of the test zone. This new criterion is not perfect, since the flame extent in replicate test explosions varies. Nevertheless, the new criterion for differentiating between a propagating and a nonpropagating explosion permits extrapolation of experimental data to longer test zones and stronger igniting sources with greater confidence than the use of the former definition.

The rock-dusting requirement of 65 percent incombustible, as prescribed in the Coal Mine Health and Safety Act of 1969, is based to a large extent on the results of tests similar to those illustrated in figure 3. Therefore, should an explosion occur in an operating mine where conditions duplicated those existing in the nonpropagating test shown in figure 3, the length of flame travel could be expected to be as much as 310 feet before the explosion flame died out. Adherence to the 65-percent incombustible requirement of the 1969 Act provides assurance that the flame extent from an explosion in an operating mine would be limited, but does not assure some explosion flame would not develop.

In applying the results from the research programs to develop the rock-dusting criteria for operating mines, no safety factor was used. This is evident from the data shown in figure 3 where 65 percent incombustible was just sufficient to produce a nonpropagating explosion. If a factor of safety were used in applying the research data, a required value of incombustible higher than 65 percent would be specified.

In the practical mining situation, however, an operator, in order to assure that all samples collected in the mine contain at least 65 percent incombustible, must on the average apply a higher concentration of rock dust because the incombustible content of mine dust in a passageway varies according to a statistical distribution. In this way a safety factor is indirectly provided.

A second indirect safety factor which sometimes comes into effect is that nearly optimum conditions (dust fineness, dust position, dust quantity and strength of igniting source) are used in the experiments. The conditions in an operating mine may or may not be optimum. For example, the igniting source used in the research is usually an accumulation of 1,600 cubic feet of a uniform mixture of methane and air at optimum concentration. With reasonable care such an accumulation of methane should not occur in an operating mine.

Experience has shown that the 65-percent incombustible content for neutralizing mine-size coal dust deposits and the 80-percent incombustible content for neutralizing float-coal deposits, as required by the Federal Coal Mine Health and Safety Act of 1969, are practical values. These incombustible concentrations are not magical values which cause flame to quench instantaneously, but are sufficient to prevent ignition of the dust by an electric arc, spark, open flame, or other weak igniting source and to limit the extent of flame travel through the dust mixture when it is ignited by a stronger source.

Ignition Source

Numerous sources for starting a coal dust explosion have been and are being used in the Experimental Mine in the research studies. All are related to conditions that did or do exist in operating mines.

The strength of any given igniting source is related to the configuration of the ignition zone. An igniting zone developed for a single-entry test has less dust-raising power and flame length if used in a double-entry or in a room configuration.

From 1910 to 1936 most of the explosion tests in the Experimental Mine were double-entry tests; since 1936 most have been in a single entry. Single-entry testing saves time and cost in performing experiments. Rice describes the Experimental Mine and discusses some of the igniting sources used in the double-entry tests (43). Hartmann describes four of the igniting sources used in the single-entry tests (18). The commonly used igniting sources are illustrated in figure 4. Certain of the igniting sources are used repeatedly in the research; these are designated as "standard" igniting sources.

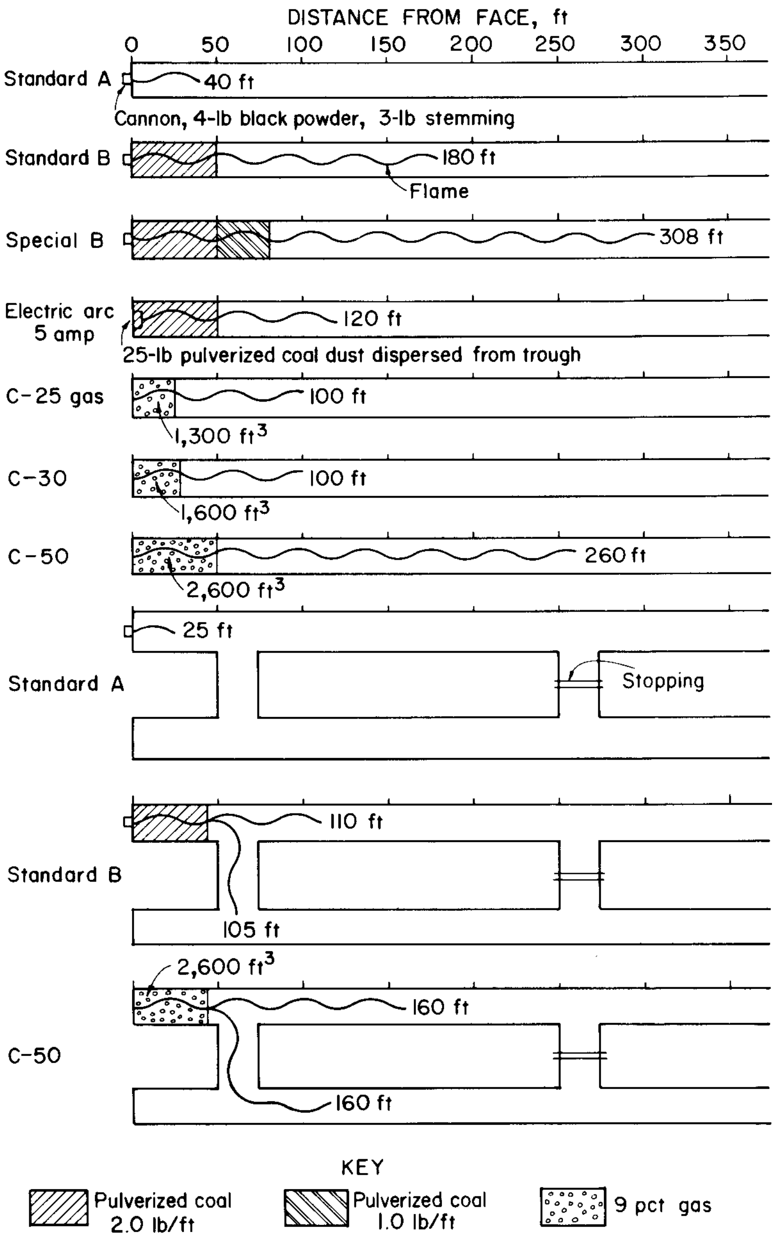

FIGURE 4. - Ignition sources used in Experimental Mine explosion tests.

In 1910 when studies were begun in the Experimental Mine, black blasting powder was commonly used in operating mines for producing coal. Consequently, ignition sources involving black powder were developed for the double-entry tests. The powder is charged in a 3-foot-long, 24-inch-diameter steel cylinder (cannon) grouted into the face of the entry. The borehole of the cannon is 2.25 inches in diameter and 21.5 inches long. When a charge of 4 pounds of black powder stemmed with 3 pounds of clay is ignited, flame projects 25 feet into the entry. This igniting source is termed standard source A. Standard source B uses the same charge but includes 100 pounds of pulverized Pittsburgh coal distributed in a 50-foot zone in front of the cannon. Other weights of black powder charge and stemming, as well as other coal dust loadings, are also used for the cannon source. Source C-50 is a uniform natural gas-air mixture confined by a paper diaphragm in 50 linear feet adjacent to the face of the main entry. The enclosed volume is 2,700 cubic feet. The gas-air mixture is ignited by a 3-ampere spark at the center of the face.

In most instances the gas concentration ranges from 9 to 9.5 percent, but other gas concentrations may be used. The standard cannon sources A and B, as well as the C-50 source for a 9-percent natural gas concentration, are shown in the bottom portion of figure 4.

The commonly used igniting sources for the single-entry experiments are also shown in figure 4. These are the same Standard A and Standard B cannon sources used in the double-entry experiments; Special B, in which an additional 33 pounds of pulverized Pittsburgh coal is added in the 33-foot zone outby the Standard B source; an electric arc (DC) over which 25 pounds of pulverized Pittsburgh coal is dispersed; and the natural gas-air mixtures C-25, C-30, and C-50, where the numerical value indicates the length of the entry containing the gas-air mixture.

The cannon sources are often used for convenience and for comparison with past results, even though black powder is no longer permitted in coal mines.

Several gas igniting zones are used. The gas concentration may be varied from 6 to 12 percent, and the length of gas zone may range from 5 to 50 feet. The length of zone is designated by the number following the letter "C." Each of the above-described igniting sources, when used, causes dispersion of the dust in the test zone and ignition of this dust by the flame of the igniting source.

The "strength" of the igniting source is related to the timeliness and quantity of dust dispersed in the test zone outby the igniting source. A direct measure of the strength of the igniting source is the incombustible required to prevent flame propagation in the test zone. In double-entry tests Standard A, Standard B, and the C-50 sources required 40, 60, and 65 percent incombustible, respectively, for mine-size Pittsburgh coal dust. In single-entry tests the electric arc, Standard B, C-30, and Special B required 59, 65, and 68 percent incombustible, respectively; the C-25 source required 65 percent also, but in a few trials 60 percent incombustible was sufficient. Data on the incombustible content required to arrest explosion propagation, the maximum pressure developed, the length of flame and the energy of the source are given in table 11.

For the ignition sources listed in table 11, the incombustible required ranges from 40 to 68 percent, the flame lengths from 25 to 310 feet, the static pressure from 4 to 40 psig, and the energy released from 4,000 to 450,000 Btu. For practical purposes, standard source B and C-30 (9.0- to 9.5-percent natural gas) in the single entry are equivalent in that the same inert content is required outby in the dust loading to arrest explosion propagation.

TABLE 11. - Ignition sources used in Experimental Mine

Ignition source	Inert to arrest coal dust explosion, percent	Max. static pressure, psig	Flame length, feet	Energy, k/Btu
	DOUBLE-ENTRY TESTS			
Standard A..............	40	4	25	4
Standard B..............	60	6	110	250
C-50[1]...................	65	20	160	250
	SINGLE-ENTRY TESTS			
Standard A..............	NA	10	40	4
Electric arc.............	59	15	120	250
Standard B..............	65	15	120	250
C-25[1]...................	60	17	100	125
C-30[1]...................	65	26	100	150
C-50[1]...................	NA	40	250	250
Special B...............	68	18	310	450

NA - Not available.
[1]9.0- to 9.5-percent natural gas-air mixture.

Coal Dust

Definitions

At the turn of the century when research on the explosibility of coal dust began in earnest, there was much diversity of opinion as to the definition of "dust." George S. Rice (35), one of the most notable American mining engineers, in 1911 writes:

> For the consideration of coal dust as it affects mining, the writer proposes tentatively a definition based on the capacity of the dust to propagate flame in the incipient stages of an explosion, as determined at the Pittsburgh station under the conditions hereafter stated. By this definition coal particles passing through a 20-mesh wire sieve (20 wires to the linear inch) will be termed dust. In the Pittsburgh gallery tests only partial flame propagation was obtained under the prescribed conditions with coal that passed through the 20-mesh and remained on the 40-mesh sieve, but the partial propagation was sufficient to indicate that under slightly more severe conditions, namely, a larger initiating charge of black powder, the propagation might be complete.

Rice (36) states that the tests were made successively on pure coal dust in four sizes: (1) dust passing through 80 and over 100 mesh; (2) through 60 and over 80 mesh; (3) through 40 and 60 mesh; and (4) through 20 and over 40 mesh. Between 1913 and 1918 additional tests were made with coarse particles and the conclusion was as follows:

> Later it was found that for the length of the standard test zone particles larger than those passing through a 20-mesh wire screen (openings between wires of about 1/30 inch square) had no appreciable

influence upon the explosions, so 20-mesh dust was accepted as the upper limit in the size of dust (36).

In 1964 a series of laboratory tests were made with a spark source on aluminum powder and cornstarch (both dusts presenting a more severe explosion hazard than coal dust). It was found that particles passing a U.S.A. Standard No. 40 sieve (particles less than 0.016 inch) did contribute to an explosion in the laboratory bomb. The 0.016-inch particle diameter was recommended as the definition for dust in surface industry. Thus two definitions of dust exist. For coal mines, dust consists of particles passing a U.S.A. Standard No. 20 sieve (particles less than 850 microns) and for surface industries dust consists of particles passing a No. 40 sieve (particles less than 425 microns). The use of two definitions is not incongruous as the potential igniting sources in a coal mine can be much more severe than those in surface industries.

The definition of particle size of dust affects the percentage requirement for inerting which is based on the weight of the dust. The coarser particles in the dust ordinarily contribute a larger proportion to the weight than the finer particles. However, the use of the minus 20-mesh dust particles in the definition of coal dust does not cause undue burden. If the 425-micron diameter definition were used, the required inert concentration would be higher than 65 percent, as is shown by tests in the Experimental Mine.

Several other terms require definition. "Coal mine dust" means solid particles with sizes ranging from sub-microscopic to microscopic, including but not limited to coal dust and rock dust. "Rock dust" means pulverized limestone, dolomite, gypsum, anhydrite, shale, adobe, or other inert material, preferably light colored, 100 per centum of which will pass through a sieve having 20 meshes per linear inch and 70 per centum or more of which will pass through a sieve having 200 meshes per linear inch; the particles of which when wetted and dried will not cohere to form a cake which will not be dispersed into separate particles by a light blast of air; and which does not contain more than 5 per centum of combustible matter or more than a total of 4 per centum of free and combined silica.

"Float coal dust" means the coal dust consisting of particles of coal that can pass a No. 200 sieve. This definition of float coal dust is arbitrary but is consistent with the physical observation that particles of this size can remain suspended in air for some time.

"Mine-size dust" is a term used often in research. It was adopted in about 1925, and means coal dust all of which passes a U.S.A. Standard No. 20 sieve and contains 20 percent minus 200-mesh particles. Mine-size coal dust is a standard size coal used in research. The justification for adopting it is given on page 12 of a Bureau of Mines Technical Paper (41):

> Information on the size of dust in mines was obtained by collecting representative samples from passageways not rock-dusted and subjecting them to screen analysis. Dust from ribs, roof, and timbers were the finest, and 40 to 75 per cent of the material passed

a 200-mesh sieve. Floor dusts were much coarser, and samples contained 5 to 40 per cent of material passing through a 200-mesh sieve. The sizes were found to vary considerably between points in a single mine, and the quantity of dust also varied. An average value is open to criticism; but it was impossible to test all of the sizes found, and no alternative was open. The averages were weighted as far as possible, and for 80 per cent of the mines the final figure ranged from 15 to 25 percent through 200-mesh. Averages higher than 25 per cent were found occasionally and those below 15 per cent rarely. The dust used in explosion tests was of the standard size nearest to or next finer than the weighted average for the mine furnishing the coal. Thus, dust having 20 per cent through 200-mesh was used most in explosion-hazard investigations.

Fineness

The effect of particle size of coal dust on explosibility is best illustrated by the incombustible required to arrest explosion propagation. This is illustrated in figure 5. The curve shows the amount of incombustible required to prevent propagation for coal dust containing 10 to 75 percent particles passing a No. 200 sieve. With 10 percent minus 200-mesh, 55 percent incombustible is required, and with 90 percent minus 200-mesh, 80 percent incombustible is required. In these experiments the coal and limestone dusts were premixed prior to their distribution in the test zone.

Volatile Content

Figure 6 shows the incombustible required to arrest explosion propagation of mine-size and pulverized coal dusts having volatile ratios ranging from 0.06 to 0.49. The volatile ratio is calculated from the prox analysis:

$$\text{volatile ratio} = \frac{\text{volatile content}}{\text{volatile content} + \text{fixed carbon}}$$

This method for calculating the volatile ratio produces a value independent of the natural or added incombustible in the coal. The natural ash is added to the incombustible of the admixed rock dust to give the total incombustible in the dust. The curves show that coals having a volatile ratio less than 0.12 do not propagate explosion even without added inert. Those coals having a volatile ratio less than 0.12 are anthracite, for which rock-dusting is not required. The curves rise sharply from a ratio of 0.12 to about 0.20. Above the value of 0.20 the curves are relatively flat. These curves show that each coal, depending on its volatile ratio, requires a specific value of incombustible to prevent explosion propagation. The decision to require all coal dusts except anthracite to have 65 percent incombustible was made in 1927 by the Mine Safety Board.

The Mine Safety Board was established in 1924 by the Director of the Bureau of Mines to make decisions and set policy. Decision No. 5 relating to rock-dusting was published in 1927 (40) and this was superseded and

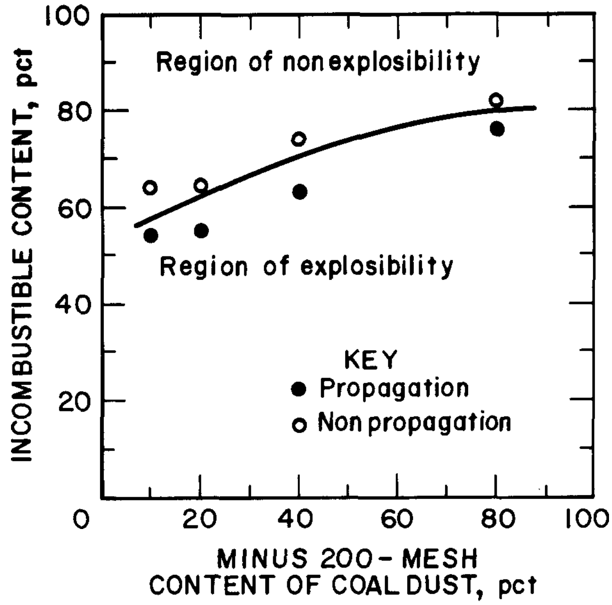

FIGURE 5. - Effect of particle size of coal dust on explosibility.

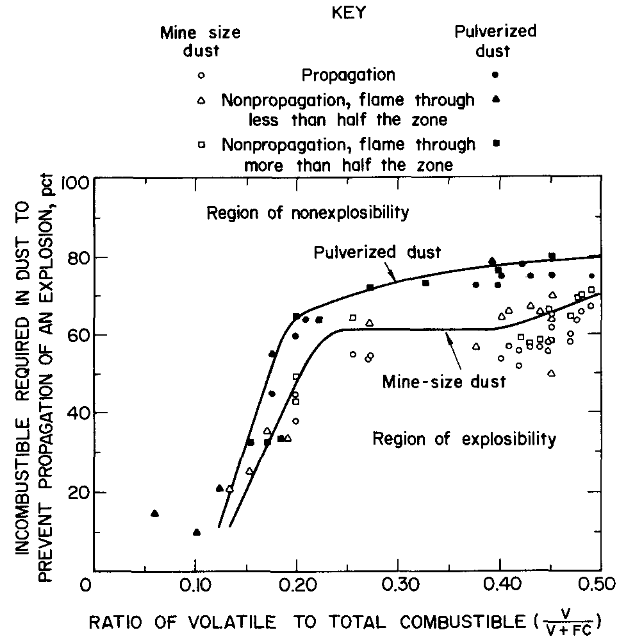

FIGURE 6. - Effect of volatile ratio of coals on explosibility.

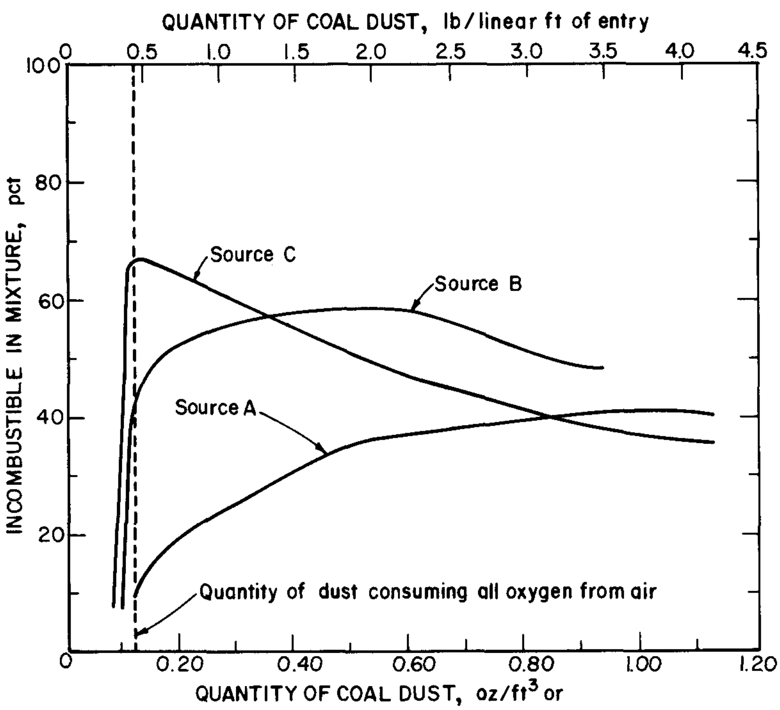

FIGURE 7. - Effect of quantity of coal dust in a mine passageway on percent incombustible required to prevent explosion propagation for three igniting sources.

clarified by Decision No. 32 (27). All Federal mine codes and laws since then have contained the same requirement. The requirement to have a 65 percent incombustible content for all coals but anthracite was made to simplify rock-dusting practices. Coals having a volatile ratio of less than 0.20 are provided a greater degree of explosion protection than coals having volatile ratios higher than 0.20.

Quantity of Dust

The effect of quantity of coal dust lying along a passageway on the incombustible required to prevent propagation is shown in figure 7. The data on coal dust loading are shown as ounces per cubic foot and pounds per linear foot. The concentration value (oz/cu ft) applies to any mine passageway; whereas the quantity value (lb/linear ft) applies only to the Experimental Mine entry where the average entry width and height are 9.6 feet and 6.3 feet (60 square feet).

The data for figure 7 were obtained in double-entry tests with mine-size dust for ignition sources A, B, and for C-50 (9.0 natural gas-air)(43). The curves illustrate the relative dust-raising strength of the three igniting sources. Source C-50 develops a sharp, intense pressure pulse with air movement which effectively disperses the dust outby the ignition zone into the air, affecting explosion propagation. Standard source A produces a strong shock wave but has low air movement and poor dust-raising power; thus the optimum dust quantity for standard source A is 10 times higher than that for source C-50. The dust-raising power of standard source B is between that for source C-50 and standard source A.

An incombustible content of 65 percent is required for source C-50 at a dust loading equivalent to 0.13 oz/cu ft or at a loading of 0.5 pound of dust per linear foot of entry. With greater quantities of coal dust, a lesser percentage of inert is required to arrest propagation. For example, at 0.5 oz/cu ft (1.9 lb/ft) 50 percent incombustible is required. If more than the optimum quantity of coal dust is present, the excess quantity dispersed into the air absorbs heat and cools the flame. In a general way the effect of quantity of coal is the same for standard sources A and B as for source C-50.

The lower explosive limits of mine-size and pulverized coals suspended in air, determined using source C-50 in double-entry tests, are 0.09 and 0.05 oz/cu ft, respectively. Tests made in laboratory show that the presence of methane in the atmosphere produces a linear reduction in the value of the lower explosive limit for minus 200-mesh coal dust. These data are shown in figure 8. Thus, for example, if the mine air contains 3 percent methane the quantity of minus 200-mesh coal dust required for an explosion is only 0.03 oz/cu ft.

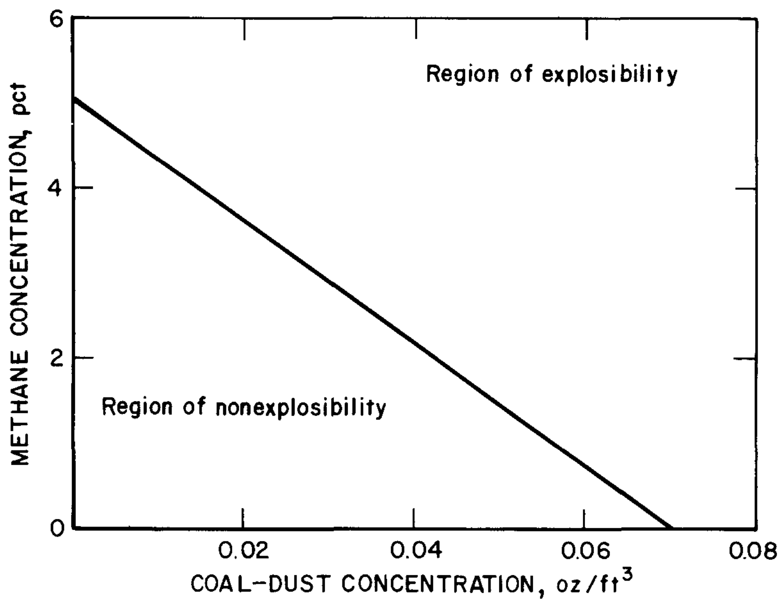

FIGURE 8. - Reduction in lower explosive limit of coal dust by methane in the atmosphere.

Experiments were made in 1952 in the Experimental Mine to determine the visibility through a coal dust cloud at the lower explosive limit. The general

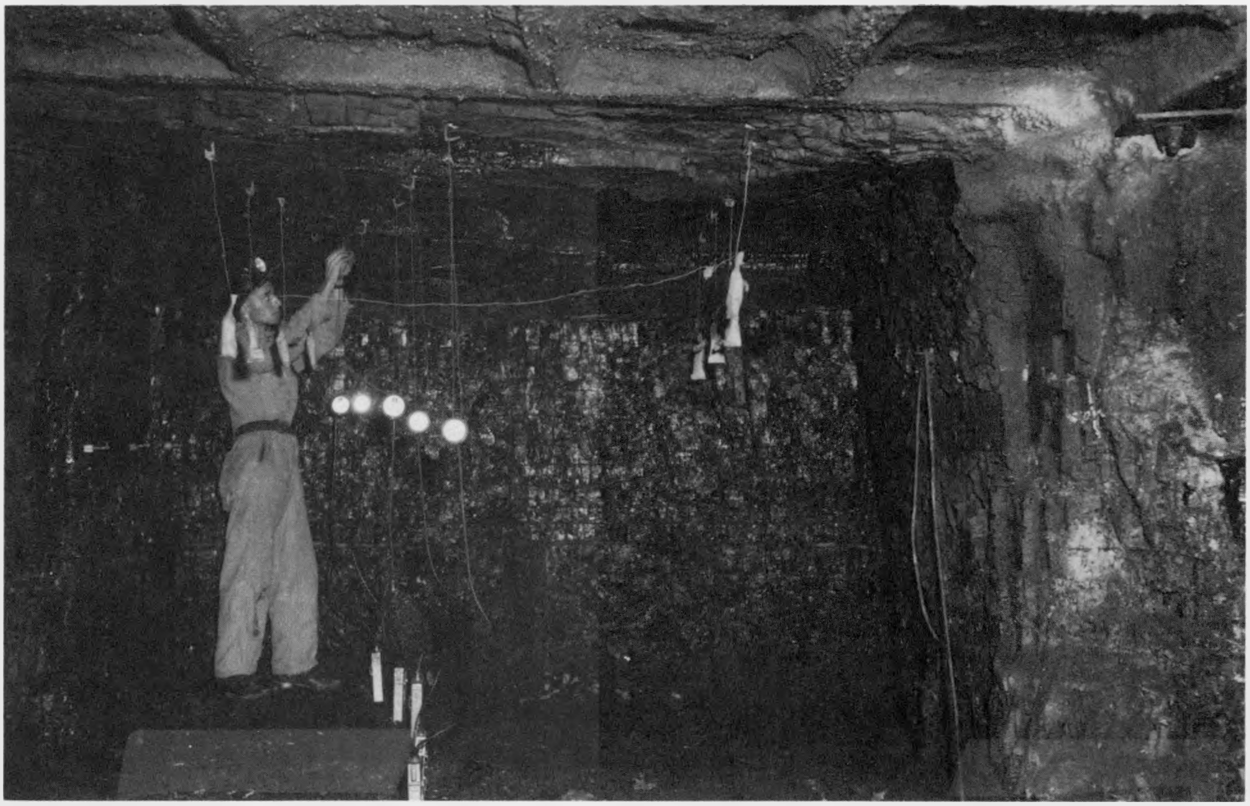

FIGURE 9. - Visibility through a coal dust cloud at the lower explosive limit. Top, Cap lamp arrangement prior to dust dispersion. Bottom, Cap lamps 10 seconds after dust dispersion.

arrangement of the test is shown in the top view of figure 9. When pulverized coal dust at 0.05-oz/cu ft concentration was dispersed in an entry, a cap lamp 10 feet within the cloud was not visible to observers standing in front of the dispersed dust. A cap lamp 4 feet within the cloud appeared to be only 25 percent as bright as normal. The bottom picture in figure 9 shows the cap lamps 10 seconds after dust dispersion when some of the dust had settled.

In this and other experiments with dispersed coal dust, it was found that a person cannot breathe in an atmosphere containing dust at the lower explosive concentration. The heavy dust cloud produces a suffocating atmosphere, forcing a person to retire to a less dusty atmosphere. The amount of dust in the air at the lower explosive limit is 25,000 times greater than the average concentration of respirable dust to which a coal miner may be exposed over a shift under current Federal regulations, 2.0 mg/cu m.

There are few if any locations in an operating coal mine that do not contain sufficient coal dust to propagate an explosion if the dust were to be

FIGURE 10. - Special coal dust loading for upper limit determination.

dispersed into the air. The lower explosive limit of pulverized dust suspended in air of 0.05 oz/cu ft is equivalent to $\frac{0.05}{16} \times 60 = 0.19$ pound per linear foot in the Experimental Mine where the cross-sectional area is 60 square feet. When this amount of dust is spread uniformly on the floor, the loading per square foot is $\frac{0.19}{9.5} = 0.02$ lb/sq ft. As the apparent density of pulverized coal is 46 lb/cu ft, the thickness of the dust layer on the floor would be $\frac{0.02 \times 12}{46} = 0.005$ inch.

If the same amount of dust were spread evenly on the floor, rib, and roof surfaces, the dust loading per square foot would be $\frac{0.19}{2 \times 9.5 + 2 \times 6.3} = 0.006$ lb/sq ft and thickness of the deposit would be $\frac{0.006 \times 12}{46} = 0.002$ inch. Such thin coal dust deposits are almost unobservable.

Experimental tests have shown an upper explosive limit exists for coal dust just as for the methane fuel. The tests (31) with coal were made in the single entry of the Experimental Mine with mine-size dust and with source B ignition. The pure coal dust loading was placed on transverse board shelves in the upper two-thirds of the entry as shown in figure 10. With a loading of 3.8 oz/cu ft a low-velocity explosion propagated, and with a loading of 5 oz/cu ft flame was arrested within a 10-foot distance outby the ignition source. No tests were made with pulverized dust, but it could be expected that the limiting value for the finer dust would be less than 5 oz/cu ft; Cybulski (6) reports an upper limit value of 1 oz/cu ft for pulverized coal.

It should be noted that the coal dust loading for these tests was in the upper two-thirds portion of the entry. An equal or larger quantity of dust loading on the floor would not be raised into suspension to quench the flame. In fact, tests were made with 17 oz/cu ft coal dust, in heaped piles on the floor (see fig. 11) and propagation was obtained. In an operating coal mine thick coal deposits on the floor will not be dispersed to provide a concentration above the upper explosive limit.

Position of Dust

The position of coal dust along the perimeter of an entry is a more important factor affecting explosion propagation than is commonly recognized. The dust on the ribs, roof, or other elevated surfaces (overhead dust) is dispersed by explosion much more readily than dust on the floor. If the overhead dust is coal dust, the explosion hazard is intensified. If the overhead dust is primarily rock dust, the explosion hazard is reduced. Depending on quantity, the overhead rock dust compensates for a deficiency of rock dust on the floor. The converse is not true.

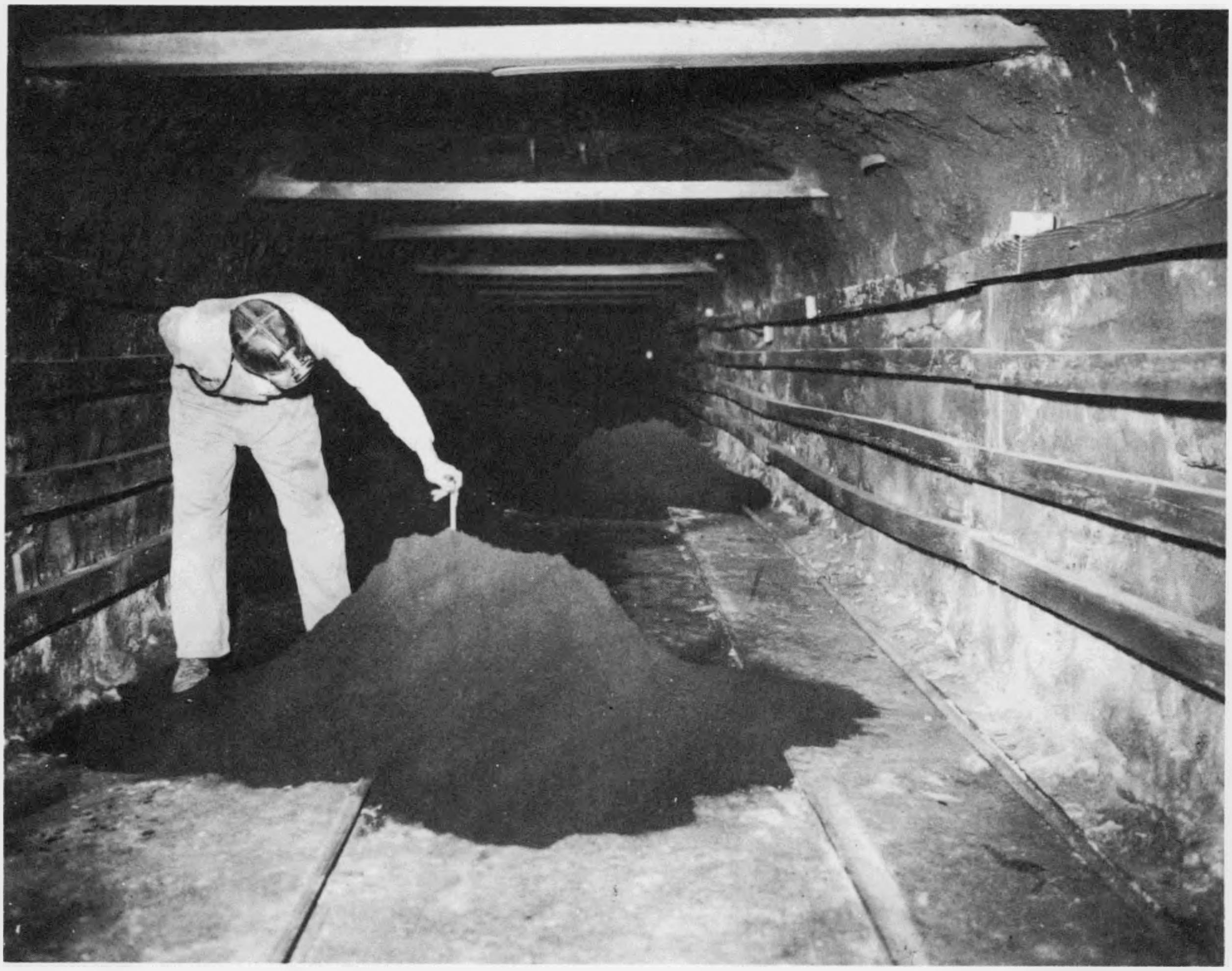

FIGURE 11. - Dust heaps on mine floor for explosion test.

In tests in the Experimental Mine, early recognition was given to the effect of position of the dust and a standard distribution of the dust loading was developed. For standard distribution, one-third of the dust is placed on the floor, one-third on side shelves along both ribs and one-third on overhead cross shelves which are about 1 foot below the roof. In operating mines where continuous mining machines produce a smooth surface and where crossbars or other roof supports are not used, the quantity of overhead dust may be small. Roof supports of timbers or steel beams and plates provide surfaces where coal dust can accumulate. Rough ribs contain copious quantities of coal dust.

A dust explosion will propagate if coal dust is present only on the floor. In single-entry tests with the electric arc source, only 41 percent incombustible was required to arrest propagation with 90 percent of the dust placed on the floor. With standard distribution, 59 percent incombustible was required (18). In similar tests with the C-25 source, 44 percent incombustible was required with 90 percent of the dust on the floor; whereas 64 percent was required with the standard loading. Similar results are reported in the referenced paper for Standard B and Special B igniting sources.

"Blanket rock-dusting" is the term used to describe the practice of placing copious quantities of rock dust on the floor, usually by breaking bags of rock dust with a shovel and neglecting rock dust application to the overhead surface. In a single-entry test (20) with Source B ignition, pure rock dust at the rate of 12 pounds per linear foot was placed on the floor and pure coal dust was placed on the overhead surfaces. Strong explosion propagation was obtained even though the average incombustible in the dust loading was 96 percent. The results of these tests effectively illustrate the greater dispersibility of overhead dust.

Although "dustless" zones do not exist in operating mines, experimental trials were made to determine if a dustless zone would affect explosion propagation. Dustless zones are achieved in the Experimental Mine by cleaning with compressed air jets. The tests (20) showed that a dustless zone several hundred feet in length offers no protection against a developed explosion, as dust is carried into the zone by the moving air stream.

Double-entry tests (41) were made in 1925 with mixed dust loadings on the floor or on overhead surfaces and with strips of coal dust along the entries simulating spillage from coal haulage. Conclusions from this work are as follows:

1. 10 to 15 percent more incombustible was required to arrest propagation when dust loading was standard (1/3, 1/3, 1/3) than when the dust was on the floor and ribs alone.

2. With 40 percent rock dust in the mixture, propagation was not obtained with dust placed only on the floor, only on the side shelves or only on the overhead shelves, but flame extent was least with the dust on the floor. Propagation was not obtained with dust loading placed on the side shelves and floor, and propagation was incomplete (propagated in one entry) when this mixed dust was placed either on the overhead shelves or on the floor. Propagation was obtained however, with the dust on both the floor and overhead shelves. For standard dust loading 65 percent incombustible is required to arrest flame propagation.

3. Pure coal was placed on the floor and pure rock dust was placed on the side shelves (average incombustible content of 80 percent). Propagation was obtained as the rock dust at the ribs did not neutralize the coal dust on the floor. In related tests coal dust in a strip along the center of the entry was not rendered safe by inert dust at the ribs.

Float Dust

Research data on the occurrence, transport, explosion hazard and control measures of float coal dust were published in 1965 (30).

Float coal becomes airborne at the working face where coal is produced, or at any other location where coal is transferred. The average particle size varies with the time in suspension and the turbulence of the air current. An approximate average particle diameter is 20 microns. The production of float dust at the face varies with the type of mining machine, coal seam, and use of water sprays or other dust abatement procedures. As reported in 1965, the quantity observed in 13 operating mines outby the last open crosscut ranged from 0.01 to 0.18 pound per ton of coal mined; the average value was 0.1 pound of float coal per ton. Float coal is transported by the ventilating air and deposits on the rib, roof, floor, and other appurtenances in the mine. Less than 1 percent adheres to the roof, less than 5 percent deposits on the ribs, and more than 94 percent deposits on the floor. The deposit on the floor is relatively uniform across the width of the entry. The quantities of float coal deposited within 500 feet from the source are 74, 69, and 58 percent for air velocities of 100, 200, and 300 ft/min, respectively; the corresponding values within 100 feet of the source are 65, 56, and 40 percent, respectively. Because the face in an operating coal mine is continually advancing, uniformly thick float coal deposits are found in return airways independent of the distance to the working face.

Float coal presents a serious explosion hazard for the following reasons:

1. Float coal dust is very fine in particle size.

2. Float coal dust is easily dispersed and ignited.

3. Float coal overlays rock-dusted surfaces.

4. The quantity of float coal present cannot be readily assessed.

As shown in figure 5, 80 percent incombustible is required to arrest propagation for pulverized dust, which is slightly coarser than float coal dust. In this determination the coal and rock dusts were premixed. With the float coal dust on top of mine dust the float dust is dispersed preferentially by the explosion. With a strong igniting source, C-30 single-entry or C-50 double-entry, generally some of the mine dust beneath the float coal is dispersed, and if sufficient incombustible is present (80 percent or more) the explosion flame is arrested. With a weak igniting source such as a 5- to 6-percent methane-air mixture, or with layered methane, a lesser amount of the underlying dust is dispersed and a strong explosion may develop. In the Experimental Mine tests more than 90 percent incombustible was required to arrest flame propagation when the weak igniting source was used. The tests showed that if the float coal deposit was "light," meaning a float coal dust layer amounting to 0.1 oz/cu ft or less, the hazard could be neutralized by the underlying rock dust. Heavy float coal dust deposits, those exceeding 0.1 oz/cu ft, could not be neutralized by the underlying rock dust.

The explosion hazard of float coal dust was forcefully demonstrated to the mining public and Federal coal mine inspectors by a demonstration at the Experimental Mine on Apr. 1, 1969. In the demonstration, 475 pounds of float coal were applied over 1,285 pounds of limestone dust in an 830-foot length of entry. The average incombustible content was 75.5 percent. The quantity of float coal was equivalent to 0.15 oz/cu ft and is considered to be heavy loading. The explosion was initiated by flame from a 1,300-cubic-foot volume of 6.5-percent natural gas-air mixture in a 25-foot zone, ignited by a spark at the face. This weak igniting source developed a static pressure of 2.5 psig. When the gas mixture was ignited, a very violent float dust explosion developed. The flame speed exceeded 3,000 ft/sec and the static pressure exceeded 75 psig. Flame extended the full length of the 1,308-foot entry and projected a short distance on the surface. An intense shock wave developed which was heard 30 miles away. Window panes were broken 7 miles from the station. The surface damage was intensified by an atmospheric temperature inversion which reflected and focused the shock wave.

Float coal deposits can be neutralized by new applications of rock dust, by use of trickle rock-dusting, by mixing the float coal with underlying rock dust, by general clean-up, and by washing the rib and roof surface. Currently research is in progress by the Bureau of Mines to evaluate control by a salt encrustation method.

Inerting

Rock Dust

The effectiveness of rock dust in arresting explosion propagation has been proved by experiment and practice. The precise mechanism by which rock dust (generally limestone dust) quenches flame has not been fully explained, but is believed to be absorption of thermal energy from the heated gases and absorption of radiant energy which reduces the preheating of unburnt coal particles ahead of the flame front. Limestone dust is not a chemical flame inhibitor although some decomposition does take place.

The effect of particle size of limestone was reported in 1933 (43) and is shown in figure 12. The curve shows that the particle size of the rock dust is not critical, but the most effective material is one in which about 70 percent passes a No. 200 sieve. Practical experience has shown that superfine rock dust tends to agglomerate.

The effect of methane in the air current on the rock dust required to prevent explosion flame propagation was studied in 1929 (41). The tests show a linear increase in the incombustible between that required with no methane in the air and 100 percent incombustible required with 5 percent methane in the air current.

Wet Rock-Dusting

In mechanized mines, where dry rock-dusting by machine is difficult because of the dust raised into the air, rock dust can be applied to the floor by shovel without raising appreciable dust. This does not provide protection for the rib-roof coal dust. In 1955 tests (20) were made with limestone dust and water premixed to form a slurry and with the rock dust and water fed through a nozzle for mixing the dust and water. The experiments showed the following:

1. A premixed slurry of limestone dust and water applied through a nozzle disperses very little dust in the air. About 7 to 8 gallons of water per 100 pounds of limestone was required to form a free-flowing slurry.

2. Equally effective was limestone dust mixed with water at the nozzle. Six gallons of water mixed with 100 pounds of limestone dust was found to give a satisfactory mixture for nozzle application.

3. At least 4 pounds of wetted rock dust was needed per linear foot of entry to cover the ribs and roof completely.

4. About 80 to 85 percent of the wetted rock dust adhered to the rib and roof surfaces. In comparison, during normal dry rock-dusting by machine at most only 30 to 35 percent adheres to the rib and roof.

5. During dry rock-dusting the airborne dust 25 feet downstream was as high as 5,000 million particles per cubic foot; 100 feet downstream the count was about 3,000 million particles per cubic foot. When the slurry was applied, the dust count was less than 0.5 percent of the above values; and when limestone dust and water were mixed by a nozzle, the dust count ranged from 1 to 10 percent for the above values.

6. The time required for wetted rock dust to dry depends on the relative humidity and the airflow. At normal airflow, when the relative humidity of the air was below 80 percent, the rock dust dried in 1 to 3 days; when the humidity was 80 and 90 percent the dust dried in about 1 week; and at still higher humidities several weeks were required.

7. Surface-treated, moisture-resistant limestone was considerably more dispersible after drying than ordinary limestone. In an effort to increase the dispersibility of ordinary limestone, which remained somewhat caked after drying, about 25 different powders were mixed with the limestone before wetting. Several of them appeared to increase the dispersibility after drying, but none was completely effective in preventing caking of the limestone. Single-entry explosion tests were made to determine the effectiveness of the wetted limestone dust.

The overall results of the investigation were as follows:

1. Dry rock dust distributed by machine was more effective than wetted rock dust.

2. Wetted limestone dust was more effective after complete drying than after partial drying.

3. Dry rock dust distributed on the floor and wetted rock dust applied to the rib-roof surfaces provide protection against explosion. When the rock-dusted zones started 50 feet from the face, explosions were arrested readily except when the rock-dusted surfaces were covered with a heavy layer of float coal dust.

4. When the rock-dusted zones started 100 feet outby the face, difficulty was encountered in stopping explosions either by dry or by wetted rock dust application.

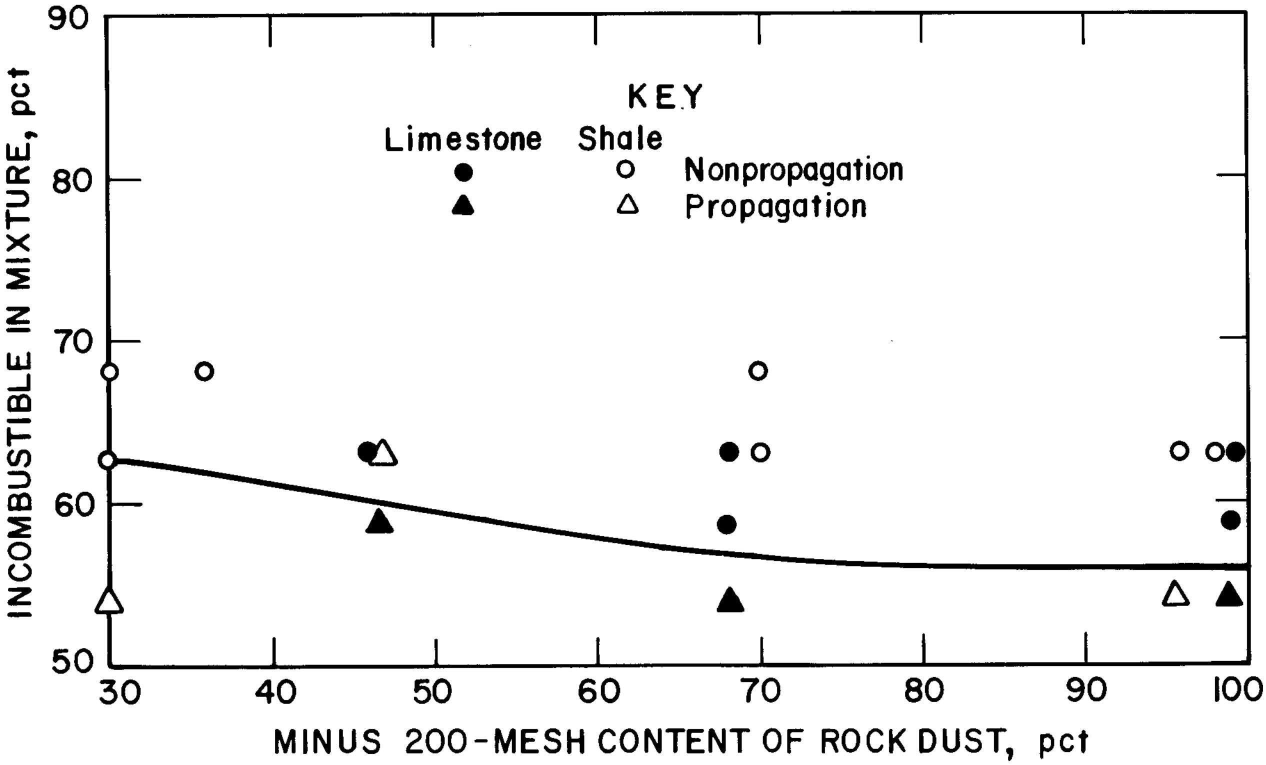

FIGURE 12. - Effect of particle size of rock dust on incombustible required to arrest explosion propagation.

Water

The effectiveness of water for inerting coal dust was studied in 1962 (28). In general, water is 2.2 times more effective on a weight basis than limestone dust. Whereas 65 and 80 percent limestone dust are required in limiting mixtures of mine size and pulverized coal dust, 30 and 36 percent water provided equal protection. The relative effectiveness of water and rock dust is consistent with the ratio of specific heats of steam (0.5) and limestone dust (0.22), indicating that absorption of heat is the primary quenching action. The quantity of water and limestone dust required in mixtures of mine-size and pulverized coal dusts is shown in figure 13. As pointed out in the research paper, coal dust does not readily absorb water. To achieve the limiting mixture of water needed with coal dust, a wetting agent had to be used.

Pulverized coal dust containing 30 or more percent water is a fluid mixture having the consistency of ketchup. To effectively neutralize the explosion hazard, all of the coal dust in the entry must be wetted to this consistency. Float coal dust does not mix with water; in fact, float coal dust floats on water even though coal has a greater specific gravity than water.

Water, though effective in neutralizing the explosion hazard of coal dust, is not recommended as the sole safeguard for the following reasons:

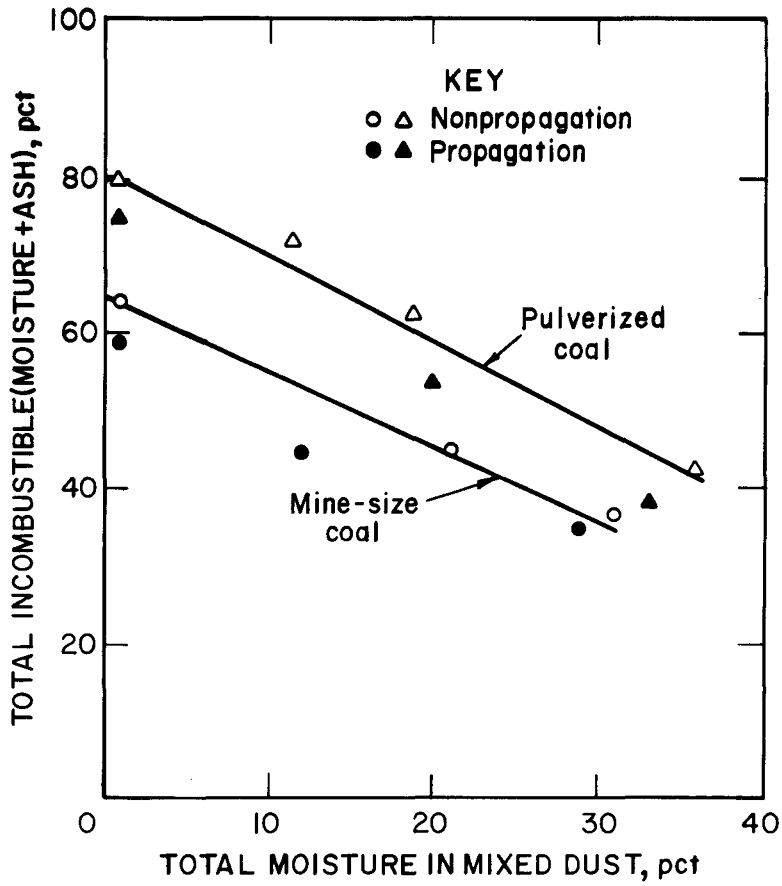

FIGURE 13. - Limiting incombustible (moisture and ash) required to arrest explosion propagation.

1. Water evaporates readily from moistened coal. Thus, in a passageway where adequate water exists in the dust, changes in the weather or the ventilation system could dry the dust and make it unsafe in a short period of time. Where adequate rock dust has been applied, drying is not a factor.

2. The water content of mine dust along an entry or in floor or rib-roof deposits may vary appreciably. There may be sufficient moisture at one location and a deficiency a short distance away. With generalized rock-dusting applied systematically, variation in incombustible content is less likely.

3. Coal dust, although not wetted by water, adheres better to wet than to dry rib-roof surfaces. Fresh coal depositing from the air current will remain dry even though the undersurface is wet; hence, the explosion hazard in a wet area may be intensified.

4. Visual observation is a poor method for estimating the moisture content of mine dusts. The general tendency is to overestimate the moisture content. As an example, an experienced mining engineer was asked to collect wet, wet-to-damp, and damp-to-dry mine dust samples. Analysis showed the respective samples to contain 11, 8, and 4.4 percent moisture. These dusts are in the dry range.

5. Where methane is present in the ventilating current, the water inert does not compensate against this additional hazard and generalized rock-dusting must be used.

6. Standing pools of water in a passageway do not provide protection against explosion. Water on the floor rarely is dispersed effectively to assist in quenching explosion flame. For example, in the Kinlock Mine explosion of Mar. 21, 1929, explosion flame traversed passageways in which there were 200 or more feet of standing pools of water on the floor.

7. Moisture from the ventilating-air current will not make coal dust wet even though the humidity is high. Wetting agents are effective in increasing the rate of absorption of water by dust.

Other results from the study of explosion control showed the following:

1. Excessive deposits of coal dusts on the rib and roof surfaces can be washed to the floor by water. During the washing operation only some of the coal dust becomes wetted.

2. The use of rock dust or water barriers should be considered where it is difficult to rock-dust in wet locations.

3. In sampling mine dust for adequacy of moisture, separate rib-roof and floor samples should be collected. A moisture deficiency in either dust sample indicates that rock dust should be applied.

Experience has shown that mine explosions occur at a higher frequency in winter than summer, primarily because in winter the mine atmosphere (and surfaces) contain less water; the coal dust is more dispersible, has less inert content, and requires less energy for ignition.

Most mine explosions start in face areas. Any measure which reduces the ease of ignition and inhibits flame propagation is a highly desirable safety precaution. Water sprayed on the face and in the face area is such a precautionary measure; its effectiveness is increased when a wetting agent is used. Water applied to mine surfaces containing coal dust measurably reduces the dispersibility of coal dust, adds inert to the coal dust and increases the energy required for ignition of the coal dust. Such applied water, however, will not eliminate the explosion hazard.

Barriers

Barriers are used rather extensively in Europe for coal dust explosion control. The records show a greater proportion of failures than confirmed successes; however, most of the failures were due to improper location, improper design, poor installation, and inadequate maintenance. Propagation of a coal dust explosion can be arrested by a properly designed, located and maintained barrier. In the present state of technology, barriers should be at least 200 feet from the ignition point to insure operation. This has the disadvantage that it permits an explosion to develop destructive force.

Current tests in the Experimental Mine indicate that a conveyor belt containing coal dust, in an entry otherwise containing 65 percent incombustible, renders the entry unsafe against explosions. Similar tests showed that parked mine cars topped with coal dust present an explosion hazard. In longwall, shortwall, and retreat mining systems, the equivalent of 100 to 800 linear feet of non-rock-dusted entry exists. Unless adequate wetting or other dust control measures are used, coal dust may be deposited at the coal transfer points, making generalized rock-dusting ineffective. These are some locations in coal mines where it is difficult to maintain effective explosion control by generalized rock dusting, and rock dust or water barriers may be the solution.

Elementary forms of barriers were tested in France as early as 1911. More sophisticated versions were developed in Poland and Germany. Considerable research was conducted in the United States during the 1920's (42), and the requirements for a successful barrier installation are now being studied. Research is also being conducted by the Bureau of Mines on a quenching device which is a special type of barrier designed to quench a methane ignition.

Salt

Many inert materials for explosion control have been studied in the laboratory and a few have been tested in the Experimental Mine. Two chemicals which have merit are salt (sodium chloride) and ammonium phosphate. Both of these inhibit flame development by chemical action; hence, lesser quantities are needed than with limestone dust. Salt as a substitute for limestone dust was examined in the 1930's (11). Salt was 2 to 6 times more effective on a weight basis than limestone, depending on the limestone-salt proportion. Salt may be economically attractive, but is hygroscopic and may be corrosive. Currently, the Bureau of Mines is studying means to overcome these disadvantages.

A different type of application of salt for explosion control has been used in Europe, particularly in Germany, where the method of salt encrustation was developed. A brine solution is applied to the rib and roof surfaces. As the water evaporates the salt crystallizes, entrapping the underlying coal dust and coal dust which deposits during the crystallization. When most of the water has evaporated, a new brine solution is applied. Coal dust, including float coal particles, is bound tightly in the crystals. For the salt encrustation method to be effective, the relative humidity should be less than 75 percent; otherwise, crystallization does not occur. In most mines in the United States the relative humidity is above 75 percent, even in winter months. This system for dust control is being studied by the Bureau of Mines.

Ammonium phosphate, the principal ingredient in all-purpose dry-chemical fire extinguishers, has been tested for neutralizing the coal dust explosion hazard on shuttle car roadways (32). Initial tests had shown that with adequately rock-dusted rib and roof surfaces, 26 percent water in the coal dust on the roadway was sufficient to arrest explosion flame propagation. However to procure the 26-percent water content, the dust on the floor has to be sloppy wet, a condition which cannot be tolerated for safe movement of vehicles. An attempt was made to use an ammonium phosphate solution to inert the floor dust. The tests showed that 15-percent water retention was required in the floor dust if an ammonium phosphate solution were used. With a 15-percent water retention in the roadway dust, the roadway was usable for shuttle cars. This system has not been used in operating mines for the following reasons:

1. The initial findings could not be verified by repeated tests.

2. All of the dust on the floor had to be treated with the solution and there is no assurance that such treatment could be maintained in an operating mine.

3. No practical method has been developed for sampling and testing to assure that the minimum quantity of ammonium phosphate is present in the floor dust. With this system the visual appearance of treated and untreated floor dust are the same.

Miscellaneous Factors

Binders

The concept of applying an adhesive binder in mine passageways to consolidate the coal dust and bind it to the mine surface to prevent dispersion evolved in the 1920's. Initial trials were made with sulfite waste liquor, a byproduct of the paper industry; later sodium silicate (water glass) and about a dozen specific chemicals were used. With almost all of these materials, laboratory trials show strong binding of coal dust, particularly when a wetting agent is used to assist penetration of the solution. Bound dust resembling shoe leather is readily obtained.

A large-scale explosion test (32) was made in 1970 on the most promising of the adhesive binders, a special latex wetting-binding agent. The results of the test were discouraging. Adequate binding of all of the dust was not achieved. To be effective all of the floor dust must be bound. High spots were not bound nor were deposits against the ribs. Dangerous loose coal deposits existed even though the quantity of binding agent applied was far in excess of that which would be economical. The findings from this and all other previous trials showed that aqueous solutions are not effective where humidity exceeds 80 percent. In the laboratory atmosphere of 50-percent or less humidity, drying is rapid and effective. Binding agents which are not water solutions and cure by chemical reaction are not economical. The quantity of binding agent required, even with an aqueous solution, is generally too great to be economically competitive with rock dust.

Ventilation

The mine ventilating air has no direct effect on explosion propagation; however, if the ventilation system is inadequate, the chance of a methane accumulation is increased. The amount of water vapor in the mine atmosphere is too small to affect explosion development.

The velocity of a strong ventilating air current is very much less than that of the slowest explosion. For example, at a ventilation velocity of 800 ft/min the air velocity is 13½ ft/sec or less than one-tenth of the velocity of the slowest explosion that propagates flame. Moreover, the static pressure developed by the slow explosion is at least 10 times greater than the highest ventilation pressures used. In actual explosion tests made with ventilating air velocities ranging from 0 to 850 ft/min, no significant effect of the direction of the ventilating air was observed on explosion development.

The idea that coal dust explosions "always go against the air" arises from the fact that in cold weather the intake air tends to dry the dust, whereas the return air is usually saturated and the dust is damp and less dispersible.

Non-uniform Dust Deposits

Limited trials were made in the Experimental Mine to determine the effect on explosion propagation of alternate zones deficient in incombustible. In these trials the average incombustible in the test zone was sufficient to arrest flame propagation. Alternate zones beginning 78 feet from the face contained 9 percent less than and 9 percent more than the limiting incombustible content. The coal dust explosions were initiated by C-25 in the single entry. The first dust zone, outby the gas igniting source, always contained the required proportion of incombustible to arrest propagation.

When the alternate zones were 60 and 90 feet in length, flame propagation was arrested. When the alternate zones were 120 feet in length explosion propagation was obtained. These data indicate that about 100-foot length of entry, 9-percent deficient in incombustible has a significant effect on explosion propagation even though the rock dust deficient zones are compensated by excess incombustible in adjacent zones. It can be presumed that if the deficiency in incombustible were greater than 9 percent--for example, if no rock dust were applied in the deficient zone--a smaller length of zone than 100 feet would affect explosion development.

Face Area

The documentation of the rationale for not requiring that rock dust be applied closer than 40 feet from the face is poor. The Bureau of Mines in May 1924 (37) presented tentative specifications for rock-dusting. Subsequently, a sectional committee of the American Engineering Standards Committee was appointed to formulate recommendations for standard practice in rock-dusting. The report of this committee was approved on Dec. 30, 1925. In March 1927, the recommendations published (38) by the Bureau specified rock-dusting to within 40 feet of the face. This was followed in June 1927 by a statement of Bureau policy on rock-dusting (40). This statement recommended rock-dusting to within 50 feet of the face. However, Mine Safety Board Decision No. 5, approved in April 1927, presented a formal Bureau policy and required compliance with the 40-foot distance. Decision No. 5 was superseded by Mine Safety Board Decision No. 32 in March 1940 (27), which restated the requirement of rock-dusting to within 40 feet of the face. The Recommended American Practice for Rock Dusting Coal Mines was also published in 1927 (39) with the 40-foot requirement. The reasoning and justification for the 40-foot rock-dust-free zone adjacent to the face is not given in any of these publications.

From a technical point of view, maximum safety would be achieved if rock-dusting were maintained to the face. This, of course, is not always practical, even considering the fact that the 40-foot distance was decided upon before the advent of continuous mining machines which range up to 30 feet in length and often are used with a loading machine 25 to 30 feet in length.

The following possible reasons are offered as the rationale for the 40-foot rock-dust-free zone at the face:

1. Generally a run-up distance of about 40 feet is required before an explosion develops appreciable air movement. However, this is not true in the case of a well-mixed methane-air mixture.

2. Coal dust on mine surfaces adjacent to the face does not contribute materially to the intensity of a methane explosion at the face.

3. Prior to 1936 the ignition zones in the Experimental Mine were essentially 50 feet in length. Using these sources of ignition the rock dust requirements outby the ignition zone were determined in a set of double entries. Subsequent to 1936 most explosion experiments were made in a single entry using shorter ignition zones. These latter tests showed that mine dust closer than 50 feet to the face aided in explosion development.

4. A 40-foot zone represents a reasonable, practical distance that could be allowed for mining activities.

5. Research at the time of decision and subsequent to the decision showed that a rock-dust-free zone longer than 40 feet at the face materially intensifies a dust explosion and requires significantly higher incombustible content outby to arrest flame development.

Housekeeping and Training

Good housekeeping at the working face and at all other locations in the mine where a potential ignition may occur cannot be emphasized too much as a primary safeguard against explosion. Good housekeeping means minimizing coal and coal dust accumulations, elimination of extraneous materials, orderly stacking of supplies and proper care of machines and equipment. Most explosions result from direct or indirect errors by management and workers; generally, more than one error is made at a given time and location. Training and appreciation of the causes and hazards by all parties are fundamental.

Mine operators sometimes question why the authorities emphasize the danger of coal dust accumulations on mine floors and ribs, while seeming to overlook the hazard of the same amount of coal dust on a conveyor belt or in a mine car. Both types of accumulation could contribute to explosion propagation. However, the accumulation of coal on the conveyor belt or mine car is a necessary result of the mining operation, and under normal conditions will be transported out of the mine. Coal dust on the mine floor and ribs cannot be justified by the mining process.

DUST SAMPLING

Visual Estimation of Incombustible Content

Research and practical experience have shown that inert material intimately mixed in sufficient proportion with the coal dust will quench explosion flame. Rock dust and water are the two inerts generally used. Although limestone dust is the most common of the solid inerts, many dark or off-color stone dusts are used. Most limestone dusts consist of calcium carbonate but some contain magnesium carbonate or other inorganic metallic compounds. Shale dust, an effective inert, is nearly black. The natural inerts which are present in all coal seams are dark grey or black in color; these natural inerts are clay, shales, stone strata below or above the coal, and impurities in the coal seam. Most of these inerts are more friable than coal and readily degrade into dust. Hence, the dust deposited during mining has a higher incombustible content than that of the coal seam. In addition to these natural inerts, the dust in many mines is contaminated with sand, particularly where track haulage is used, and with soluble and insoluble materials from the strata. The most common staining minerals are iron oxide and iron sulfate, which are dark red in color. Other contaminants are the lubricating oil, hydraulic oil, greases and soots, particularly where diesel engines are used. These impart a blackish color to dust. Other factors affecting the color of dusts are moisture content, particle size, smoothness of surface, lighting, stratification of dusts, background, direction of light source and spectral distribution of light source. The apparent color of dust as reported by an observer is obviously related to color blindness, astigmatism, and other eye defects as well as experience.

Despite all of these difficulties, visual observation for inert proportion must be relied upon at this time. Studies (17, 28) have been made to evaluate the accuracy of visual estimation for incombustibility. The findings are as follows:

1. The observer (an experienced mine inspector) judged correctly, by visual estimation, that mine dusts grossly deficient in incombustible contained less than the required incombustible. In a set of 211 mine dust samples, 17 contained 15 to 50 percent incombustible. All of these were judged by the inspector as being deficient in incombustible.

2. An observer cannot estimate with precision the exact percentage of incombustible in a mine dust sample. The difference between visual estimation and chemical analysis ranged from -17.5 to +27.5 percentage points of incombustible for rib and roof samples (102 samples). The difference between visual estimation and chemical analysis ranged from -22.5 to +22.5 percentage points of incombustible for floor samples (107 samples).

3. The use of a prepared mixture in a glass container as a reference standard of known incombustible content did not materially improve the ability to estimate the exact percentage of incombustible of a mine dust. However, the reference standard facilitates judgement of a "lighter" or "darker" color. Using a prepared standard mixture (74.5 percent incombustible) as a guide, the observer correctly judged all mine samples having less than 65 percent incombustible to be darker in color than the standard sample. In other words, no dust sample deficient in incombustible was judged to be safe. The use of a synthetic coal and rock-dust mixture for a reference standard is not as effective as using mine dust collected from the same mine for which the incombustible was determined by chemical analysis.

4. Visual observation is a poor method for estimating the moisture content of mine dust.

Sampling to a 1-inch Depth on Floor

Documentation of reasons for the decision to sample floor dust deposits to a 1-inch depth is poor. The earliest sampling instruction having this requirement is given in 1922 (36); it was restated in 1940 (34).

In 1954 an intensive study of mine dust sampling was made in an Illinois mine (17), which provides for the 1-inch sampling, concluding,

> The incombustible in the top 1-inch layer of dust on roadways gives a good estimate of the incombustible in the full-depth dust deposit.

The 1-inch-deep sample will provide a good estimate of the full-depth layer in mines where proper rock-dusting practices are in effect. Obviously, if a mine has never been rock-dusted previously, the composition of the top 1-inch layer could differ from the bottom dust.

Sampling to a depth greater than 1 inch is not necessary even if the 1-inch sample is not representative of the full-depth deposit. In most coal mines the floor is relatively flat and dust is seldom eroded deeper than 1 inch during an explosion. In fact if the dust on the floor were coal, and if a 1-inch-thick layer were dispersed, the airborne dust concentration would be about 9 oz/cu ft. This amount of coal dust would be sufficient to quench an explosion without any limestone dust present. Sampling to a depth less than 1 inch would run the risk of obtaining nonrepresentative samples.

Band Sampling

Band sampling, or the combining of the mine dust into a single sample from collection from the floor, ribs, and roof (perimeter) was adopted in 1952 by the Bureau of Mines. Band sampling reduces the time required for collection, quartering, packing, handling, and chemical analysis, thus promoting the possibility of sampling in more locations in mines. In most mines the quantity of dust on the floor is many times greater than that on the ribs and roof. Consequently, band samples tend to represent the dust on the floor. Thus, band sampling should only be used where it is obvious from visual examination that the rib-roof surfaces are adequately rock-dusted. Dust on all mine surfaces--namely, the ribs, roof, and floor--should be neutralized by rock dust. Where an obvious deficiency in rock dust exists on one of these surfaces, separate samples should be taken.

Dust Sampling Studies

Dust sampling studies (17, 24, 26, 34, 47-48) are being made to develop new and simplified sampling techniques, to evaluate the significance of collected samples and to determine proper inerting practices in mines.

Despite concerted efforts, a simplified sampling technique for collecting dust has not been developed. The procedures and techniques described in 1940 (34) are still the most acceptable methods. Vacuum cleaners have been tried, but these do not collect all of the dust from the mine surfaces, especially if the dust is damp; they permit loss of dust from hang-ups within the system; and they require power for operation. Gamma ray (2, 47) backscatter instruments for direct evaluation of the incombustible content of mine dust have yet to be proved to be sufficiently accurate; moisture and low-molecular-weight impurities in the dust cause difficulties. Other studies show that a 1-inch-wide scoop can be used to collect a floor sample which does not appreciably differ in combustible from that when a 6-inch scoop is used. Use of the 1-inch scoop would minimize quartering of the dust sample. A method for sampling float dust deposits is also being developed. Some of the findings from recent studies of dust sampling in operating mines are as follows:

1. Grab spot samples are not representative of the 6-inch strip samples on the floor.

2. The incombustible content of dust in adjacent 3-foot squares differs significantly.

3. The incombustible content of the dust in a 3-foot square differs significantly from that of the adjacent 6-inch wide strip.

4. The incombustible content of rib-roof dust cannot be correlated with the corresponding incombustible in the floor dust.

5. The average incombustible of samples collected at 200- and 500-foot intervals differed from the incombustible of samples collected at 100-foot intervals, but under most circumstances sampling at the longer distance would give acceptable results.

6. The incombustible content of mine dust at intersections tends to be less than that of mine dust between solid ribs.

7. The incombustible in the top 1-inch layer of dust on roadways gives a good estimate of the incombustible in the full-depth deposit.

8. The weight of dust deposited on the floor exceeds by many times the dust on rib-roof surfaces.

9. The quantity of coal dust and the incombustible contents in dust samples exhibit log-normal distributions.

10. Proper dust sampling in mines requires care and the use of uniform techniques. Those methods and procedures described by Owings (34) should be followed to obtain satisfactory results.

METHANE EXPLOSIONS

Energy for Ignition

A methane-air mixture is readily ignited by a weak electrical spark, a frictional spark, a heated surface, or an open flame (29). The minimum electrical energy of a spark causing ignition varies with gas concentration, humidity, oxygen content of the atmosphere, temperature, and turbulence. As little as 0.3 millijoule of electrical energy is required; this is equivalent to 1/120000000 of energy used in 1 second by a 50-horsepower motor or about one-fiftieth of the static electricity accumulated by an average-sized man walking on a carpeted floor on a dry day.

The minimum ignition temperature of methane is about 1,000° F (537° C). Visually, this temperature is equivalent to that obtained when an object is heated to a dull red in a darkened room (just above the threshold of visibility). Resistors in controllers or other electrical components may exceed this temperature. Methane-air mixtures may be ignited by frictional sparks (19) generated by machine bits cutting sandstone or pyrite; however, not all visible frictional sparks are sufficiently incendive to ignite gas. When an electrical spark flashes in a methane-air mixture, a discrete time period is required for flame propagation and measurable pressure to develop. In large-scale tests in the Experimental Mine, this time decreased from 2 seconds at the lower explosive concentration to one-third of a second at the optimum concentration. Above the optimum concentration the time for pressure development can be more than 8 seconds as the upper limit is approached; in one test with 14 percent methane a delay of 1½ minutes was observed. The time delay and the pressure development for 6.0-, 9.5-, and 13.5-percent gas-air mixtures are shown in figure 14.

The location of the point of ignition within a gas-air body at a face markedly affects the severity of the explosion which develops (29). Explosion severity (pressure and flame velocity) is greatest when ignition occurs at the face; severity decreases as the point of ignition is moved from the face toward the outby end. When the gas is ignited at the outby end, rapid burning rather than explosion develops; this burning does not create sufficient disturbance to disperse enough coal dust from mine surface to propagate an explosion. The static explosion pressure as measured in the Experimental Mine tests was less than 1 psig when a 40-foot gas body was ignited at the outby end, 27 psig when ignited at the center, and 39 psig when ignited at the face.

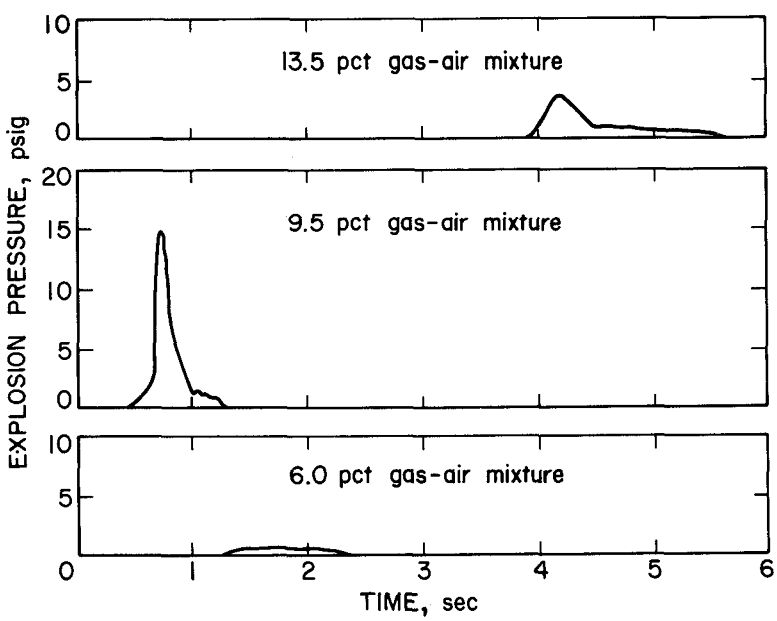

FIGURE 14. - Pressure-time records 50 feet from face for explosions of 6.0-, 9.5-, 13.5-percent natural gas-air mixture in a 25-foot-long zone.

The range of flammability of methane-air mixtures varies with the strength of igniting sources, turbulence, direction of flame propagation, size of apparatus and enclosure. Generally, laboratory experiments are made with open-end, vertical glass tubes, 4 inches or greater in diameter and with ignition at the base. The lower limit of flammability is about 5 percent methane in air and the upper limit is about 15 percent. With a closed tube or with ignition at the top, different values for the lower and upward limits are obtained. In large-scale tests in the mine, the limits for ignition are about the same as for the open-end vertical glass tube with ignition at the base.

Pressures and Rates of Pressure Rise

The static pressure from explosion of methane-air mixtures varies with methane concentration, uniformity of mixture, degree of confinement, location and intensity of igniting source. In a closed vessel the volume has little effect on the maximum pressure, but in an open mine entry the magnitude of pressure increases with increase in volume of the gas body.

Data on explosion pressure and rate of pressure rise in a 1-cubic-foot closed cubical vessel with central ignition are shown in figure 15.

The rate of pressure rise from a methane explosion in a closed vessel depends on the volume of the vessel (33). A longer period of time is required for flame to travel to the wall of a large vessel than to the wall of a small vessel. For equivalent conditions the maximum rate of pressure rise decreases inversely as the cube root of the vessel volume. In figure 14 the highest rate of pressure rise for the 1-cubic-foot vessel is 2,200 psi/sec. In a 1,000-cubic-foot vessel the maximum rate of pressure rise for the same gas would be the ratio of the cube roots of the respective volumes or 220 psi/sec. This cube root relation is true for spherical or nearly spherical vessels. In nonspherical vessels a more precise relation is

$$\left(\frac{dp}{dt}\right)_{max} = \frac{kS}{V}$$

where k is a constant, S the surface area of the vessel and V the vessel volume. For spherical vessels the above equation becomes

$$\left(\frac{dp}{dt}\right)_{max} = \frac{3k}{V^{1/3}}$$

which is equivalent to the cube root relation.

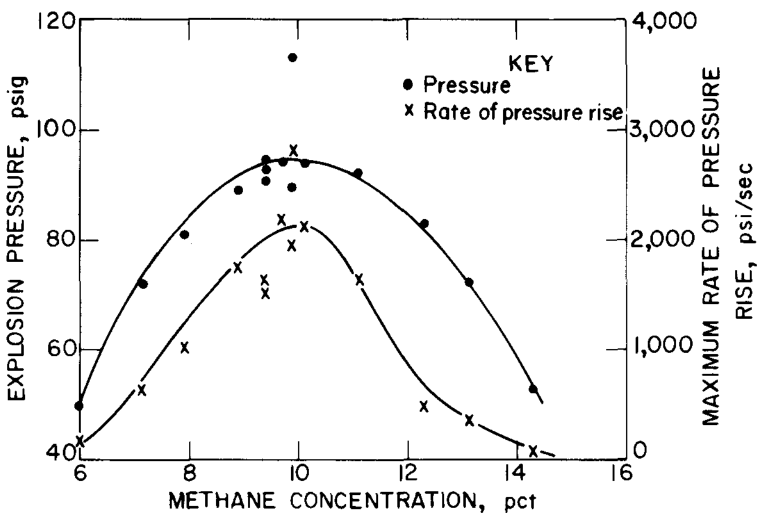

FIGURE 15. - Pressure and rate of pressure rise developed by explosion of nonturbulent methane-air mixtures in a 1-cubic-foot vessel.

Turbulence

Turbulence induced by a fan or the atmosphere moving past an obstacle has only a slight effect on maximum static pressure developed by the explosion. Data (33) show turbulence caused about a 10-percent increase in the static pressure in the closed vessel. Turbulence has a major effect on the magnitude of the rate of pressure rise and may cause a six-fold increase. Turbulence within the vessel induces a wrinkled flame front which enhances flame spread. The rate of pressure or flame development for a turbulent coal dust dispersion is only slightly less than that for a quiescent methane-air mixture.

In a mine a gas-air body can be nonturbulent, for example, if the ventilation current stops and methane accumulates. Alternatively, the mixture can be turbulent, as would be caused by movement of the miners and machines in a face or other area where methane is being liberated. The turbulence of the mine ventilating air tends to promote the mixing of methane with the air, but often the degree of turbulence of the moving air current adjacent to the mine strata, where methane is released, is too low to induce a high degree of mixing. Hence methane layers can form.

Methane Concentration

The data presented graphically in figure 14 show the effect of methane concentration on explosion pressure and rate of pressure development in a closed vessel. At 6- and 14-percent methane concentrations the explosion pressure is about half of that developed for the 9.5-percent mixture. The methane concentration affects the rate of pressure rise similarly. In large-scale tests in the Experimental Mine, the ignition of a 6- or 14-percent methane-air body causes sufficient disturbance to disperse and ignite coal dust outby in the mine entry.

The maximum static pressure produced in a mine entry from ignition of a methane-air body varies similarly with methane concentration as for the closed-vessel experiments; however, the magnitude of the pressure is significantly less. As one would expect, the methane concentration has a similar effect on the velocity of the flame in the entry. Detailed data on these parameters have been published (25).

Degree of Confinement

The maximum static pressure developed by a methane-air explosion in a closed vessel is rarely attained in the mine environment because of the pressure release afforded by the open passageway. However, if the methane accumulation is extensive, pressure piling or detonation can occur and higher pressures are attained.

In 1952, Schultze-Rhonhof (50) made two experiments using a 9.5-percent mixture in a 984-foot-long zone in an abandoned mine. The peak pressure attained was 145 psig and the flame velocity was nearly 3,000 ft/sec. The sustained static pressure was of the order of 40 psig.

Length of Accumulation

The volume of the gas-air body in a mine passageway affects the maximum static explosion pressure that develops. In Experimental Mine tests made with 260-, 1,300- and 2,060-cubic-foot volumes, the explosion pressures from ignition of these 9.5-percent mixtures were 9, 33 and 53 psig, respectively, as measured at the face of the entry. In the tests (25), the maximum flame extension following ignition was approximately 5 times greater than the length of the original methane-air mixture. Maximum flame extension occurred at a 12-percent methane-air mixture. At lesser or higher concentrations the flame extension was smaller.

Location of Accumulation

Highest static explosion pressure from ignition of a methane-air body is obtained when the mixture is at the face of the entry (29). The maximum pressure decreases as the distance of the gas body from the face area increases. In the Experimental Mine tests with the 40-foot-long zone 0, 200 and 500 feet from the face, the maximum explosion pressures were 39, 23 and 13 psig, respectively. In all instances the highest pressure occurred at the face rather than at the location of the gas body. In fact the pressure at the face was two to four times higher than that recorded 500 feet outby the face.

For a gas body 200 feet from the face, the point of ignition within the gas body has only a moderate effect on explosion pressure. The values for ignition at the inby end, center and outby end are 23, 19 and 13 psig, respectively. For the gas zone 500 feet from the face a pressure of 13 psig was obtained with ignition at the center. The height of the igniter above the floor of the entry does not materially affect the explosion development.

Layering

Tests (25) were made in the Experimental Mine with four gas-air patterns: (a) homogeneous mixture, (b) methane outlet 5 inches above the floor, (c) methane outlet at 35 inches or midheight, and (d) methane outlet at 65 inches from the floor or 6 inches below the roof. The same quantity of methane was introduced into the 25-foot-long zone for each pattern. Different types of methane layers were formed at the face in patterns b, c and d, and the thickness of the explosive mixture decreased as the methane outlet was raised above the floor.

The most violent explosion developed with pattern (a), the homogenous mixture, and the least violent with pattern (d), when the methane was liberated just below the roof. Data from these experiments are given in table 12.

These data show that the violence of the explosion is directly related to the degree of mixing of the methane and air. For all patterns the disturbance caused by ignition was sufficient to initiate a coal dust explosion in mine dust located outby the gas zone.

TABLE 12. - Layered methane tests in a 25-foot-long zone

Pattern	Height of gas inlet, inches	Height of[1] flammable mixture, inches	Portion of zone containing flammable mixture, percent	Maximum explosion pressure, psig	Maximum flame velocity, ft/sec
a[2]	([2])	-	100	33	870
b	4	6	85	17	530
c	35	28	25	6	420
d	65	43	10	4	190

[1]Lower level of flammable mixture from floor.
[2]Gas mixed by air circulation.

Tests (6) were made in the Polish Experimental Mine to determine minimum thickness of a methane layer at the roof which could initiate a coal dust explosion under optimum conditions. They found that with a 125-foot-long layer, ignition of coal dust was readily obtained with a methane layer 2 inches thick at the roof. Extrapolation of the experimental data showed that as little as ½-inch-thick layer of methane at the roof could be sufficient to initiate a coal dust explosion.

Minimum Quantity of Methane and Coal Dust To Initiate a Coal Dust Explosion

The minimum quantity of methane required to initiate a coal dust explosion was studied in 1930 (43). Tests made in the Experimental Mine closely simulated conditions existing in an operating mine. The data show that the minimum quantity of methane confined at the face which, when ignited, would disperse and ignite coal dust was 13 cubic feet. The 13 cubic feet of methane was mixed with air to form a total volume of 140 cubic feet of mixture. It can be envisioned that under some circumstances a smaller quantity of methane could cause an explosion if, for example, a methane-air mixture were confined in a drill hole and coal dust laid on a machine outby along the axis of the drill hole.

No similar direct experiments were made to determine the minimum quantity of coal dust which, when dispersed and ignited, would develop a sustained coal dust explosion. Indirect experiments were made with the electric-arc ignition source in single entry trials. Twenty-five pounds of coal dust are used with that ignition source. With a refined dust-dispensing arrangement, similar flame development was obtained when the coal dust quantity was reduced to 5 pounds; further reduction was not tried. By calculation it can be shown that the minimum quantity of coal dust required to initiate an explosion is approximately 5 pounds. The heat liberated in the combustion of 13 cubic feet of methane is approximately 12,000 Btu. One pound of coal, when burned completely, liberates 12,000 to 14,000 Btu, the same as 13 cubic feet of methane. However, as will be discussed later, only about one-fifth of the coal dust enters into an explosion reaction. Hence, about 5 pounds of coal dust is required to produce heat release equivalent to that from 13 cubic feet of methane. The rate of pressure or flame development for the turbulent coal dust dispersion is only slightly less than that for a quiescent methane-air mixture. It must be recognized that under special circumstances, just as for the methane-air mixtures discussed above, a lesser amount of coal dust could initiate a mine explosion.

One cannot conclude that 5 pounds of coal dust or 13 cubic feet of methane represents an imminent danger to miners. Considerable quantities of unneutralized coal dust are present in face areas, on belts and in mine cars; in many mines gob areas hold many cubic feet of methane. Although a potential for an explosion may exist, other factors must be considered, such as the length of passageway having unneutralized coal dust, the locations of the unneutralized coal dust or methane relative to active portions of the mine, potential igniting sources, ventilation, and potential for producing a dust cloud, as well as the practices of miners and management.

FUNDAMENTALS OF EXPLOSION PHENOMENA

Mine explosions have occurred with dusts of bituminous coal, lignite, gilsonite, and iron sulfide; gaseous fuels were methane, hydrogen, and acetylene. Vapors from the liquids of hydraulic oil, diesel oil, and gasoline have exploded. Although records of past explosions in mines have not been found for sulfur dust, ammonia, hydrogen sulfide or carbon monoxide, these fuels also explode. Steam and compressed air explosions have occurred in mines and at surface facilities of mines. The most common fuels for explosions are methane gas and bituminous coal dust.

The combustion reaction of the fuel with oxygen releases heat energy which is transferred to the oxygen-depleted air and to the gases of combustion. In most combustion reactions during a mine explosion, the volume of the gaseous combustion products is about the same or slightly greater than that of the oxygen reacting; hence, the expansion is primarily due to the increase in temperature. Within seconds after the explosion flame dies out, the gases cool as the heat energy is transferred to mine surfaces and other objects, and a partial vacuum develops. This causes movement of the gases in the reverse direction. Depending on the size, shape and type of explosion reaction, the pulsating gas movement can exceed more than 10 cycles.

Temperature

The maximum temperature in a combustion explosion depends on the type and quantity of fuel, its concentration, inert material present, the amount of oxygen available, pressure relief, turbulence, and particle size if the fuel is a dust. Explosion temperature may be measured directly in experiments, but information on maximum temperature can be calculated easily from the explosion pressure, assuming no change in volume of the gases.

The pressure and temperature are approximated by

$$P_1/T_1 = P_2/T_2 . \qquad (1)$$

The maximum pressure (33) from an explosion of Pittsburgh coal or methane in a closed vessel is about 100 psig or 115 psia. Hence the explosion temperature is calculated as:

$$\frac{15\ \text{psia}}{300°\ \text{A}} = \frac{115\ \text{psia}}{T_2} \qquad (2)$$

or T_2 = 2,300° A = 2,000° C = 3,600° F. Using this calculated temperature, the expansion ratio of the explosion gases can be approximated from:

$$\frac{V_1}{T_1} = \frac{V_2}{T_2} \quad (3)$$

or

$$\frac{V_1}{300° \text{ A}} = \frac{V_2}{2{,}300° \text{ A}} \quad (4)$$

and

$$V_2 = \frac{2{,}300}{300} V_1 = 7.7\ V_1 . \quad (5)$$

Thus the maximum volume expansion is about 7.7 times. In actual explosions the expansion is about 5 times rather than 7.7 because optimum explosion conditions are rarely obtained. If a methane-air mixture occupied 10 linear feet of entry and were ignited, the flame would fill approximately 50 linear feet of passageway.

Pressure

A gas or dust explosion in a mine passageway develops two types of pressures--"static" and "dynamic." The heated gases expand and exert a force equally in all directions. This force, termed "static" pressure, is the pressure measured in the laboratory bomb. In an open mine entry the heated gases expand and flow through the passageway, pushing air ahead. This flow of gas at high velocity produces a wind or "dynamic" pressure. The dynamic pressure is directional like the wind forces in a storm which one can avoid by retreating around the corner of a building or going behind a tree.

Both static and dynamic pressures cause damage during a mine explosion. The static pressure destroys stoppings in crosscuts, and the dynamic pressure moves objects such as a parked mine car or a rectifier in the entry downwind of the origin. The maximum static pressure for a coal dust or methane explosion in the closed laboratory bomb is about 100 psig. In an open mine entry the average maximum static explosion pressure is generally less than 100 psig because of pressure release offered by air flow in the passageway. However an instantaneous maximum pressure during a mine explosion may exceed the experimentally determined maximum pressure of 100 psig under two circumstances--detonation and pressure piling.

In a detonation, shock waves may develop at the flame front. These shock waves advance ahead of the flame and reinforce each other in the unburned fuel-air mixture. When the energy in these shock waves is sufficient, self-ignition of the mixture occurs and new flame fronts develop; the instantaneous static pressure from the detonation may be several times higher than 100 psig. The duration of the detonation peak pressure is very short; nevertheless, these shock pressures may cause damage. Following decay of the detonation shock wave, the sustained static pressure from the combustion reaction prevails.

In the second circumstance, "pressure piling," the fuel-air mixture ahead of the flame front is compressed--as may occur in a restricted or dead-end entry where pressure equalization is hindered. The explosion pressure in a precompressed mixture is proportional to the absolute pressure. Thus if the precompression is 3 atmospheres, the instantaneous explosion pressure could be as high as 300 psig. Pressure piling can only occur when the physical configuration inhibits flow of gases for pressure equalization. Cybulski (6, p. 284) discusses a coal dust explosion experiment in a dead-end entry in which the peak static pressure was at least 595 psig. This explosion did considerable damage. Dust explosion pressures in open entries exceeding 150 psig have been developed during explosion tests in the Experimental Mine. The calculated theoretical value of maximum static pressure for coal dust (33) or methane explosion in a closed vessel is higher than the observed experimental 100 psig value and is about 140 psig. In an experimental explosion complete combustion does not occur and heat is lost to the enclosure, which accounts for the lower pressure.

When an explosion occurs in a mine passageway, the heated gases expand in all possible directions. The expanding gases are hindered by the frictional resistance at the mine surfaces, by obstacles, and by the inertia of the air in front of the expanding gases. The speed of the expanding gases in the entry is always less than the velocity of sound. A shock wave, such as that which would develop from the firing of a charge of explosives or from ignition of a methane-air body near optimum concentration, moves through the entry at sonic velocity.

The force generated by the dynamic or wind pressure is calculated from the following formula which is valid up to about 0.4 the speed of sound:

$$F = \tfrac{1}{2}\,\frac{A\rho v^2}{g} \qquad (6)$$

where F = total force, pounds,

A = area of the object, square feet,

ρ = density of the moving air, lb/cu ft,

v = air velocity, ft/sec,

and g = acceleration of gravity, ft/sec^2.

Thus a mine car having an exposed area of 20 square feet when subjected to an air velocity of 300 ft/sec, would be pushed by the force

$$F = \tfrac{1}{2}\,\frac{20 \times .075 \times 300^2}{32} = 2{,}110 \text{ lb.} \qquad (7)$$

The impulse developed by the moving gases is defined as the force multiplied by the time of its duration. Assuming the explosion gases move at a constant speed of 300 ft/sec for 2 seconds the impulse on the mine car of 20 square feet area would be 4,220 pound-seconds. Such an impulse could propel the car for an appreciable distance. Normally in an open entry the explosion flame would overtake the car and the subsequent air flow in the opposite direction would stop or even reverse its movement.

The rate of pressure rise during an explosion is an important parameter for a closed bomb test in the laboratory, as it identifies the rate of chemical reaction. In the mine environment the rate of pressure rise does not have this significance, as the rate of pressure rise is a function of the explosion volume and the pressure release in the open passageway. Both of these factors are generally impossible to determine in a mine.

The static pressure in a mine explosion can be as little as a fraction of a pound per square inch or as much as 600 psig if detonation occurs. The static pressure for a weak explosion will be less than 5 psig. A moderate explosion develops a static pressure up to about 15 psig and a strong explosion up to about 40 psig. In a violent explosion the static pressures will exceed 40 psig. The apparent violence on the surface from an underground explosion does not always indicate the severity underground because of the attenuation of the pressure with distance. A strong or violent explosion more than 4 miles from the portal may produce little disturbance on the surface.

The magnitude of the static pressure developed in a mine by an explosion will increase as the volume of the fuel zone increases. The static pressure increases as the fuel concentration increases up to an optimum concentration and then decreases. For dust explosions, the static pressure increases as the particle size of the dust decreases because the rate of reaction increases; turbulence in the fuel-air mixture also tends to increase the pressure. Static pressure is affected by mine geometry particularly as it affects pressure relief either ahead of or behind the flame front. A fuel-air body ignited in an open passageway will develop less static pressure than a similar fuel-air body located at the face in a dead-end-entry.

Air and Flame Speeds

The speed and duration of the moving air in a mine explosion disperse dust from the mine surfaces and are the factors causing most damage in the underground workings. Although the static pressure may destroy stoppings in crosscuts, most destruction is caused by the dynamic pressure. The dynamic pressure increases as the square of the air speed. A hurricane on the surface causes terrible damage when the wind speed is 150 to 200 mi/hr (230 to 290 ft/sec). In most mine explosions the air speed exceeds 200 mi/hr (290 ft/sec). In fact, a coal dust explosion will generally die out if the air speed is less than 100 mi/hr (150 ft/sec).

In early research in the Experimental Mine, flame speed was measured. These data represent the speed of the flame relative to fixed locations in the mine entry. Flame speeds are relatively simple and air speeds are difficult to measure. Richmond (44) has published a curve showing the relation between air speed and flame speed (fig. 16). This curve shows that flame and

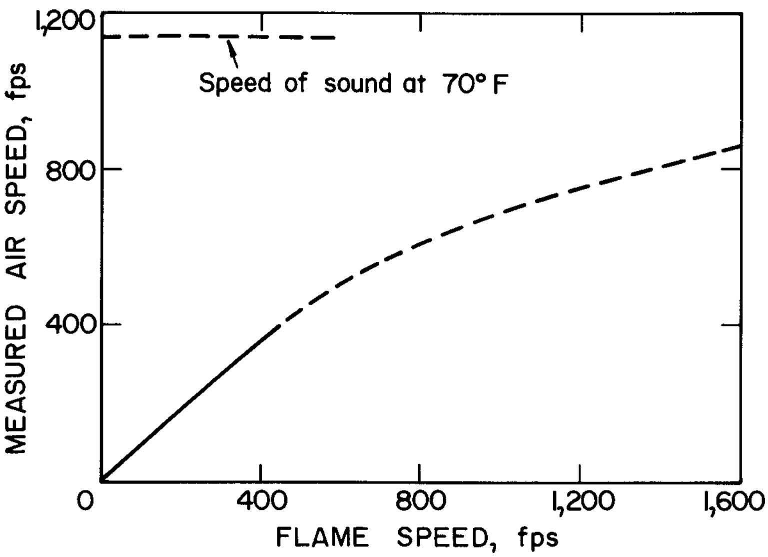

FIGURE 16. - Air speed as a function of flame speed for gas and coal dust explosions.

air speeds are the same up to about 400 ft/sec (270 mi/hr). At higher flame speeds the air speeds are less than the flame speed. It is fundamental that the air speed does not exceed the speed of sound. Flame speeds as high as 5,000 ft/sec have been measured in experimental explosions; in at least 30 tests the flame speed of explosions exceeded 3,000 ft/sec.

In the preceding reference (44) Richmond shows that the relation between static pressure and flame speed is approximately linear for the observations up to 30 psig (fig. 17). From his two curves, the relation between air speed and static pressure can be obtained. Richmond points out that for air speeds below 400 ft/sec, the acoustic approximation for incompressible one-dimensional air flow is consistent with the data. The equation is

$$P = \frac{\rho c v}{144g}, \tag{8}$$

where P = maximum static pressure, psig,

ρ = initial gas density, lb/cu ft,

c = velocity of sound, 1,120 ft/sec,

v = maximum air velocity, ft/sec,

and g = acceleration of gravity, ft/sec^2.

By substitution in this formula, an air speed of 400 ft/sec is produced by a static pressure of 7.2 psig.

Dust in a layer is dispersed by the moving air, hence the air speed required to erode and disperse dust is an important factor. Tests (51) show that in a 2- by 3-inch cross-section wind tunnel, coal dust is eroded at a minimum air speed of about 25 ft/sec and rock dust at about 50 ft/sec. Dawes, in a British paper (7) reports values of about 16 and 25 ft/sec for erosion of coal dust and rock dust respectively. The air velocity required to disperse clumps from a layer is higher than that to produce erosion. The minimum air velocity which disperses sufficient dust to result in a self-sustaining explosion is about 100 ft/sec.

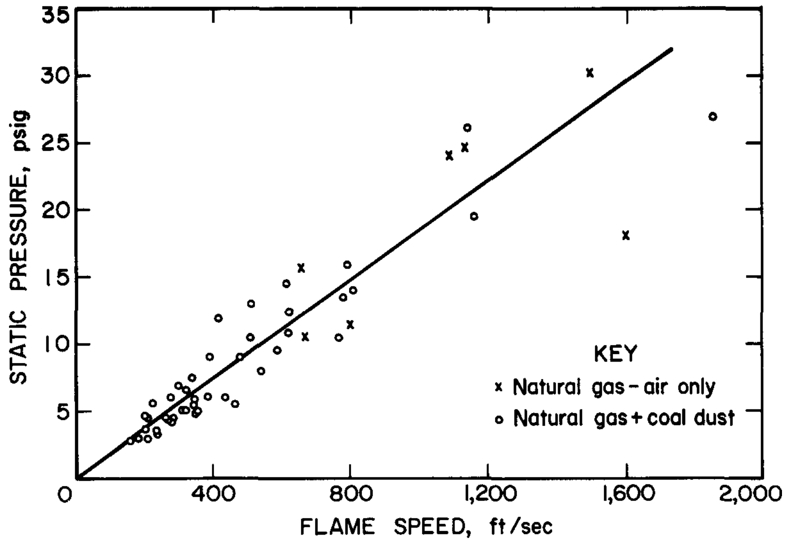

FIGURE 17. - Pressure as a function of flame speed for gas and coal dust explosions.

Comparison of Methane and Dust Explosions

Both a methane and a coal dust explosion produce the same effects on miners and mine workings. However, there are some differences which distinguish between the two fuels. The explosion flame of coal dust is reddish to orange in color; at the optimum concentration without inert dust present the flame becomes pale white. At low concentrations methane flame is a pale blue; at optimum concentration it is bluish white, and at high concentrations a yellowish white.

Except at very low dust concentrations, coal dust produces the black smoke of unburnt carbon. After the explosion, the smoke settles in a layer over the mine surfaces and equipment. Up to the optimum concentration, a methane explosion produces no smoke. Above the optimum concentration black smoke is formed from unburnt carbon. A methane explosion in the mine may raise dust into the air which appears like smoke, or it may ignite some coal dust which produces smoke.

Except at very low dust concentrations, a coal dust explosion produces coke particles either from the air-borne particles or on the mine surfaces. A methane explosion may heat and coke coal dust on mine surfaces. Normally coke, which is transported or deposited in layers, indicates a coal dust explosion.

The particle size of coal dust is a primary factor affecting rate of pressure development, as the coal dust combustion is a surface reaction affected by the rate of oxygen diffusion to the particle surface. Figure 18 shows the decrease in pressure and rate of pressure rise with increase in particle diameter for explosion of Pittsburgh coal dust in the laboratory closed vessel. Float coal dust, normally having an average particle size of about 20 microns, produces high pressure and rate of pressure rise. Pulverized coal, having an average particle size of about 40 microns, has a slightly lower pressure and rate of pressure rise. Both the pressure and the rate of pressure rise are relatively low for coarse coal. In a mine explosion where the ignition source is stronger and flame duration is longer, the decrease would not be as rapid as is shown for the laboratory tests.

The explosive range for methane is 5 to 15 percent by volume or 0.033 to 0.1 oz/cu ft. The explosive range for coal dust is about 0.05 to 5 oz/cu ft.

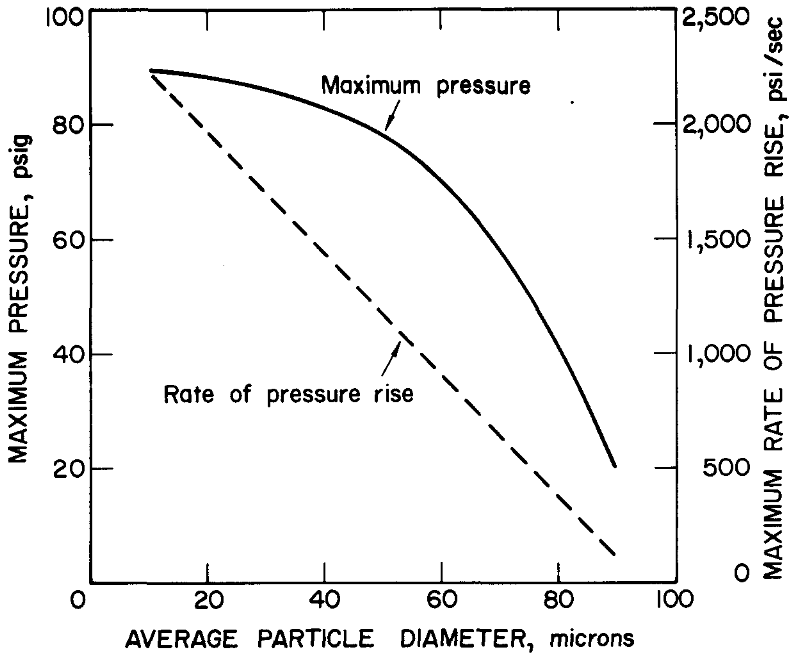

FIGURE 18. - Effect of particle diameter on static pressures and rates of pressure rise developed by Pittsburgh coal dust explosions in the Hartmann tube.

Thus on a weight basis, the explosive range of coal dust is 30 times wider than that of methane.

Composition of Explosion Gases

The composition of the atmosphere after an explosion will vary with the fuel concentration, uniformity of mixture and to a lesser extent with other variables such as humidity, turbulence and other materials in the explosion zone. Data on the composition of the atmosphere in a closed bomb explosion for methane or coal dust fuel are given in table 13.

In a methane explosion below the stoichiometric concentration, almost all of the carbon in the fuel burns to carbon dioxide. Above the stoichiometric concentration, carbon monoxide and hydrogen as well as some carbon dioxide are formed.

TABLE 13. - Composition of atmosphere in closed vessel after explosions of methane or coal dust

Explosion results[1]	Methane concentration prior to explosion, percent			Coal dust concentration prior to explosion, oz/cu ft				
	8	9	12	0.1	0.2	0.5	1.0	2.0
Explosion gases, percent:								
CO	-	0.5	8.0	0.1	0.7	2.8	4.6	4.0
CO_2	9.2	10.7	5.9	3.2	9.1	12.3	11.7	12.2
H_2	-	.3	8.5	-	-	1.0	3.0	2.3
CH_4	.03	.2	.4	[2]Tr	[2]Tr	.1	.6	1.1
O_2	3.8	.5	.5	17.0	9.6	3.1	1.5	1.5
N_2	86.0	86.8	75.8	78.8	79.6	79.8	77.5	77.8
A	1.0	1.0	.9	.9	.9	.9	.9	.9
Jones-Trickett ratio	.48	.48	.49	.82	.80	.79	.70	.71

[1]Considering water vapor has condensed to water.
[2]Tr - Trace.

If all of the carbon in the methane were to burn to carbon dioxide at the stoichiometric concentration, the reaction would be:

$$CH_4 + 2O_2 + 7.47N_2 + 0.08A \longrightarrow CO_2 + 2H_2O + 7.47N_2 + 0.08A$$

and the concentration of carbon dioxide would be 9.5 percent with water present as a vapor or 11.7 percent with the water condensed to a liquid. The concentration of nitrogen would be 70.8 percent if the water vapor were present or 87.3 percent if the water vapor were condensed.

Examination of the composition of the atmosphere after an explosion of Pittsburgh coal dust at 0.1 oz/cu ft dust concentration shows that only a fraction of the coal dust is burned, as 17.0-percent oxygen remains in the vessel. The stoichiometric concentration of the coal is 0.123 oz/cu ft. Even at a coal dust concentration of 0.2 oz/cu ft, only about half of the oxygen is consumed. As for the methane fuel, hydrogen and carbon monoxide are formed at coal dust concentrations higher than the stoichiometric. Jones and Trickett (22) made a detailed analysis of gases from methane and coal dust explosions. They developed a ratio which permits differentiation between a methane and a coal dust fire or explosion from the composition of the combustion gases. The Jones-Trickett ratio, included in table 13, is calculated from the composition of explosion gases as

$$JTR = \frac{CO_2 + \frac{3}{4}CO - \frac{1}{4}H_2}{O_2 \text{ used}}.$$

In the ideal instance the ratio is 0.5 or less for methane and 0.85 or less for coal dust combustion reactions.

Numerous gas samples were collected from the mine atmosphere during explosion tests in the Experimental Mine (36). The data are given for 260 samples. The atmosphere was sampled just before, during and after the arrival of flame. The maximum values listed for the several gases are: 18 percent carbon dioxide, 11.7 percent carbon monoxide, 4.1 percent methane and ethane, 6.5 percent hydrogen and 1.3 percent ethylene. The minimum value for oxygen was 0.1 percent. The composition of the atmosphere during an explosion depends on the time with reference to the flame front, fuel concentration, inert dust, water present, strength of igniting source, violence of explosion, and methane present prior to the explosion. Methane, ethane, hydrogen, and ethylene are formed when no methane is present as a fuel for the explosion. The distillation products--methane, ethylene and hydrogen--begin to appear in the samples while the oxygen concentration is high. It appears that these products are distilled from the coal dust by the heat of the explosion prior to the arrival of the flame. The ratio of CO to CO_2 and the relation of this ratio to the oxygen content vary greatly in the different samples. In tests with mixtures of dry dusts, the presence of large percentages of carbon monoxide, sometimes exceeding that of carbon dioxide, indicates that a large excess of coal dust was dispersed in the air current. The samples collected in tests with mixtures of dust and water indicate that the water is reduced by incandescent carbon, giving carbon monoxide and hydrogen.

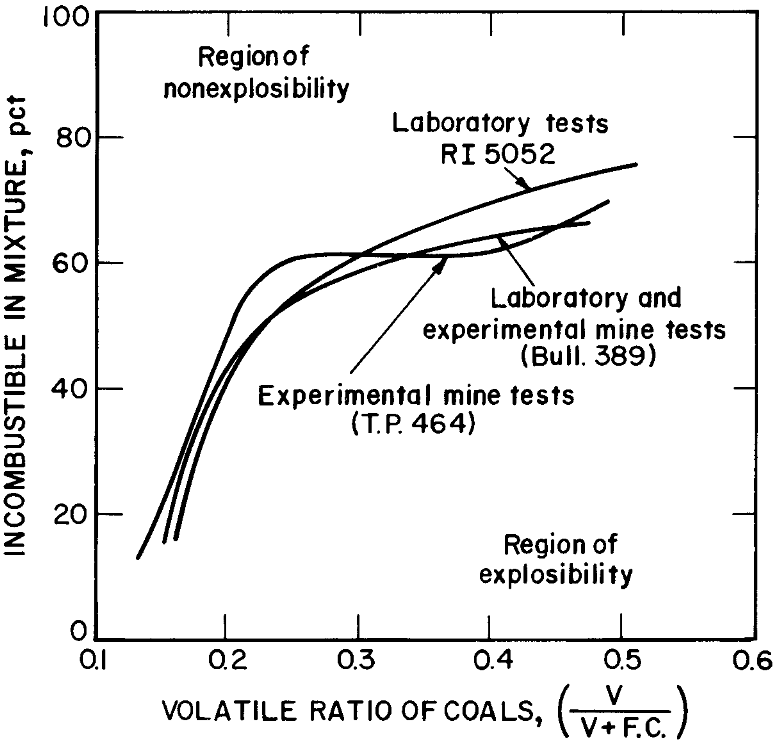

FIGURE 19. - Explosibility of mine-size coal dusts based on tests in the Experimental Mine and laboratory.

Laboratory Studies

Although most of the research data given in this paper are from large-scale tests in the Experimental Mine, the total effort spent in laboratory work far exceeds that in mine studies. Some parameters of explosion phenomena best examined in the laboratory are fundamentals of combustion, reaction rates, ignition phenomena, oxygen concentration and pressure development. The Bureau of Mines and many other organizations study and publish findings on these subjects. The work in the Experimental Mine is best for translating the laboratory studies into practical application and for resolving parameters which are affected by the scaling from the small scale to the large scale environment, such as the effects of strength of igniting source, incombustible required to arrest propagation, barriers, and position of dust in the perimeter of the entry.

Small-scale tests, calibrated in terms of the large-scale tests, can effectively save time and effort. Figure 19 shows data from the Experimental Mine and small scale furnace test on the incombustible required to arrest flame propagation of coals having different volatile contents. The laboratory data, obtained at a fraction of the cost of the mine data, offer good approximation.

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THE EXPLOSION HAZARD IN MINING

by

John Nagy

ERRATA

On table 2, page 5, the eleventh explosion listed, the mine name should read "Dutch Creek" instead of "Dutch Creek No. 2."

On page 19, under "Statements on Ignitions From Coal Mine Inspectors' Reports," the first explosion listed actually occurred Dec. 28, 1965, instead of Dec. 28, 1956. Because of the error in date, this explosion is also out of time sequence in the listing; instead of appearing first on the list, it should have appeared between No. 19 and No. 20.