Secondary Explosion Hazards During Blasting in Oil Shale and Sulfide Ore Mines

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By Eric S. Weiss, Kenneth L. Cashdollar, Michael J. Sapko, and Eugene M. Bazala

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

bar/s	bar per second		L/t	liter per ton (metric)
cal/g	calorie per gram		m	meter
cm	centimeter		m/s	meter per second
fr/s	frame per second		m^2	square meter
ft	foot		m^3	cubic meter
g	gram		m^3/min	cubic meter per minute
g/cm^3	gram per cubic centimeter		m^3/t	cubic meter per ton (metric)
g/m^3	gram per cubic meter		mbar	millibar
gal/st	gallon per short ton		min	minute
gr/ft	grain per foot		mm	millimeter
h	hour		mol/kg	mole per kilogram
J	joule		ms	millisecond
K	kelvin		pct	percent
kbar	kilobar		s	second
kg	kilogram		t	ton (metric)
kg/m^3	kilogram per cubic meter		t/d	ton (metric) per day
kJ	kilojoule		vol pct	volume percent
km	kilometer		wt pct	weight percent
km^2	square kilometer		μm	micrometer
L	liter		°C	degree Celsius

SECONDARY EXPLOSION HAZARDS DURING BLASTING
IN OIL SHALE AND SULFIDE ORE MINES

By Eric S. Weiss,[1] Kenneth L. Cashdollar,[2] Michael J. Sapko,[3] and Eugene M. Bazala[4]

ABSTRACT

The data presented in this report are the results, to date, of an ongoing Pittsburgh Research Center[5] Disaster Prevention research program on the explosion hazards associated with blasting operations in noncoal mines. Laboratory and experimental mine tests have shown that oil shale and sulfide ore dusts can be ignited given the proper predispersed dust concentrations, particle size, and kerogen or sulfur content. Methane (CH_4) gas may also be present in deep oil shale formations and can pose a significant added hazard to underground blasting operations. The most common explosive used for blasting in oil shale had been ANFO, a combination of ammonium nitrate and fuel oil; blasting in sulfide ore had used ANFO and/or dynamites.

Tests conducted at the cannon gallery at Lake Lynn Laboratory near Fairchance, Fayette County, PA, provided a means of evaluating the relative incendivity characteristics of new and existing explosive products. Three explosives—a pumpable emulsion-ANFO blend, a packaged water gel, and an emulsion blasting agent—exhibited low-incendive qualities compared with other more highly incendive products such as ANFO and some dynamites. Based on the data collected during numerous full-scale blasts in oil shale and sulfide ore mines, the low-incendive products significantly reduced or eliminated the ignition hazards while at the same time providing effective fragmentation of the rock.

Based on the positive results from the gallery and field testing, low-incendive explosives, coupled with good blasting procedures, show promise in reducing dust and/or gas ignitions associated with blasting operations in oil shale and sulfide ore mining applications.

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[5]This work originated under the U.S. Bureau of Mines prior to transferring to the U.S. Department of Energy on April 4, 1996.

INTRODUCTION

OIL SHALE MINING

Alternative fuel sources have been viewed over the past several decades as possible replacements for, or additions to, the crude U.S. petroleum supply. The development of alternative fuels could also benefit our national security by reducing the reliance on foreign oil. This lack of energy self-reliance was demonstrated quite effectively during the 1973 oil embargo by the oil exporting nations. During this period, a resurgence in the U.S. oil shale industry occurred. The embargo catalyzed the United States to seriously consider developing an industry to derive oil from shale rock.

Oil shale must be heated to convert the solid hydrocarbon (kerogen) in the rock to crude shale oil. The retorting technology can be accomplished either in situ or at a surface facility. The in situ retorting process (1-2)[6] is conducted within the shale formation. For aboveground retorting (3) operations, the shale rock is removed from the ground and transported to the surface facility. Both methods require extensive underground facilities which are generally developed through blasting operations. Many potential hazards are encountered when conducting large-scale underground mining and blasting activities, not the least of which are the explosives used for these blasting operations. Due to the high volume of material that is required to be extracted in these large mine entries, blasting agents are generally utilized instead of the more expensive cap-sensitive explosive products. A blasting agent is defined as any material or mixture consisting of fuel and oxidizer intended for blasting, which as mixed for use or shipment cannot be detonated by means of a No. 8 test blasting detonator (0.40 to 0.45 g pentaerythritol tetranitrate (PETN) base charge) when unconfined. An explosive material is considered to be cap-sensitive if it detonates with a No. 8 test detonator. Much of the knowledge gained in studying the explosion hazards in oil shale mining can be directly applied to other noncoal mining operations, such as sulfide ore mining.

The Pittsburgh Research Center, as part of its Disaster Prevention research program to improve safety for underground mine workers, initiated numerous research programs (4-5) from the 1970's through the early 1990's under the former U.S. Bureau of Mines (USBM) to define the fire and explosion hazards inherent in the mining of oil shale. In addition, the Mine Safety and Health Administration (MSHA) had reservations about the applicability of existing metal/nonmetal mine safety regulations to oil shale mining. MSHA, therefore, requested that the USBM conduct additional research to evaluate the adequacy of these regulations. This report focuses only on the hazards associated with the extraction of the oil shale through underground mining operations and summarizes oil shale research conducted by the Pittsburgh Research Center

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[6]Italic numbers in parentheses refer to items in the list of references preceding the appendix.

since the previous reports (4-5). Some of the data presented here have been published previously in various forums and are cited in the text.

Historical Development of Oil Shale

Worldwide, oil shale formations are found on nearly every continent. The U.S. Geological Survey (6, pp. 85-98) estimated that over 4×10^{15} t of oil shale exists in worldwide deposits having a potential shale oil yield of approximately 3×10^{17} L. Among the fossil fuels, oil shale reserves are second only to coal reserves in terms of heating value or energy. The larger deposits are located in Asia, Africa, North America, South America, Europe, Australia, and New Zealand. The Green River formation in Colorado, Wyoming, and Utah is the largest concentration of high-grade oil shale in the world.

The shale oil industry has experienced many high (boom) and low (bust) cycles over the years. As early as the 14th century, oil shales of Austria and Switzerland were heated to produce a rock oil, which was then refined to an ointment (6, p. 108). In the United States, Native Americans and early pioneers used pieces of oil shale in their campfires (7). Many oil shale processing plants (6, p. 108) existed in the Atlantic States before Drake's first oil well in Pennsylvania in 1859. These plants converted eastern U.S. oil shale into fuel oils. In 1874, transcontinental rail workers also discovered and used the western U.S. oil shale from the Green River formation in Wyoming in their campfires (6, p. 108). The discovery by Drake and others of crude petroleum in the United States in the late 1800's and subsequent discoveries of abundant quantities of inexpensive oil in the Texas and Arabian oilfields were the primary reasons for the bust in the oil shale industry in the United States and why it had not developed commercially, as was the case in many foreign countries. The People's Republic of China, for example, has large deposits of oil shale and has been producing shale oil for over 60 years (8). Much of China's oil shale overlies thick seams of coal. Because the coal and oil shale are mined together, production costs for shale oil in China are low (8-9). Pulverized oil shale is also mixed with coal and burned directly for power generation in China. In Estonia, most of the oil shale is burned under boilers and contributes significantly to Estonia's electrical power needs (6, p. 108).

In the United States during World War II, when concerns were raised regarding the reliability of imported fuel supplies, the U.S. Congress enacted the Synthetic Liquid Fuels Act of 1944, which authorized the USBM to develop domestic oil shale. The USBM established a facility at the Anvil Points Mine located in the Roan Cliffs west of Rifle, CO (10-11). Many technological advances in the mining and processing of oil shale were developed at this site (10-14). To further promote private development of oil shale, the Federal Prototype Oil Shale Leasing Program was enacted by the U   .   S   .

Congress in 1974. The program enabled large tracts of federally owned lands to be leased by industry to conduct exploration and research aimed toward commercial-scale operations. This, coupled with the 1973 oil embargo, created a boom for oil shale, and many experimental- and commercial-scale facilities were constructed and placed into operation for various periods of time in the 1970's and 1980's. These included Occidental Oil Shale Corp.'s Cathedral Bluffs and Logan Wash oil shale projects (*15*), Unocal Corp.'s Long Ridge project (*3*), Exxon Co. U.S.A.'s Colony Shale Oil Project (*16*), and the White River Shale Project (*17*) jointly owned by Phillips, Sohio, and Sunoco. Figure 1 is a map (adapted from figure 3 of reference *10* and from reference *18*) showing the location of these western U.S. oil shale mines.

Much of the world's oil shale deposits can be extracted through surface mining techniques (*9*). However, to extract the rich oil shales of the Western United States, underground mining techniques must be employed (*12*). Due to low worldwide oil prices, as well as the high initial capital expenditures required to develop an oil shale mine, retort facilities, and refineries, the western U.S. oil shale mines are currently uneconomical, and all had been closed by 1991. However, in the event of another energy crisis that would escalate the price of crude petroleum, many of these facilities could be made operational for only a fraction of the original development costs because the transportation, power, mine, and other infrastructure are already in place. At such time, research by the Pittsburgh Research Center into the explosion hazards of oil shale would again become needed.
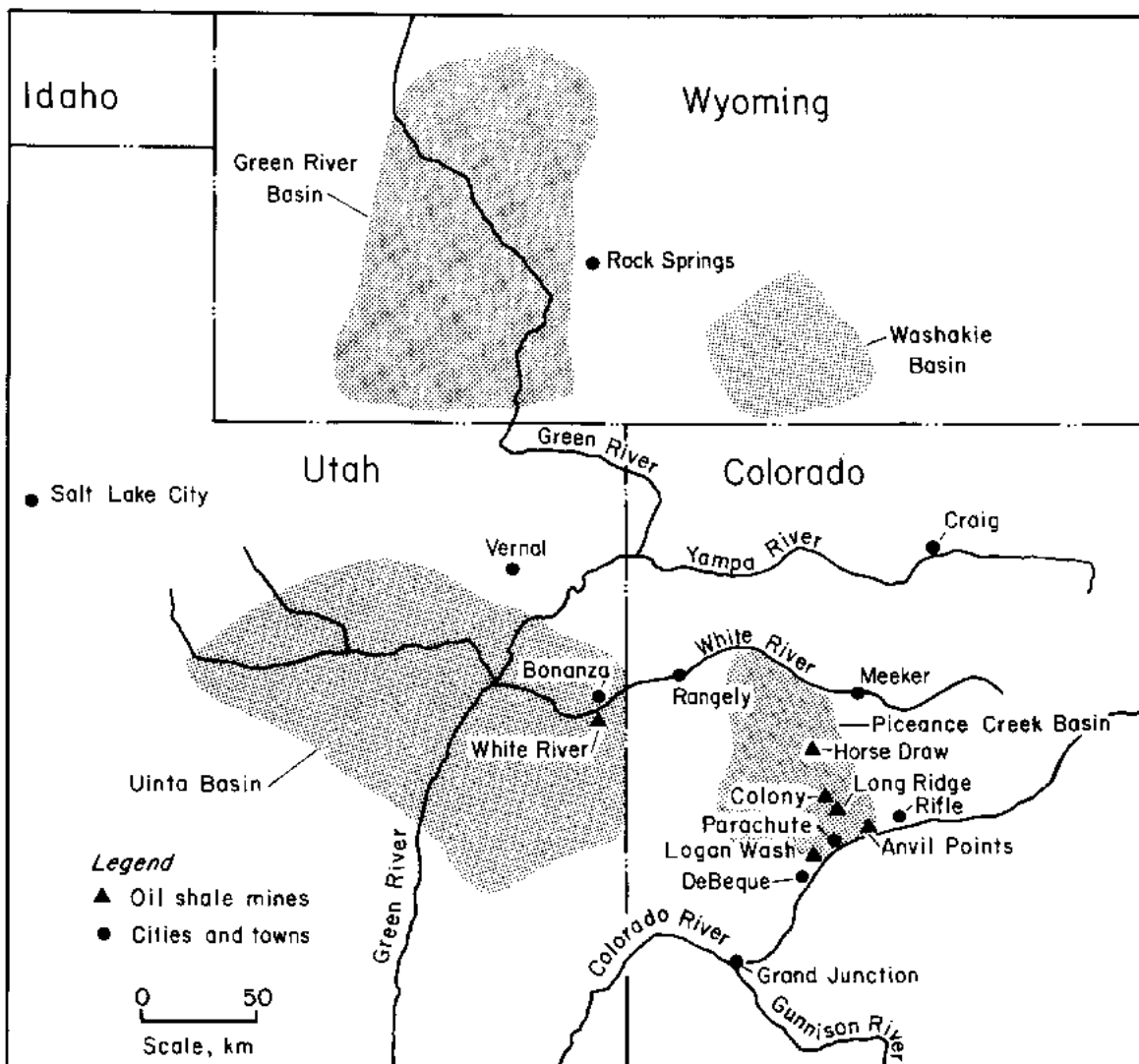
Figure 1.—Map showing four basins of the Green River Oil Shale Formation and locations of oil shale mines.

Properties of Oil Shale

To better understand the fire and explosion hazards of oil shale, the physical and chemical nature of oil shale must first be examined. Note that the research was conducted with oil shales from the Green River formation (*19-21*), and discussions in the remainder of this report are limited to only that formation. The Green River formation covers about 88,000 km^2 in Colorado, Utah, and Wyoming (figure 1). This formation is divided into four major basins: the Piceance Creek Basin in Colorado, the Uinta Basin in Utah, and the Green River and Washakie Basins in Wyoming. These oil shale formations were deposited in the Tertiary period on the bottom of the ancient Lake Uinta. Contrary to its name, oil shale is not a shale rock and does not directly contain oil. Oil shale actually is a lake-deposited sediment that contains organic matter (*6*, pp. 85-98). As the mineral silts and organic matter were deposited simultaneously, heat and pressure formed the deposits into a stable mix of minerals and solid organic matter. These oil shales are a marlstone consisting mainly of dolomite, quartz, and calcite. The solid organic material is known as kerogen.

The kerogen is comprised of hydrogen and carbon molecules that are tied to oxygen and sulfur atoms. The kerogen is bound within the mineral matrix of the oil shale rock. When the oil shale is heated (pyrolyzed) above 200 °C, vaporized shale oil and other gaseous products are formed from the kerogen. These oil vapors are condensed and further processed to obtain the final fuel product. The amounts of oil and gas that can be derived from the oil shale can be estimated by the standard Fischer assay (*22-23*). The standard notation for oil yield or assay by this method is gallons of oil per short ton of shale.[7]

The kerogen and minerals within the oil shale rock are shown in figure 2*A*, which is an optical microscope photograph of particles from pulverized oil shale. Three large individual particles and a group of minus 400-mesh fine particles are shown in figure 2*A*. The structural heterogeneity of oil shale is clearly illustrated by the optical photomicrograph and the three accompanying scanning electron microscope (SEM) x-ray maps (*24*) for the elements calcium (Ca), silicon (Si), and aluminum (Al). In the upper left portion of figure 2*A* is a 600-μm particle that is optically transparent. The three corresponding SEM x-ray maps (figures 2*B*, *C*, and *D*) were made by collecting only

the x rays from a particular element to form an image showing the distribution of that element over the same field of view as in figure 2*A*. A comparison of the three x-ray images corresponding to the 600-μm transparent particle shows that the particle contains a large amount of Ca and only trace amounts of Si and Al. Therefore, that particle is most likely calcitic in composition. In the upper right quadrant of figure 2*A* is an opaque dark particle. This particle is clearly much more heterogeneous in structure than the calcitic particle. Some areas of the particle appear quite dark and are probably high in kerogen content; lighter areas are probably mineral matter. The corresponding x-ray maps show a large amount of Si and lesser amounts of Ca and Al for this particle. Therefore, the mineral matter in this particle is probably mainly quartz. The dark particle in the lower left quadrant of figure 2*A* is even higher in kerogen content, with only a small amount of Si mineral matter as shown by the x-ray maps. The lower right quadrant of figure 2*A* contains a large number of very small particles from the minus 400-mesh sieved fraction. Probably each of these small, individual mineral particles is fairly homogeneous in structure, with most of them high in mineral content as shown by the x-ray maps.

SULFIDE ORE MINING

Sulfide mines provide several metallic ores, particularly of the nonferrous metals such as lead (Pb), zinc (Zn), and copper (Cu). That is why the sulfide ore mines are also called base metal mines. Some of the minerals that compose sulfide ores are pyrite (FeS_2), pyrrhotite (~FeS), galena (PbS), sphalerite (ZnS), and chalcopyrite ($CuFeS_2$). Although there is iron in the sulfide ores, it is usually not recovered. In addition to the base metals, some sulfide ore mines produce small amounts of precious metals, such as silver (Ag) and gold (Au). As in the oil shale mines, developmental headings and production stopes are mined by blasting.

This report summarizes research conducted by the Pittsburgh Research Center during the 1980's and early 1990's on sulfide ore explosibility and the investigation of less incendive blasting agents for mining.[8]

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[8]Some of the data were presented in an unpublished report entitled "Hazards of Secondary Dust Explosions in Sulfide Ore Mining" by E. S. Weiss, K. L. Cashdollar, and M. J. Sapko (Pittsburgh Research Center Internal Report No. 4646, Jan. 1987).

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[7]To convert from gallons per short ton to liters per metric ton, multiply by 4.17.
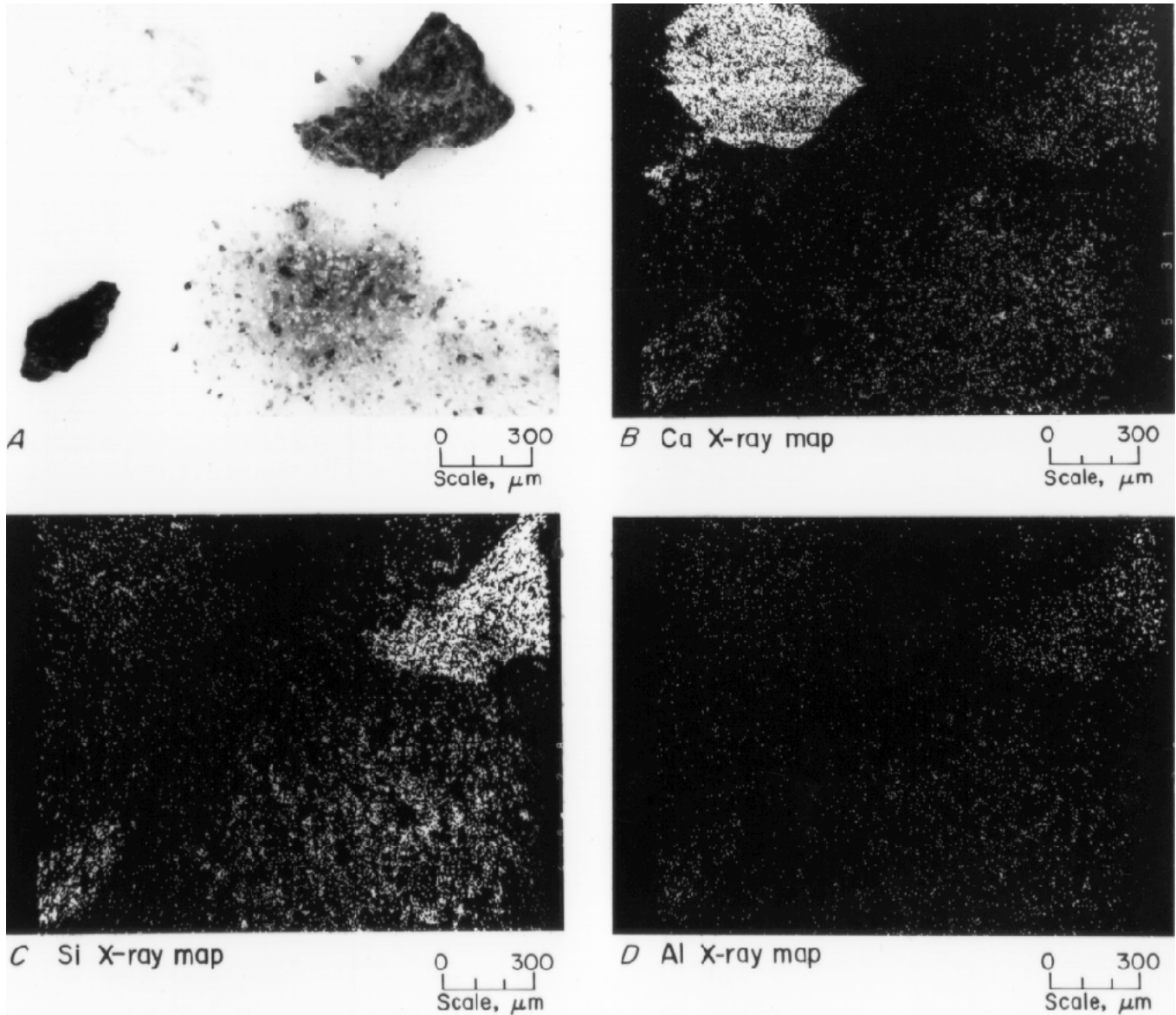
Figure 2.—Optical microscope photograph of oil shale particles *(A)*, and the corresponding SEM x-ray maps for the elements calcium *(B)*, silicon *(C)*, and aluminum *(D)*.

EXPLOSION HAZARDS OF OIL SHALE DUSTS

The rich oil shale deposits in the Western United States are generally extracted by drilling and blasting techniques (*12*) and transported to a surface retort facility. The underground headings are developed by regular room-and-pillar mining methods. Even for in situ retorting techniques (*1-2*), headings are blasted at various levels within the mine to develop the underground retort chamber and raw oil collection facilities. The mining operations in oil shale are regulated under the standards developed for metal and nonmetal mines (*25*). Permissible explosives for use in hazardous areas of metal and nonmetal mines are tested and approved in accordance with 30 CFR 15 (*25*). Concern raised by the USBM and MSHA in the 1970's regarding the adequacy of these regulations for oil shale mining led to USBM research into the explosibility of oil

shale dust.

When fine particles of a combustible dust (oil shale, sulfide ore, coal, etc.) are suspended in an atmosphere that contains sufficient oxygen to support combustion, a dust explosion can occur. In underground mining, the energy required to ignite this dust cloud is supplied by the explosives used in development and production blasting. Along with the useful work energy derived from the detonation of the explosives, large quantities of high-temperature gases and particles are released. Blasting operations also generate large quantities of dust due to the fracturing of the rock. When an ignition of this dust cloud occurs, a local aerodynamic disturbance (wind) is created that stirs up the fine dust particles that had been previously deposited along the back, ribs, and sill of the mine

workings. The flame front from the initial localized dust explosion then develops into a secondary explosion, consuming this new dust cloud and propagating through the mine heading.

LARGE-SCALE EXPERIMENTAL MINE DATA

Very early USBM tests (*26*) in a 20-cm-diam by 5.2-m-long pipe showed that some oil shales were explosible. The initial USBM large-scale studies in the 1970's determined the ignitability characteristics of oil shale dust in terms of its particle size and kerogen content. These data were collected from both experimental mine and laboratory tests. Scores of explosion tests with oil shale dusts were conducted in the 1970's through the early 1980's by Richmond and colleagues (*4, 27-29*) at the Bruceton Experimental Mine (BEM) located at the Pittsburgh Research Center. These explosion tests with oil shale dust were conducted in the main entry of the BEM shown in figure 3. The main entry is 400 m long and has a cross section 1.8 m high by 2.8 m wide. Several types of instrumentation acquire data on the explosions, as shown in figure 4. Pressure transducers that measure the static pressures generated by the explosion were mounted into the face and outby into the rib. At these same stations, 16 optical flame sensors were mounted outby to 200 m. These sensors transmitted signals from which the flame velocity and duration were determined. Optical dust probes (*30-31*) and three-color pyrometers (*32-33*) were also positioned outby the ignition zone, as shown in figure 4. The dust probes measured the relative airborne dust concentrations entrained by the explosion. Flame temperatures could be calculated from the pyrometer data. The raw data signals from the various instruments were recorded by a high-speed computer data acquisition system, as well as on high-speed photographic chart paper oscillographs.

The dust test zone for the oil shale mine tests was usually 92 m long. The oil shale dust was distributed on cross shelves located near the roof of the mine on 3-m intervals. At the face, a 30-m^3 volume of 8.5-pct methane (CH_4) in air was used as the ignition zone, as had been used previously for coal dust tests. An electric match located at the face served as the point source for ignition of the gas. The initial gas explosion would both disperse and ignite the dust.

Based on the BEM tests (*4, 27-29*), it was determined that the explosibility of oil shale dust was a function of its kerogen content and particle size. The data showed that the oil shales that contained higher kerogen content required lower dust concentrations to produce an explosion. The reason is that the dusts with the higher kerogen content emit more combustible volatiles per mass of dust. The particle size distribution of the dusts tested in the BEM were classified as either pulverized or coarse dust. For the pulverized dusts, 100 pct of the dust was

minus 20-mesh (<850 µm) and 57 to 85 pct minus 200-mesh (<75 µm). For the coarse dusts, 90 pct of the dust was minus 20-mesh and 27 pct minus 200-mesh. Data from the BEM tests (*4, 27-29*) showed that explosions did not occur in pulverized oil shale dusts that assayed less than ~85 L/t (~20 gal/st). This is to say that the pure oil shale dusts with incombustible contents greater than 88 pct did not produce explosions. For oil shales with assays from 92 to 208 L/t (22 to 50 gal/st), the minimum nominal explosible concentration ranged from 600 to 200 g/m^3, respectively. Data from the BEM tests also showed that the finer, pulverized oil shale required less dust for an explosion than coarse oil shale of the same kerogen content. The explanation is that the finer particles can pyrolyze and produce volatiles more rapidly in the flame front.

Legend

Bulkhead w/door
‖ Stopping
-□- Stopping w/door
× Overcast

Scale, m
0   50   100
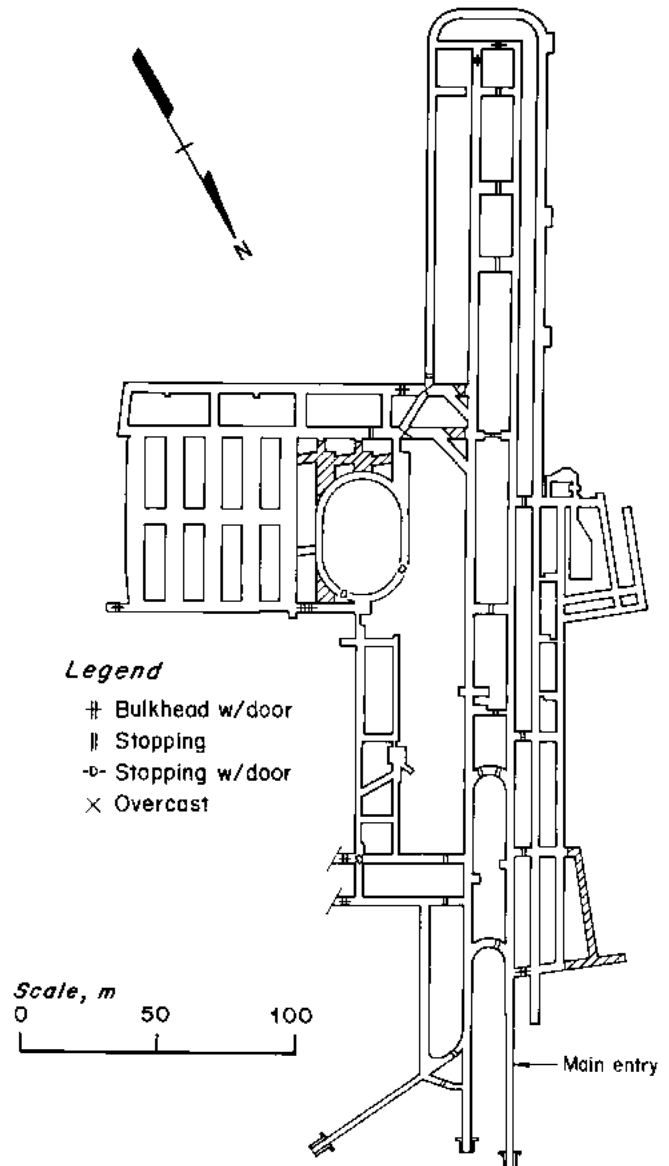
Main entry

Figure 3.—Plan view of the Bruceton Experimental Mine.

Figure 4.—Side view of the face area in the main entry of the Bruceton Experimental Mine showing instrumentation positions.
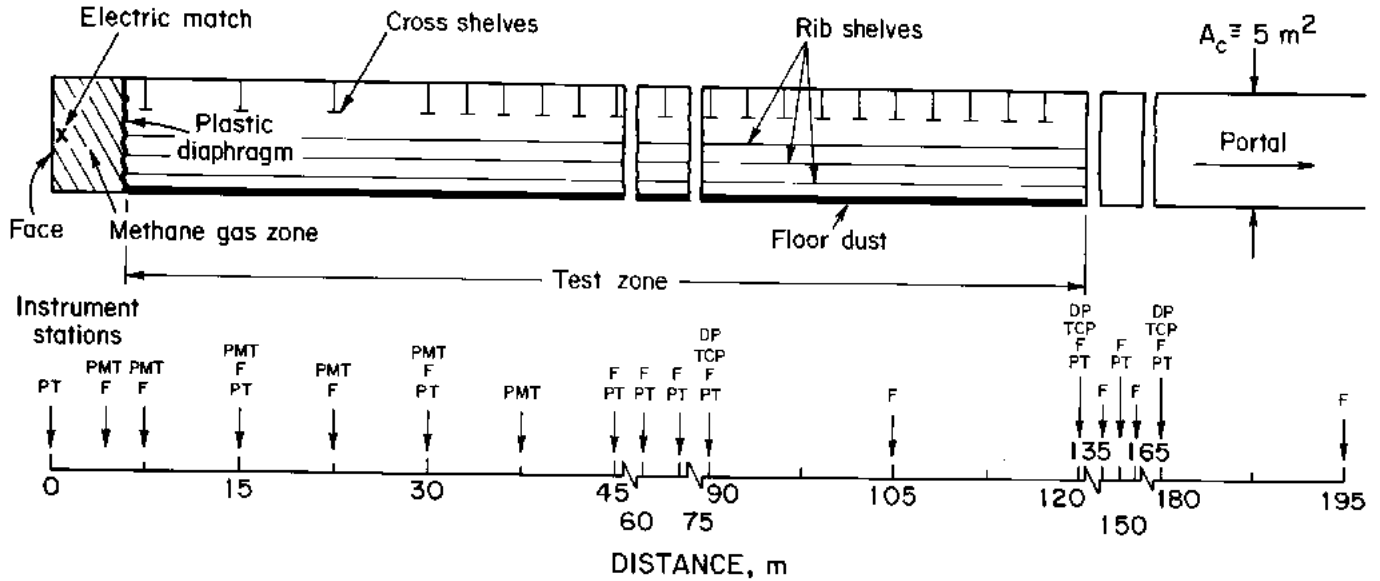
Key

PT	Pressure transducer	TCP	3-color pyrometer
F	LS-400 flame	DP	Dust probe
PMT	Photo multiplier	A_c	Cross-sectional area

LABORATORY DATA

The standard laboratory test chamber used by the Pittsburgh Research Center for studying the explosibility and inerting of combustible dusts is the 20-L chamber (34-36) shown in figure 5. The test procedure included the partial evacuation of the chamber and the dispersion of the dust by a blast of air from the bottom. The ignition source was energized after the pressure had returned to about 1 bar absolute and the dust had been uniformly dispersed. At ignition, there was a moderate turbulence level. Details of the operating procedures and dust dispersion uniformity measurements are in reference 34. Instrumentation with the 20-L chamber includes a pressure transducer, optical dust probes (30-31) for measuring dust dispersion uniformity, and multichannel infrared pyrometers (32-33) for measuring explosion temperatures. The data from the various instruments are collected by a high-speed PC-based data acquisition system. The ignition sources used for the 20-L tests were electrically activated, pyrotechnic ignitors manufactured by Fr. Sobbe GmbH of Germany. For the data reported here, 2,500- or 5,000-J ignitors were used. The 5,000-J ignitor energy was used previously for hard-to-ignite dusts, such as coal and rock dust mixtures (35-36). The oil shales are also hard to ignite because they also contain a large amount of inert material.

The chemical and physical properties of the pulverized oil shale and comparison dusts are listed in table 1. The oil shales are identified by a four-digit number used in previous USBM
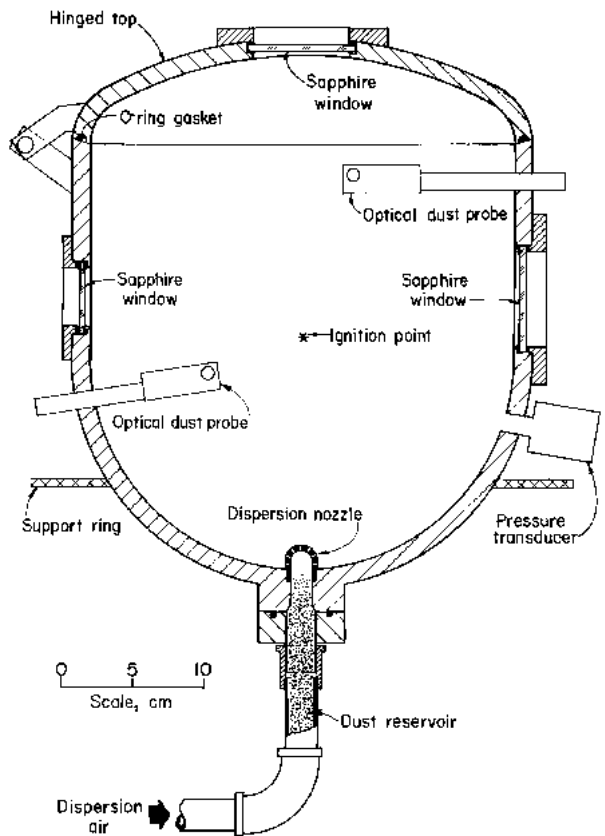
Figure 5.—Vertical cross section of the Pittsburgh Research Center's 20-L dust explosibility test chamber.

Table 1.—Properties of the oil shale and comparison dusts

Properties	Oil shale dusts					Coal dusts[1]		
	5082	6114	6238	0000	5084	hvb	lvb	anthra
Assay, gal/st	19	23	34	42	49	NAp	NAp	NAp
Assay, L/t	82	95	140	174	205	NAp	NAp	NAp
Volatility, pct	9	10	15	19	22	37	17	5
Heating value, cal/g	1,010	1,190	1,790	2,280	2,610	7,700	8,150	7,140
Minus 200-mesh, pct ...	85	57	74	84	78	80	75	77
D_{med}, μm	21	35	19	29	30	48	52	18
$\bar{D}_W$, μm	37	115	53	43	51	51	58	37
$\bar{D}_S$, μm	14	16	12	17	17	32	39	13
$\bar{D}_S$(%T), μm	16-34	12-27	12-29	17-35	16-37	27-58	24-50	24-34

NAp Not applicable.

[1]"hvb" refers to high-volatile bituminous, "lvb" refers to low-volatile bituminous, and "anthra" refers to anthracite.

publications (4-5). These were the same pulverized oil shale dusts used for the mine tests. The comparison coals include high-volatile bituminous (Pittsburgh Seam), low-volatile bituminous (Pocahontas Seam), and anthracite. The Fischer oil shale assay is listed in terms of both gallons per short ton and liters per metric ton in the table. The uncertainty in the Fischer assays is 1 to 2 gal/st. Based on additional data, the Fischer assays have been revised slightly from the values in earlier reports. The volatility is the sum of the oil and gas amounts from the Fischer assays (22-23). Based on the relationship in references 4 and 37, the sum of the oil and gas in percent is about 0.45 times the oil shale assay in gallons per short ton. The total organic content is 0.58 times the oil shale assay. The heating value was measured in an adiabatic bomb calorimeter. The percent passing through a 200-mesh sieve is listed next. The size data are from a combination of sonic sieving and Coulter (electrolytic conductivity through a small orifice) counter size analyses. For the Coulter data, the dusts were dispersed in isopropyl alcohol. D_{med} is the mass median diameter, $\bar{D}_W$ is the volume or mass mean diameter, and $\bar{D}_S$ is the surface mean diameter. Lastly, table 1 lists $\bar{D}_S$ as calculated from the optical dust probe transmission (%T) data measured while the dust was dispersed in the 20-L chamber. As described in reference 30, the dust probe transmission T is related to $\bar{D}_S$ by Bouguer's law:

$$T = \exp(-3QC_m\ell/2\rho\bar{D}_S), \qquad (1)$$

where   Q = a dimensionless extinction coefficient,

C_m = the mass concentration,

ℓ = the path length,

and ρ = the density of a particle.

The 20-L detailed explosibility data (5, 38) for three oil shale dusts of varying kerogen content are shown in figure 6. The 5,000-J ignitors were used for these data. The explosion pressure (gauge) and the pressure rise rate (dP/dt) are plotted versus the dispersed dust concentration. It should be noted that the dP/dt data depend greatly on the particular turbulence level in the 20-L chamber at the time of ignition (36). Summary data for the three oil shales from figure 6 plus a fourth oil shale are shown in figure 7, where they are compared with the data for the two bituminous coals and the anthracite. For these data, 5,000-J ignitors were used for the oil shales and the anthracite; 2,500-J ignitors, for the bituminous coals. For the dusts shown in the figure, the high-volatile bituminous (hvb) coal dust has the lowest minimum explosible concentration (MEC) or lean flammable limit (LFL). It also had the highest explosion pressure and dP/dt. In terms of MEC and explosion pressure, the next most hazardous dust was the low-volatile bituminous (lvb) coal. The 205-L/t (49-gal/st) oil shale had a higher MEC and lower explosion pressure than those of the two bituminous coals. The dP/dt of the 205-L/t oil shale was significantly lower than that of the hvb coal, but was slightly higher than that of the lvb coal. The lower assay oil shales had progressively higher MEC's and lower explosion pressures. For the 82-L/t (19-gal/st) oil shale dust at very high concentrations (800 to 1,400 g/m^3), only about one-half of the tests produced explosions in the 20-L chamber. These laboratory results are comparable with the full-scale mine data (4) that showed that fine-sized oil shales with Fischer assays greater than ~85 L/t (~20 gal/st) could propagate explosions. For the anthracite coal, the slight pressure rise observed was due only to a small amount of burning around the ignition source and does not signify flame propagation. Therefore, this 5-pct volatile anthracite is considered nonexplosible in air in the 20-L tests. Full-scale mine tests (27) have also shown that anthracites do not propagate explosions.

The data in figures 6 and 7 show no evidence for a rich explosibility limit for the dusts. At the higher dust concentrations, the maximum pressures and rates of pressure rise level off as all of the oxygen in the chamber is consumed. At even higher dust concentrations, although the mixtures are nominally fuel-rich, the pressure nevertheless remains constant. The normal rich flammability limit observed for hydrocarbon gases such as CH_4 is not observed for the dusts. An explanation of this effect, at least for many dusts, is that the solid phase fuel must first devolatilize before it can mix with the air. As soon as sufficient volatiles are generated to form a stoichiometric concentration of volatiles in air, the flame front propagates rapidly through the mixture before excess fuel volatiles can be generated.
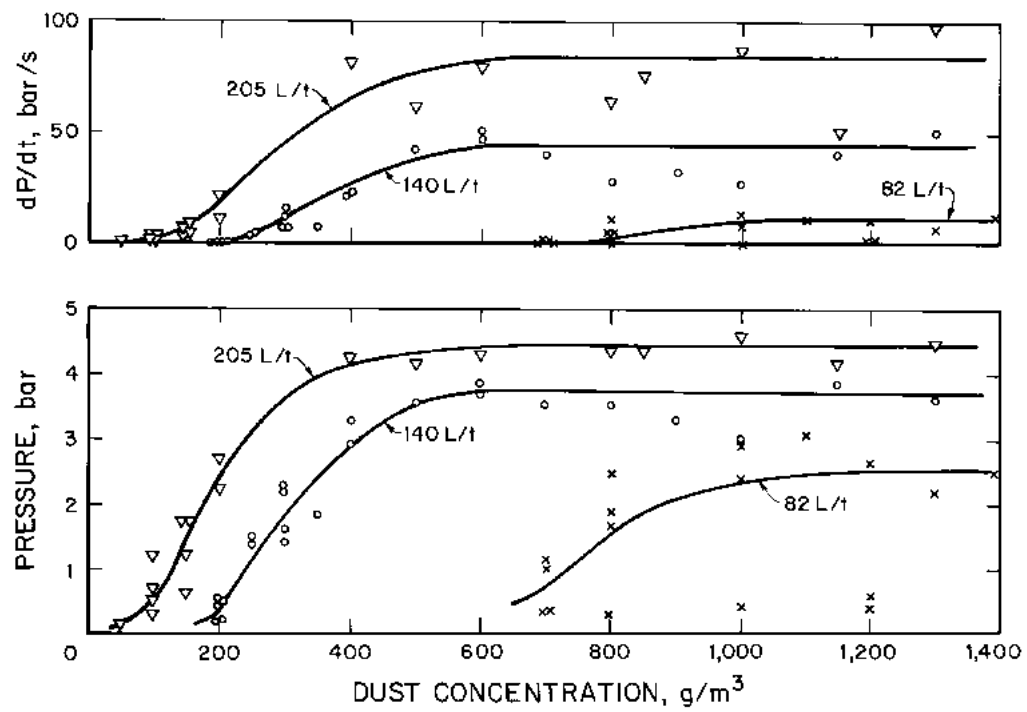
Figure 6.—Explosibility data from 20-L chamber for three oil shales.
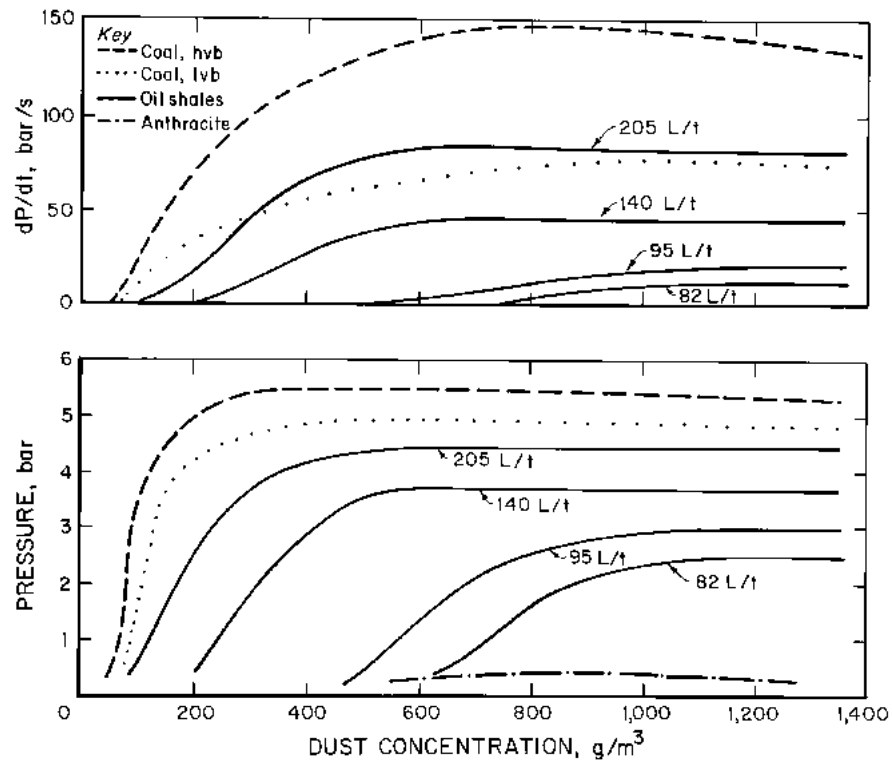
Figure 7.—Summary explosibility data from 20-L chamber for four oil shales compared with high- and low-volatile bituminous coals and anthracite. hvb = high-volatile bituminous; lvb = low-volatile bituminous.

A summary of the laboratory explosibility and ignitability data for the oil shales and comparison coals is presented in ta-ble 2. The MEC data were measured at 2,500 and 5,000 J. The variation in the measured MEC values with ignition energy indicates the ease or difficulty in igniting the various dusts. The lower grade oil shales could not be ignited with the 2,500-J ignitors. The maximum explosion pressures (P_{max}) and the maximum pressure rise rates (dP/dt)$_{max}$ were taken from the data curves in figure 7. As shown in figure 6 and table 2, the 82-L/t (19-gal/st) oil shale only produced explosions at very high concentrations in the 20-L chamber, even with the 5,000-J ignitor.

The minimum autoignition temperature (MAIT) data shown in table 2 were measured in a 1.2-L furnace (*5, 39*). The MAIT values listed in table 2 were modified slightly from earlier reported MAIT values (*5, 38-40*) based on a revised ignition criterion (*41*). The new criterion for ignition in the 1.2-L furnace is flame observed out of the furnace within 3 s, as opposed to the previous criterion of flame within 1.5 s.

The presence of even a small amount of CH_4 gas can lower the minimum explosible concentrations of the oil shale dusts. Hybrid mixtures of oil shale dusts with CH_4 gas were also studied in the 20-L chamber using 2,500-J ignitors. Data for hybrid mixtures of the 205-L/t (49-gal/st) oil shale and CH_4 are shown as the triangle data points in figure 8; the 140-L/t (34-gal/st) oil shale mixtures are shown as the circles. For comparison, data are also shown for mixtures of Pittsburgh hvb coal and CH_4. The area above and to the right of each curve is explosible, or flammable, for that dust; the area below and to the left of each curve is nonexplosible, or nonflammable. The data for mixtures of Pittsburgh coal and CH_4 in figure 8 show a linear or near-linear mixing relationship similar to Le Chatelier's law for hydrocarbon gases (*42-43*). The measured LFL for the pure CH_4 with the 2,500-J ignitor is 4.4 pct, but this is an overdriven system as shown by tests in a larger 120-L chamber (*44*). The more appropriate LFL for CH_4 is the 4.9-pct value measured with a 1,000-J ignitor in the 20-L chamber. The data for the two oil shale dusts show some curvature. This is probably due to the great difference in ignitability between the oil shale and the CH_4, i.e., the dust becomes more easily ignited as small amounts of CH_4 are added.
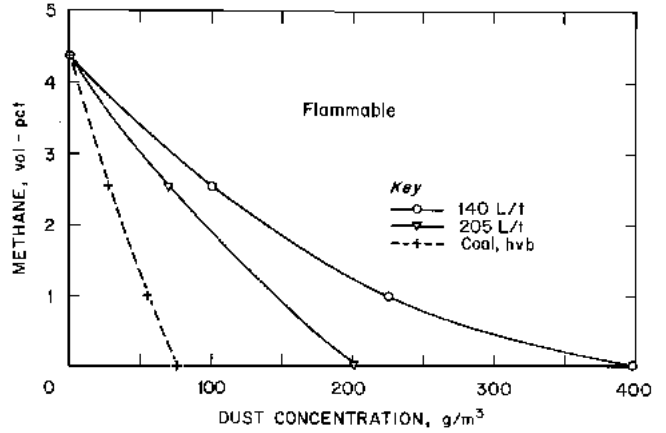
Figure 8.—Minimum explosible concentrations for hybrid mixtures of coal or oil shale dusts with methane gas from 20-L chamber.

Therefore, the curvature is more likely an effect of ignitability rather than of flammability.

There were also some 20-L tests to determine whether CH_4 added to a high concentration of oil shale dust would produce a rich mixture that would be incapable of propagating an explosion. The data in figures 6 and 7 showed there was no rich explosibility limit for the oil shales out to high dust concentrations. In the additional tests, 2.5 pct CH_4 added to 1,000 g/m^3 of the 205 L/t oil shale in the 20-L chamber still produced an explosion.

In summary, laboratory and experimental mine research at the Pittsburgh Research Center has shown that oil shale dust clouds can be ignited and therefore could present a hazard during underground blasting operations. This research has also shown that small amounts of CH_4 can significantly reduce the lower limits of explosibility for oil shale dusts. If one combines the above results with the highly incencive (or flame-generating) blasting agents typically used in oil shale mining, then a potentially hazardous condition exists in the oil shale mining industry. For these reasons, field investigations at several oil shale operations were initiated to measure dust generation, gas emissions, flames, and pressures during commercial-scale mining operations.

Table 2.—Laboratory explosibility data for oil shales and comparison dusts

Explosibility data	Oil shale dusts					Coal dusts		
	5082	6114	6238	0000	5084	hvb	lvb	anthra
Assay, L/t	82	95	140	174	205	NAp	NAp	NAp
MEC, kg/m^3								
@ 2.5 kJ	NA	NI	0.40	0.20	0.20	0.08	0.12	NA
@ 5 kJ	~0.8	0.6	.25	.16	.13	.06	.09	NI
P_{max}, bar	2.7	3.0	3.7	4.5	4.5	5.6	5.0	0.4
$(dP/dt)_{max}$, bar/s	13	22	45	~88	88	145	77	<1
MAIT, °C	490	~490	490	470	450	530	~610	~670

NAp Not applicable.   NA No data available.   NI Nonignitable.

DUST, PRESSURE, AND FLAME DURING BLASTING IN OIL SHALE MINES

The Pittsburgh Research Center was invited to participate in studies of blasting practices at several western U.S. oil shale mines during the 1980's and early 1990's. A summary of these field tests is listed in table 3. The individual blasts are discussed in detail below and in the sections of this report entitled "Full-Scale Field Studies With ANFO" and "Full-Scale Field Tests With a Low-Incendive Blend." In addition to the full-face blasts listed in the table, there were additional single-hole blasts at some field sites.

In March 1983, April 1984, and August 1985, Exxon invited the Pittsburgh Research Center to participate in monitoring several of its full-scale oil shale blasts (45-46). This provided an excellent opportunity to compare data from laboratory and small-scale tests with data obtained from a commercial-scale operation.

Exxon's Colony Shale Oil Project (16) is located north of Parachute, Garfield County, in western Colorado (figure 1). The mine site was developed from an outcrop of the Green River oil shale formation that occurs at an elevation of approximately 2,150 m in this part of the Rocky Mountain Range. Figure 9 is a partial mine map showing the entrance and the blasting zone. This underground test site was designed to accommodate commercial-scale mining. Ceilings 9.1 m high and headings 16.8 m wide were typical dimensions in this type of mining operation.

The first and third blast conducted at Colony Mine originated at the face as shown in figures 9 and 10. The second blast started the development of crosscut 9. Figure 10 is an enlarged diagram of the blasting zone. The positions of the instruments, dust collection plates, and other equipment utilized in the third blast are detailed in this diagram.

In August 1985, one additional face blast (46) was monitored at Colony Mine, and two blasts were monitored at Unocal's Long Ridge Mine. The emphasis on these tests was to acquire more accurate and detailed data in several areas of uncertainty from the earlier blasts. These areas included more accurate sampling of the dust loadings on the rubble pile, measurements of airborne dust concentrations and static pressure generated during the blasts, and the collection and analysis of postblast gas samples.

Unocal's Long Ridge oil shale mine (3) is located north of Parachute, CO (figure 1). The mine enters the Green River oil shale formation from an outcrop 300 m above the valley floor at an elevation close to that of Colony Mine. The mining level includes the rich Mahogany zone. Unocal used a room-and-pillar mining method that extracted about 13,500 t/d for its on-site retorting operations. The shale realized an average crude oil yield of 158 L/t (38 gal/st). The locations of the two face blasts at Long Ridge Mine are shown in figure 11; blast 6 at Colony Mine is shown in figure 12.

One area of concern that was evaluated during these 1984-85 blasts was the amount of combustible oil shale dust generated during the blasting operation. As shown in figure 10, more than 40 sample pans were distributed from 30 m to over 150 m outby the face in room 1 during the third blast at Colony Mine. The pans were 0.36-m^2 galvanized steel plates that were bolted to the mine floor. These plates served as collection surfaces that permitted a representative measurement of the dust fallout

Table 3.—Full-face blasts during oil shale mine field tests

Date	Blast No.	Blasting agent[1]		Company	Mine
Mar. 1983	1	ANFO (u,n)	. . . .	Exxon	Colony.
	2	ANFO (u,n)	. . . .	Exxon	Colony.
Apr. 1984	3	ANFO (u,n)	. . . .	Exxon	Colony.
Aug. 1985	4	ANFO (u,n)	. . . .	Unocal	Long Ridge.
	5	ANFO (u,n)	. . . .	Unocal	Long Ridge.
	6	ANFO (u,n)	. . . .	Exxon	Colony.
Sept. 1988	7	ANFO (u,n)	. . . .	Unocal	Long Ridge.
	8	ANFO (u,n)	. . . .	Unocal	Long Ridge.
	9	ANFO (s,e)	. . . .	Unocal	Long Ridge.
	10	ANFO (s,e)	. . . .	Unocal	Long Ridge.
Aug. 1989	11	Emulsion (s,e)	. . . .	Unocal	Long Ridge.
	12	Emulsion (s,e)	. . . .	Unocal	Long Ridge.
	13	Emulsion (s,e)	. . . .	Unocal	Long Ridge.
	14	Emulsion (s,e)	. . . .	Unocal	Long Ridge.
June 1990	15	Emulsion (s,e)	. . . .	Unocal	Long Ridge.
	16	Emulsion (s,e)	. . . .	Unocal	Long Ridge.
	17	Emulsion (s,e)	. . . .	Unocal	Long Ridge.
	18[2]	Emulsion (s,e)		Occidental	Logan Wash.

[1]"u" refers to unstemmed, "s" refers to stemmed, "n" refers to nonelectric initiation, and "e" refers to electric initiation.
[2]Methane-air zone added at face.
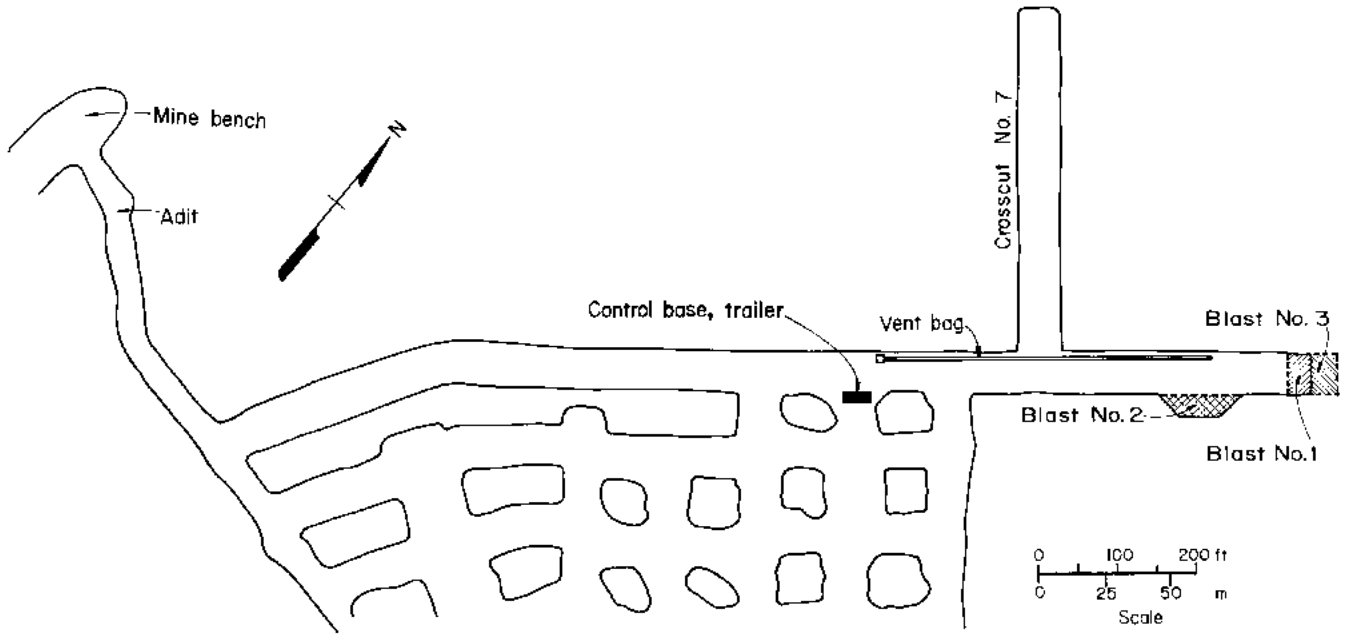
Figure 9.—Map of part of the Colony Oil Shale Mine showing entry and blasting area.

LEGEND

⊢–o–⊣	Curtain stopping
▶	Camera
wwwww	Camera flash
R	Respirable dust sampler
×	Dust sampling pan
⊗	Sampling pan that survived blast
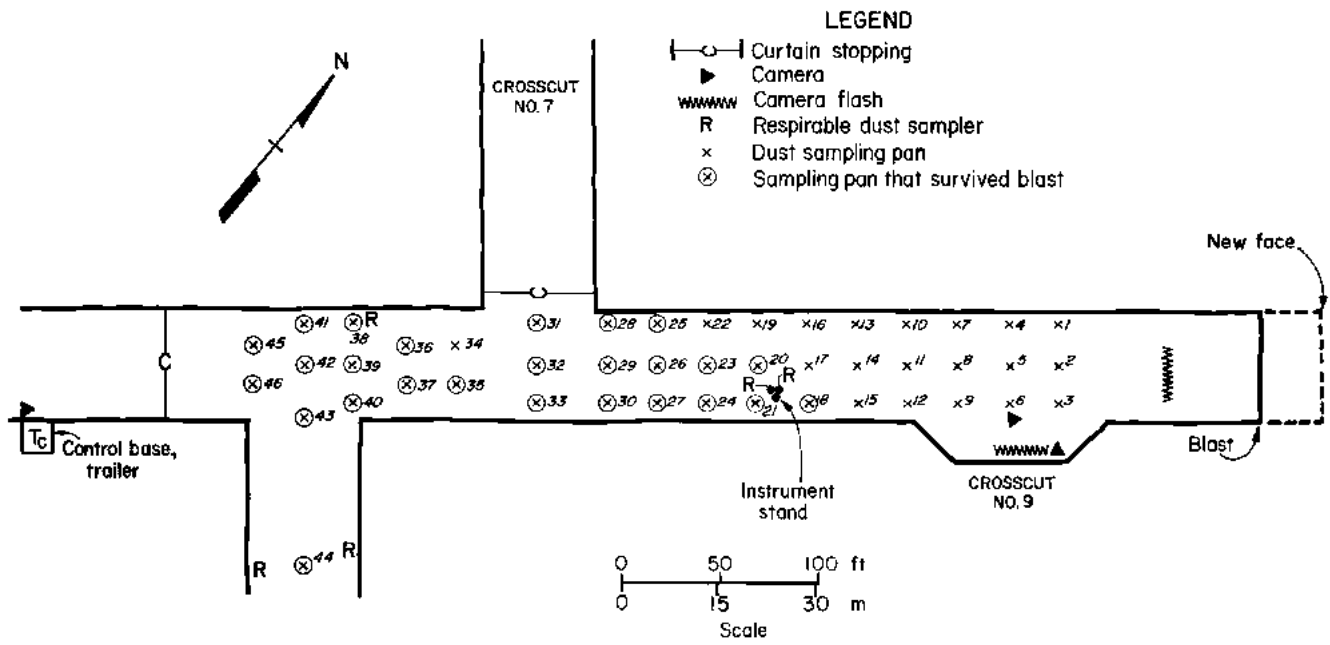
Figure 10.—Map of blasting area showing instrumentation and sampling pan positions for blast 3 at Colony Mine.
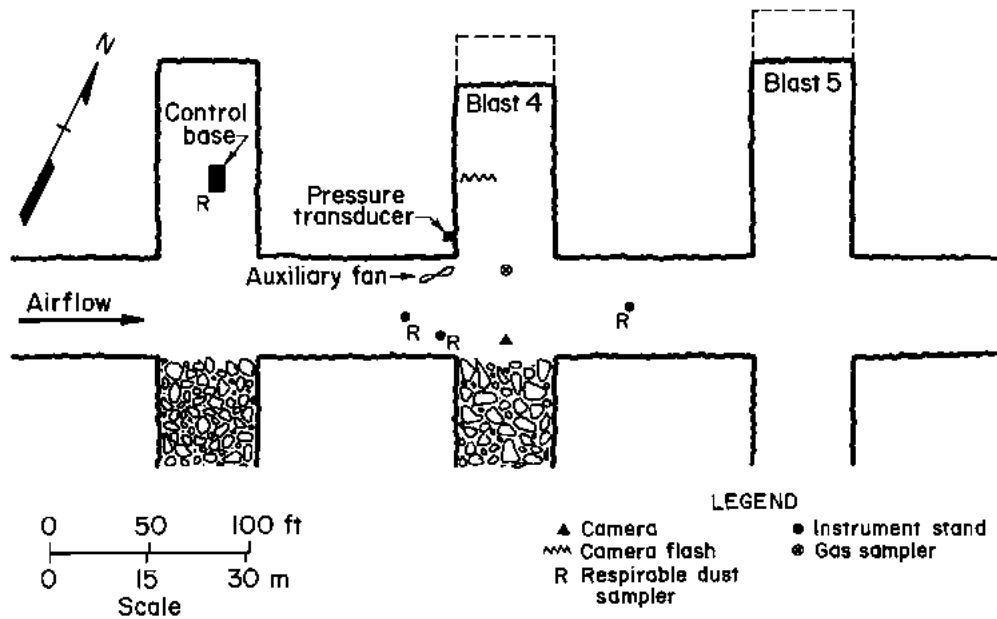
Figure 11.—Map of blasting area showing instrumentation positions and faces for blasts 4 and 5 at Long Ridge Mine.
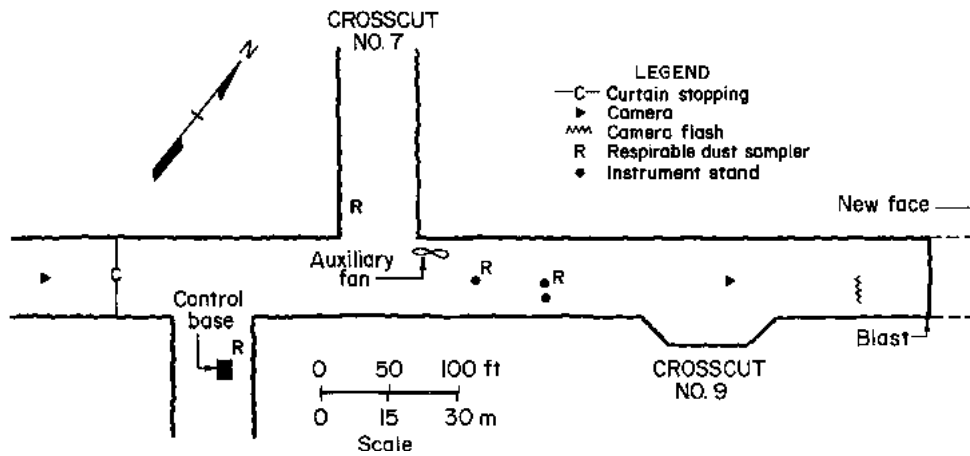
Figure 12.—Map of blasting area showing instrumentation positions for blast 6 at Colony Mine.

associated with the oil shale blast. Based on dust sample size from the first two blasts, the surface area of the pans had been increased for the third blast to allow for larger dust samples, which were needed for more accurate laboratory analyses. As will be described later in this report, sampling of the floor dust loadings deposited during the first three blasts at Colony Mine in 1983 and 1984 provided a reliable estimate of the airborne or float dust concentrations up to 165 m downstream of the original face. However, the postblast rubblization generally covered the collection plates within 40 m of the original face. Therefore, for blast 5 at Long Ridge Mine and blast 6 at Colony Mine, careful attention was directed toward sampling dust from the rubble pile to

determine if the concentration nearer the face may have been of sufficient quantity to represent an ignition hazard. The entry floor, roof, and ribs were washed with water before each of the six blasts to eliminate contamination of the sample plates by preexisting dust. Postblast dust samples were then collected from the plates and the rubble pile. The dust samples collected after each blast were weighed and sized in the laboratory. Only the minus 20-mesh dust was analyzed, eliminating small flyrock and other debris that may have accumulated on the plates.

For blasts 3 through 6 in table 3, respirable dust samplers were also positioned throughout the area to measure the airborne fine dust particles generated from the blast. These units were not meant

to represent the respirable dust to which miners would be exposed because miners were not present during the blasting. The total dust samples collected from the filter cassettes and cyclone cups were weighed, and the filters were also analyzed for quartz content. The sampler units were turned on approximately 0.5 h before the event and continued sampling for 1.5 h.

The instrument stand for blast 3 was positioned approxi- mately 75 m outby the face near the rib as shown in figure 10. This location permitted relative safety from flyrock and other debris that would be generated by the blast. The risk of damage to the instruments from flyrock was the reason for positioning the stands in the crosscuts in blasts 4 and 5 (see figure 11). For blast 6 at Colony Mine, one stand was located approximately 85 m outby the face at midentry; the other stand was 15 m further down the entry (see figure 12). For this blast, the instrument stands were in direct line of sight to the face. Instrumentation on or near the stand locations consisted of drag probes, optical dust probes, pressure transducers, and cameras. Data from the various instruments were recorded on a high-speed chart recorder. Two drag probes of different sensitivities were used in the measurement of the dynamic pressure during blast 3. One drag probe utilized a 9-cm-diam target disk; the other, a 20-cm-diam disk.

Optical dust probes (*30-31*) were installed to monitor the airborne oil shale dust produced during the blasting process. The dust probes measure the transmission through the dispersed dust cloud. Air jets directed over the light source and sensor windows eliminate obscuration due to dust coating the windows. The optical path length for the dust probe was 5 cm. The dust probe data, in conjunction with the floor dust data, allowed for a more comprehensive understanding of the dust generation problem as it related to various blasting techniques. For blast 3, the probe was mounted approximately 1 m from the mine floor on the instrument stand, 73 m from the face. Three optical dust probes were utilized in each of blasts 4 through 6 to provide a better estimation of the dust concentration and to attempt to calculate, from the dust cloud arrival times at the probes, the wind velocities of the dust clouds. In blasts 4 and 5 at Long Ridge Mine, the dust probes were positioned within the first open crosscut about 40 m from the face (see figure 11). Two probes were mounted on one stand: the first 1.5 m and the second 3 m above the mine floor. Across the entry, the other probe was about 1 m off of the floor.

Pressure transducers were installed during each blast to measure the static overpressures developed from the shock waves of the detonating explosives during the blasting operation. A pressure transducer was flush mounted to the right inby rib near the instrument stand for blast 3. A static pressure transducer was flush mounted to the left inby rib approximately 1 m in from the crosscut for blasts 4 and 5. For blast 6, a transducer was located on the right inby rib about 85 m from the face; several other transducers were about 350 m further downstream. A pressure transducer was also installed at the control base for each blast.

Cameras were used to obtain a visual record of the blasts to determine if any flame was evident, to study the dust generation, and, if possible, to witness the actual face destruction. Two 16-mm movie cameras (filming at 32 fr/s) were installed in explosion-resistant steel boxes 40 m from the face during blast 3 (see figure 10). One camera was mounted on the mine roof and viewed the face. The other camera was positioned perpendicular to the

blast heading at the rear of crosscut 9 to observe the turbulent dust clouds resulting from the explosion and also to observe the blast effect on a roof-mounted ventilation bag. A multiple flash bulb system was used for each camera to provide the necessary lighting. The flash sequence was designed to start 1 s before detonation and to illuminate each camera's field of view for approximately 8 s. This was accomplished by using long-duration flash bulbs and firing sequential series of these flash bulbs through the use of time delay relays. During blasts 4 and 5, two cameras were mounted to the mine roof (see figure 11) about 40 m outby the face to document any flame occurrence. One camera was filming at a rate of 32 fr/s; the other, at a rate of 64 fr/s. A multiple flash bulb system was also used during these tests to illuminate the blast face. During blast 6, a camera was mounted on the mine roof 38 m outby the face (see figure 12).

A prototype high-speed, evacuated-vial gas sampling system was field tested for the first time during blasts 4, 5, and 6. This system was capable of obtaining 18 test tube samples in the entry at preset time intervals. The sample time and the interval between samples could be adjusted in advance. The unit was activated by the initial blast wave and programmed to draw gas samples every 2 min immediately following the preshear detonation.

Several variations in the blast hole pattern had been tested by Exxon and Unocal to determine the most effective system for rubblization with minimal damage to the entry roof and ribs. The blast hole pattern, powder factor, and depth of round are the primary factors that can affect the fineness of dust associated with mining of oil shale. The blast hole pattern used for blasts 4 and 5 at Long Ridge Mine is shown in figure 13; that for blast 6 at Colony Mine is shown in figure 14. (The patterns for blasts 1 to 3 were very similar to these patterns.) For blasts 4 and 5, the faces at Long Ridge were 15 m wide by 8 m high and were drilled to a depth of 7.3 m. The preshear and production holes for each blast were 11 cm in diameter. Seven preshear holes were drilled into the face along each rib. Preshear blasting is a technique designed to prevent the shock pressures from the detonating production holes from severely damaging the riblines. The preshear holes were drilled in a vertical pattern parallel and close to each rib. Typically, these holes were then loaded with explosives of smaller diameter than that of the drill hole. The decoupled charges were detonated before the production holes, thereby creating a vertical fracture or crack extending from the roof along the entire length of the holes. This crack reduced the effects of the shock waves produced from the detonating production holes from traveling into and damaging the new pillars. The blast face at Colony Mine was 17 m wide by 9 m high and was drilled to a depth of 9.5 m.

KEY

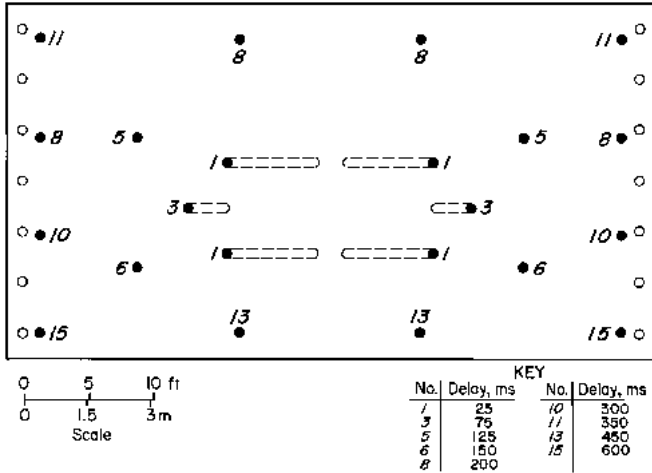| No.	Delay, ms	No.	Delay, ms
1	25	10	300
3	75	11	350
5	125	13	450
6	150	15	600
8	200		

Figure 13.—Diagram of Long Ridge Mine face showing blasting pattern for blasts 4 and 5. Open circles are the preshear charge with no delay; solid circles are the main charge.

KEY

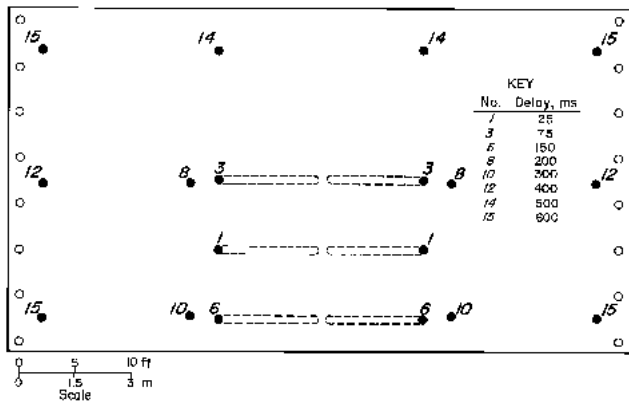| No.	Delay, ms
1	25
3	75
6	150
8	200
10	300
12	400
14	500
15	600

Figure 14.—Diagram of Colony Mine face showing blasting pattern for blast 6. Open circles are the preshear holes with no delay; solid circles are the production holes.

This pattern (figure 14) used one additional preshear hole along each rib and contained four fewer production holes. The preshear and production holes for the blast were 11 cm in diameter. The detonation of the preshear holes 25 ms before the initial detonation of the production holes provided the necessary expansion space, similar to that of undercutting a coal face, and minimized rib damage. The dashed lines in both figures 13 and 14 show the 32° V-cut at the center of the faces. These angled holes are detonated in pairs to provide a relief area for the holes detonating later in the round. Each of the 16 preshear holes in both patterns were loaded to within 1 m of the collar with 2.5-cm-diam explosive charges. The other production holes were loaded from the back to within 5 m of the collar at Colony Mine (50 kg ANFO per hole) and to within 3 to 5 m of the collar at Long Ridge Mine (20 to 36 kg ANFO per hole). Nonelectric detonators with time delays of 0 through 600 ms were used to initiate the holes.

The chart recorders, power supplies, and timer-delay initiation system were located in a mobile control base that was positioned in a relatively safe area nearby the blast face. A timer and relay system was used to sequence the start and stop of the recorder, cameras, flash system, and compressed air for the dust probes; it also initiated the blast round. The timer system activated the monitoring equipment 1 s before detonation. This allowed the equipment to achieve full power and establish stable baselines. Detonating cord was used to initiate the detonators in the blast holes.

Figures 15 and 16 illustrate the face area before and after blast 6 at Colony Mine as seen from the instrument stand. The square markings on the face (figure 15) outline the preshear holes, adjacent to each rib, and the angled center holes. Note the horizontal flash assembly mounted on the roof (about 15 m from the original face) in the upper center of both figures. The detonation of this round generated approximately 3,000 t of oil shale rubble. The shale rubble extended out nearly 70 m from the face. The fragmentation and distribution of the shale (figure 16) were also more uniform than those of earlier tests without preshear holes.

The larger boulders shown in figure 17 were typical for blasts 4 and 5 at Long Ridge and resulted from the design of the blast hole pattern. The larger material was preferred at this mine because of its retorting process and problems with processing fines. The rubble pile extended outby the original face about 50 m for these blasts.

Dust samples were collected from the collection plates and/or measured rock surfaces at various locations on the rubble piles for all face blasts, except blast 4. The samples were weighed, sized, and assayed. Only the minus 20-mesh fraction of the dust from each sample was analyzed, eliminating the small flyrock and very coarse particles, which would not contribute to an explosion flame.

Figure 18 shows the floor dust loadings generated during each of the 1983-85 blasts. Note that the distances in this figure are specified relative to the new face after the blast. The volumetric concentrations on the right ordinate are calculated assuming a uniform dispersion of the dust throughout the cross section. The dust collected during blasts 1, 2, and 3 provided a reasonable estimate of the dust loadings from 40 to 165 m outby the new face. The samples in blasts 5 and 6 were taken exclusively from the rubble piles to determine if the nominal dust concentrations nearer the face were high enough to promote secondary dust ignitions. The nominal concentrations on the rubble piles (from blasts 5 and 6) ranged from 6 to 28 g/m^3 as the distance to the face decreased. These average concentrations were an order of magnitude below the experimentally determined lean limit concentrations (*4-5, 38*) for explosions of fine-sized oil shale dusts of similar grade. The nominal concentrations outward from the pile steadily declined to a low of about 0.1 g/m^3 or less at 150 m downstream.

Some of the floor dust samples collected during the 1983-85 blasts were also analyzed for size distribution and oil assay. The surface mean diameters, $\bar{D}_s$, and the volume or weight mean diameters, $\bar{D}_w$, were calculated from the measured size distributions of the minus 20-mesh fractions. Within 45 m of the face, $\bar{D}_s$ ranged from 15 to 137 μm, $\bar{D}_w$ ranged from 50 to

Figure 15.—Mine face before blast 6. *(Photo by Kenneth L. Cashdollar, Pittsburgh Research Center.)*

Figure 16.—Mine face after blast 6, showing rubblization. *(Photo by Kenneth L. Cashdollar, Pittsburgh Research Center.)*

320 μm, and the minus 200-mesh fraction ranged from 10 to 81 pct. No systematic variation with distance was found within this area. However, the powder factor and the blast hole pattern can affect the size distribution of the dust. For the floor samples farther from the face (70 to 140 m), $\bar{D}_s$ ranged from 22 to 28 μm; $\bar{D}_w$ ranged from 70 to 140 μm. These finer sized dust samples at greater distances from the face, beyond the rubble pile, are consistent with the expected results of airborne transport of dust particles.

The floor dust samples from blast 3 at the Colony Mine were generally too small to obtain Fischer assay data; therefore, the assays were calculated from the measured heating values based on the previously determined linear relationship (4). Through-out the rubble pile, the calculated assays of the Colony Mine floor dust samples from blast 3 ranged from 125 to 142 L/t (30 to 34 gal/st). Farther from the original face, the assay dropped to about 108 L/t (26 gal/st) at 100 m and to 92 L/t (22 gal/st) at 135 m outby. These results are consistent with previous data (4-5, 38) that showed a lower assay for the finer sized particles if oil shale dust is size separated. In the face blasts, the dust was apparently size separated by the natural dynamics of the air flows, whereas previous laboratory data came from dust that was separated by mechanical sieving. For the mine dust samples from the rubble pile for blast 6 at Colony Mine in 1985, the assay of the floor dust samples was approximately 100 L/t (24 gal/st), somewhat lower than that from the earlier blast 3 at Colony Mine. For the mine dust samples from the rubble pile for blast 5 at Long Ridge Mine in 1985, the assay of the floor dust samples ranged from 63 to 103 L/t (15 to 25 gal/st).

The analyses of the dust samples collected from the respirable dust sampler devices showed that the quartz content of the samples ranged from 2 to 5 wt pct. The measured total mass concentration collected by the samplers would be consistent with the measured floor dust loadings at the same locations if the majority of the dust was deposited on a time scale of the order of a few minutes.

Figure 17.—Pittsburgh Research Center researchers with large boulders in the rubble pile.
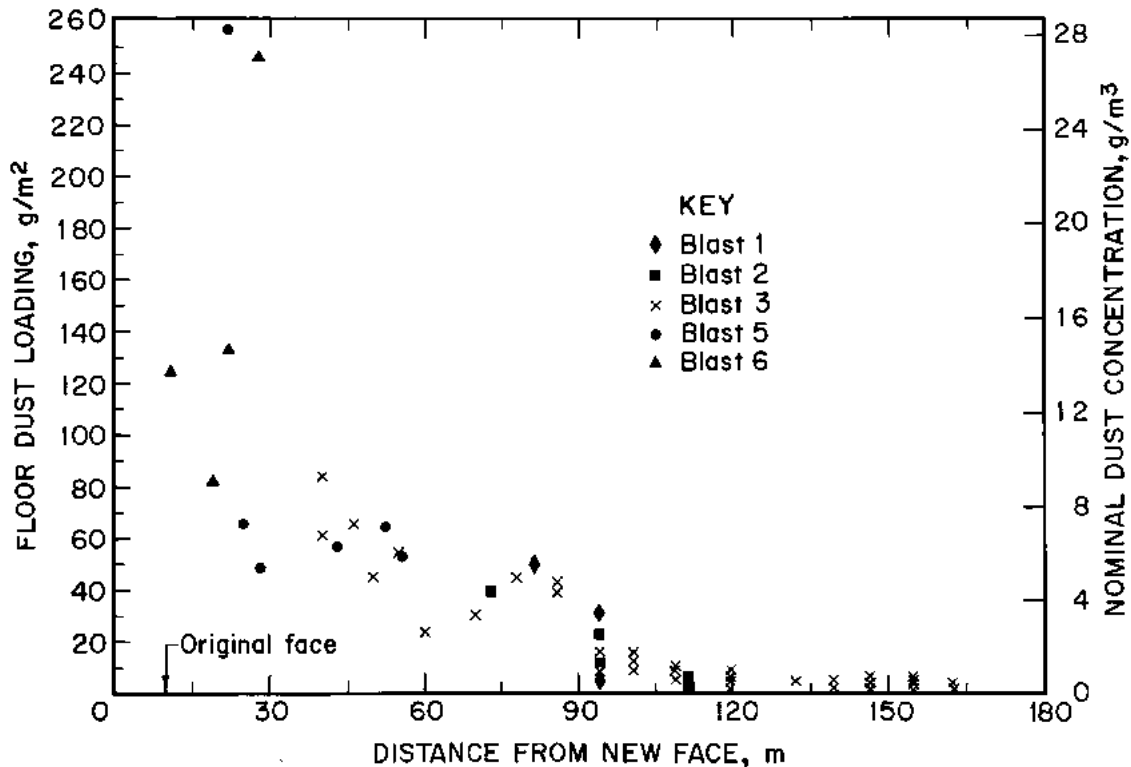

Figure 18.—Summary of floor dust loadings (minus 20-mesh fraction) after five blasts.

For blasts 4, 5, and 6, very low dust concentrations were calculated from the high transmission data (95 to 99 pct) recorded by the optical dust probes. However, even these transmissions over a 5-cm optical path length of the probe would correspond to a very low visibility over a distance of a few meters. The measured transmission values of 95 to 99 pct are consistent with the measured floor dust loadings at the positions of the dust probes. Close to the face, there were probably higher airborne dust concentrations, as shown by the billowing clouds in the high-speed movies and the higher floor dust loadings on the rubble pile near the face.

Figure 19 is a plot of the signal from the pressure transducer during blast 3. Approximately 0.25 s after the blast initiation, the static wall pressure rose to a peak of 100 mbar. This pressure pulse is believed to be associated with the preshear holes and the initial center angled holes, since it would take about 0.25 s for the pressure pulse at the face to travel at the speed of sound in air to the instrument stand 73 m from the face. The small signals earlier than 0.25 s were probably associated with vibrations of the solid mine rock itself, inasmuch as the speed of sound in rock is about 20 times the speed of air. The dynamic pressure measured by the drag probes was approximately 26 mbar, corresponding to a wind velocity of about 60 m/s.

The static pressure was approximately 0.3 bar at about 25 m from the face for blasts 4 and 5 at Long Ridge Mine. The pressure 65 m from the face was 40 mbar; the pressure 115 m from the face was approximately 20 mbar. At Colony Mine, the measured pressure was about 100 mbar at a distance of 85 m from the face during blast 6. This pressure was nearly the same as the pressures measured during blasts 1 and 3 that occurred in the same blasting zone. The peak pressure recorded at the control base located 145 m from the face in blast 6 at Colony Mine was about 50 mbar. Additional transducers were mounted on several test stoppings located over 430 m from the blast face during blast 6. The maximum recorded pressure exerted against these ventilation stoppings was about 11 mbar. A cyclic pressure wave was developed from blast 6 that was clearly evident after viewing the movie film taken of a ventilation curtain located about 155 m from the face. The curtain was being alternatively pulsed outward, then inward as it encountered the successive shock waves generated from the face detonations and rebounding pressure waves. The pulses decreased in intensity and the interval between pulses became longer as the
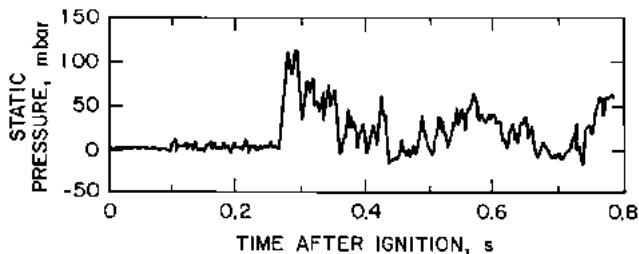
rebounding pressure waves gradually faded. In blasts 4, 5, and 6, the pressure decayed approximately linearly with distance, except when it reached an area with multiple rooms and passageways, where it decayed more rapidly. A computer model could be developed to predict the pressures that would be generated for a particular blasting pattern and powder factor; these predictions could be applied in the construction of durable ventilation stoppings for these large headings.

The use of the prototype high-speed, evacuated vial sampling system resulted in very limited samples. During two of the three blasts, flyrock damaged the unit and/or severed the pneumatic lines and supports. In one blast, however, the unit was able to obtain 18 gas samples over a 34-min period. Analyses of these samples showed trace amounts of hydrogen (H_2) and CH_4, perhaps generated from the cracking of the shale kerogen as it was heated by the explosives. Another source of these gases would be the detonation byproducts from the explosives. These measurements were in the absence of any net ventilation flow and would be reduced greatly after the main fans were turned on. The CH_4 amounts were negligible compared with those in gassy mines, such as the Horse Draw and White River Mines (*4, 29*). This was expected since both the Colony and Long Ridge Mines were near outcrops and had not detected any CH_4.

The two 16-mm movie cameras were successful in observing the face during each blast. Frames from the movies of blast 6 are shown in figure 20. The start of the blasting sequence was clearly evident by the brilliant flash (figure 20*A*) of the detonating cord as it initiated the time delay detonators in the preshear and production holes. Figure 20*B* shows flame from an unstemmed blast hole during the detonation of the angled holes in the center approximately 45 ms into the blasting sequence. Figure 20*C* shows the blasting face about 110 ms after ignition. The lighting on the upper right of these photographs (figure 20*B* and *C*) is due to the multiple flash system; that on the left is due to the flame exiting from the borehole. Flame was evident for at least 220 ms after ignition, at which time the dust and rock totally obscured the view of the cameras. Similar results were observed during blast 3 in this mine and during blasts 4 and 5 at Long Ridge Mine. The flame luminosity lasted too long to result solely from the explosives themselves; it is uncertain from the film whether the flame was the result of afterburning of the detonating cord or burning oil shale dust clouds. It is possible that there were localized areas where the dust concentrations were above the lower limit requirements for an oil shale dust ignition. However, any ignitions that may have resulted from these dust clouds would have been small localized occurrences that would not be able to propagate very far based on the measured average floor dust loadings. There was no evidence of flame luminosity after the end of the blasting sequence.

Figure 19.—Pressure generated by blast 3.
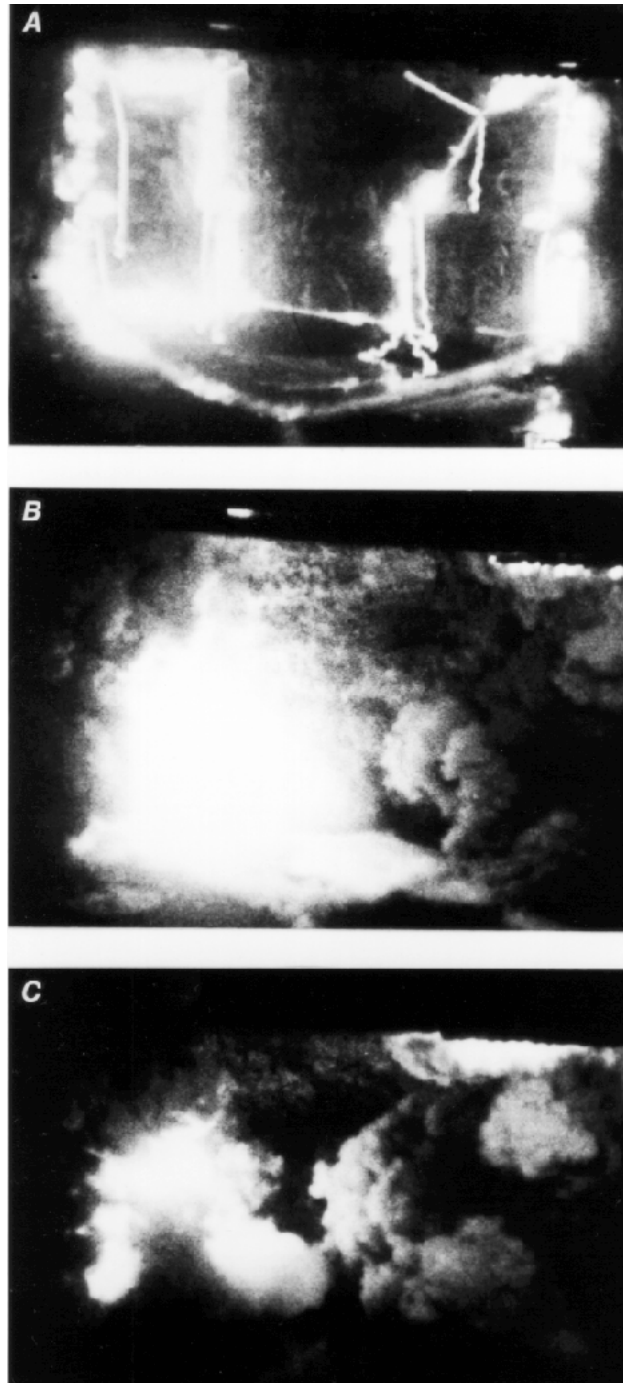
Figure 20.—Frames from high-speed movie during blast 6. *A*, initial face detonation; *B*, luminosity at face at 45 ms; *C*, luminosity at face at 110 ms, with horizontal flash system in upper right of frame.

Exxon's Colony Mine and Unocal's Long Ridge Mine were known to be nongassy due to their proximity to the outcrops. However, CH_4 posed an explosion hazard in other oil shale mines, which will be discussed in the next section of this report. Experimental results have shown that the presence of CH_4, even in small quantities, will significantly decrease the lean limit concentration of oil shale dust required to initiate an explosion

(*4*). The combination of CH_4 liberated from the fragmentation of the blasted shale, the background CH_4 emissions con- tinuously released from the developed headings, and the fine-sized dust generated in localized high concentration clouds at the face would pose a potential localized ignition hazard during some underground oil shale mining operations.

GAS GENERATION DURING OIL SHALE MINING

HORSE DRAW OIL SHALE MINE

Generally, CH_4 was not associated with the early oil shale mines in the Green River formation. All of the mines were developed from oil shale outcrops, and CH_4 was never detected. In 1977, a 732-m-deep shaft was driven into the Piceance Creek oil shale basin at Horse Draw, in Rio Blanco County, Colorado (*47-48*). CH_4 was encountered at this location, confirming earlier core data (*49-50*). Because this area was at least 24 km from an outcrop and located below two aquifers, the CH_4 did not have the opportunity to migrate out of the formation. Matta and others (*50*) indicated that the amount of CH_4 present in any particular formation is proportional to its kerogen content, distance from an outcrop, and depth. An increase in any one or more factors increased the potential for CH_4 to be encountered in the formation.

From April 1979 to October 1981, Multi-Mineral Corp. had operated Horse Draw Mine under a cooperative agreement with the USBM and the Bureau of Land Management. Part of Weichman's research (*51*) was directed toward analyses of cores for CH_4 content. Unlike the study of Matta and others (*50*), no conclusive data could be obtained to link CH_4 content to the kerogen content or depth of formation. However, the differences in the CH_4 content from the two studies may be due to geological conditions, such as fractures and/or the numerous occurrences of leached nacholite zones. These zones of porous nacholite may store CH_4 in quantities above those predicted from the original study. Oil shale, unlike coal, has a low permeability, and the CH_4 would not easily be liberated in advance of mining.

It is important to detect CH_4 in a mining operation for safety reasons. Once CH_4 is detected, it should be quantitatively monitored during the mining operations and, if possible, correlated with the CH_4 content found in cores. If this approach is successful, the quantity of CH_4 to be expected in future mining operations can be better predicted.

In 1980, the Pittsburgh Research Center installed a gas monitoring system (*52*) at Horse Draw Mine to quantify the amounts of CH_4 generated during blasting operations. A gas monitoring trailer was set up on the surface. Carbon monoxide (CO) and CH_4 were monitored continuously by means of tube bundles into the mine. Gas concentration data were recorded in analog fashion on strip charts. Pitot probes were installed in exhaust fan ducts at various levels to measure the air flows in order to compute the gas emissions at those levels. During underground blasting, the personnel were evacuated to the surface, but the gas emission and air flow data continued to be collected through the tube bundles to the surface trailer. By continuous monitoring, the total gas emissions, including emissions from blasting, mucking,

and background, were calculated and compared with the mass of oil shale realized from the blast round and from core samples.

As the mining progressed in the various levels of Horse Draw Mine, the tube bundles were extended to provide data on the CH_4 levels in the vicinity of the blast face. A CH_4 emission rate of 1.6 m^3/t of oil shale was determined to be a reasonable value for the ventilation design based on the data gathered by the gas monitoring trailer (*52*). This value was in the same range as that reported from outgassing of cores taken nearby in the formation (*49-50*). Continuous background CH_4 emissions ranged between 0.37 and 0.51 m^3/min both during the mining operations and shortly after the cessation of mining in late 1981 (*53*).

In 1988 and 1992, further gas samples were collected and analyzed as part of a continuing study of background CH_4 emissions (*54*). At the Horse Draw oil shale mine site in Colorado, several gas samples and air velocity measurements were taken from a vent pipe protruding through a concrete shaft seal (part of the 1986 site reclamation). All mining at Horse Draw had ended in late 1981. Natural ventilation occurred in the mine due to the temperature gradient between the outside ambient air and the atmosphere inside the mine. This natural draft could vary between 50 and 100 m^3/min. Analyses of the 1988 samples showed that the mine continued to liberate CH_4 at a rate of approximately 0.74 m^3/min. Samples collected in 1992 showed that the mine was still liberating CH_4 at a rate of 0.63 m^3/min. Although these rates were somewhat higher than the previous data (*53*), they are believed to be more accurate since all venting was through a single small opening instead of several conduit openings as in the earlier sampling studies.

WHITE RIVER OIL SHALE PROJECT

In 1984, the Pittsburgh Research Center monitored gas emissions during development blasting operations at the White River Shale Oil Project. An isometric view of the White River Shale Project (*17*) mine is shown in figure 21. The project is located on Federal leases Ua and Ub in Uinta County, UT. The air shaft shown on the left of figure 21 is about 335 m deep. The mine was designed to extract over 180,000 t of oil shale per day from the rich Mahogany zone of the Green River formation. Room-and-pillar mining was to be used, leaving rooms about 18 m high by 16 m wide after benching. The development headings were 4 m high by 6 m wide and drilled to a depth of 3 m for blasting rounds.

Gas Monitoring System

The gas monitoring system trailer had been returned to the Pittsburgh Research Center after cessation of mining at Horse Draw in 1981. The system was redesigned for improved performance and monitoring capabilities. In March 1984, the gas monitoring trailer was positioned near the headframe of the White River oil shale mine (figure 22). It was operated by White River Mine personnel. The trailer was environmentally controlled to extend the operating range of the gas monitoring system to any type of environment. In the event of a power outage, the trailer was furnished with a backup battery system to ensure an uninterrupted power source for an additional 4 h. A local alarm sounded to alert the mine personnel of power outages and dips.

The gas monitoring trailer was connected to a 10-tube bundle of fire-resistant tubes, each of 1-cm-diam polyethylene. The tubes were extended down the shaft; as mining progressed, some of the tubes were used to follow the face and some were used to provide local information regarding CH_4 and CO levels in case of fire. Figure 23 shows a flow diagram (53) of the gas monitoring system. Pumps installed beneath the trailer draw gas samples through the tube bundles into the gas analyzer.

Any water in the tubes and/or dust in air was removed by filters and desiccant. Scanning valves connected the analyzers to a programmed series of gas and pressure port positions selected by the operator for analysis. The system had a capacity of 24 separate sample positions throughout the mine.

Gas analyzers determined the concentrations of CH_4 and CO by measuring the infrared radiation absorbed by these gases. When any of the sampling positions were 10 pct greater than the previous sample, the computer held on that position for three consecutive samples; if there was no further increase by more than 5 pct, it would resume its programmed sequence. This feature was particularly important because it alerted the mine personnel to unexpected rises in gases during off-hours. The system could also be programmed with limits on gas concentrations and could be interfaced with local alarms to alert mine personnel when gas limits had been exceeded.

By measuring the pressure differentials and flows within individual tubes in the bundle, the system had a built-in method of self-checking for leaks and/or plugs. The air flows and pressures within the underground mine vent pipes could also be monitored, along with air velocity measurements accurate to 1 m/s in the entries or shafts.

The gas analyzer and its associated functions were all controlled by a microprocessor. The microprocessor had a large storage capacity able to hold the data until such time as it was removed from the memory remotely or to the local terminal. Strip charts were also available as an additional record in case of a computer failure. A main computer located at the Pittsburgh Research Center automatically retrieved the microprocessor data at regular intervals. The raw data were then processed and plotted using the latest calibration values.

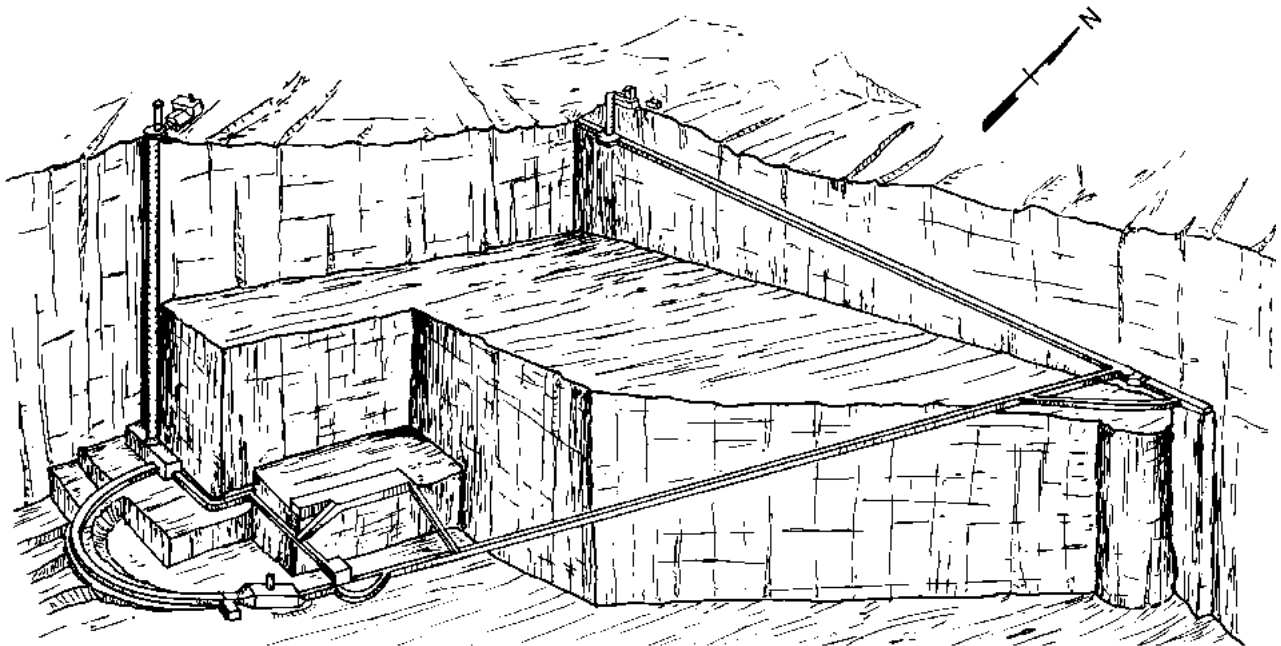Figure 21.—Isometric view of White River Shale Project mine showing decline on right and vertical air shaft on left.

Figure 22.—Gas monitoring trailer near air shaft at White River Mine.
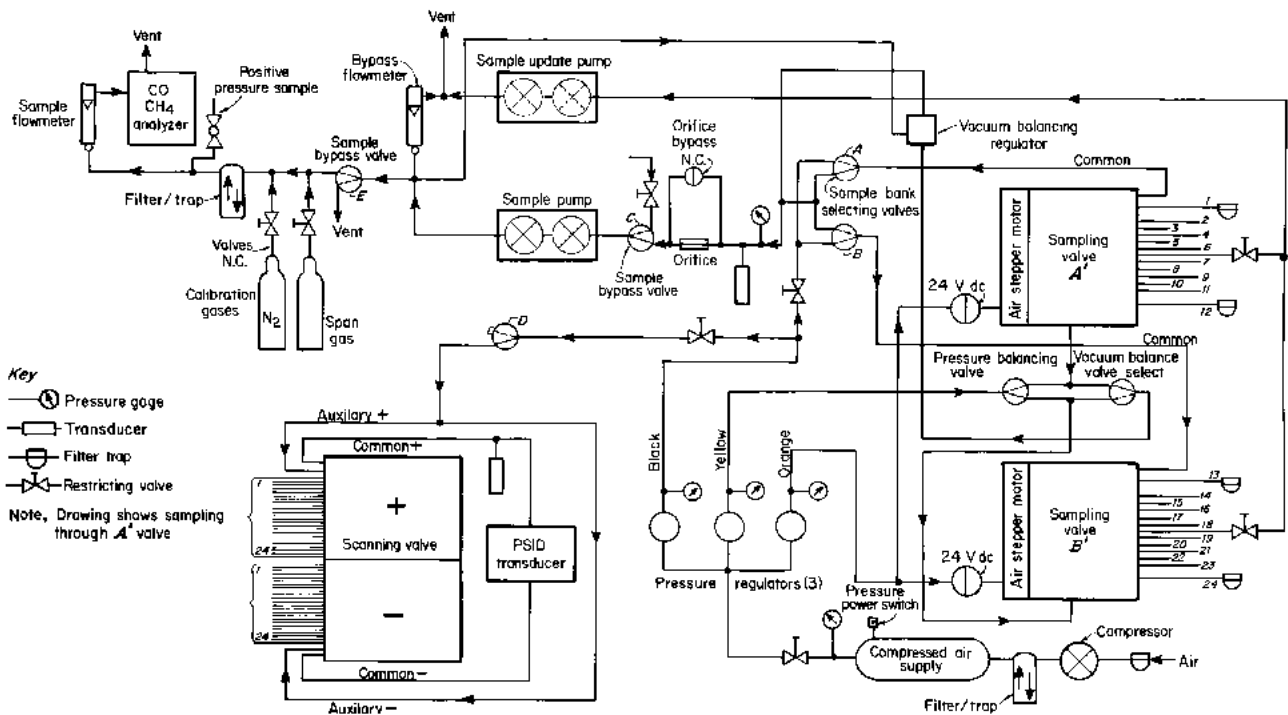

Figure 23.—Flow diagram of gas sampling system. (Drawing shows sampling through _A'_ valve.)

Methane Emissions During Mining Operations   A typical plot of CH$_4$ emissions following a blast at the White

24

River oil shale mine is shown in figure 24. Zero time on the plot corresponds to the time the CH_4 first arrived at the gas analyzer. The average background level of CH_4 (0.02 to 0.03 vol pct) was subtracted from these emission data. As seen in figure 24, the CH_4 level associated with the rubblization of the rock rose to 0.39 vol pct of the total ventilation flow. It eventually declined back to the background emission level.

The CH_4 emissions generated from the east heading development at White River in 1984 were divided into two time periods. The initial emission was defined as the CH_4 measured at the shaft collar during the first 50 min following the blast. These CH_4 emission data ranged from a low of 0.05 m^3/t to a high of 0.4 m^3/t. The average initial CH_4 emission was 0.2 m^3/t.

Following the initial blast data, the CH_4 concentration continued to be measured until the background level for the mine was reached at approximately 3 h after the blast. The average total CH_4 emission was 0.4 m^3/t over the 3-h period following the blast. Table 4 summarizes a portion of the CH_4 emission data collected at the White River Shale Oil Project. The average oil shale grade in the east heading was 125 L/t (30 gal/st).
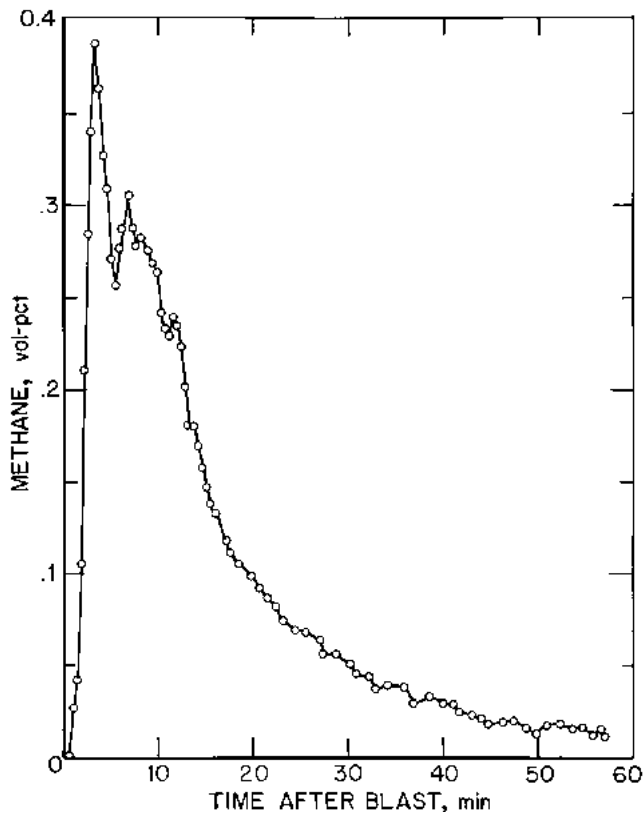
Figure 24.—Typical methane released during blasting at White River Mine as measured underground in the east heading.

Table 4.—Methane liberated during blasting at the White River Shale Oil Project

Blast No.	Initial emission, m^3/t (first 50 min)	Total emission, m^3/t
2	0.37	0.64
4	.22	.60
5	.23	.34
6	.17	.27
9	.21	.23
11	.31	.40
13	.18	.30
15	.16	.17
18	.05	.07
20	.06	.08
33	.19	.27
36	.37	.70
37	.40	.67
40	.27	.54
42	.25	.40
44	.29	.43
46	.34	.48
49	.17	.30
56	.24	.73
59	.35	.53
61	.34	.58
Average	.25	.42
Standard deviation	.10	.20

Methane Emissions From Core Analyses

The amounts of CH_4 released during blasting operations were compared with the CH_4 measured in core samples. The analyses of the oil shale core samples were conducted by Matta and others (50) in 1976. Sections of core were sealed off, and escaping CH_4 was measured as a function of time and extrapolated back to a zero time, thereby obtaining an estimation of the total volume of gas contained in the original core. The total included gas released by the solid core and that released after crushing. This volume divided by the weight of the core gave the specific concentration of CH_4 in m^3/t. Oil assays were also performed on the cores. Figure 25 (from figure 7 of reference 50) shows a plot of gas concentration versus oil assay yield of a core near the White River shaft. The figure summarizes direct-method field CH_4 sampling results for four sections of core taken from the same drill hole in the Uinta Basin. Matta and others (50) indicated that the kerogen contained in the oil shale could absorb 10 times more CH_4 than could be accounted for through its containment in the pore spaces alone. The four core samples in figure 25 were within 6 m of the Mahogany Marker where a drill stem equilibrium pressure of approximately 21 bar was measured at a depth of 330 m. The dashed line in figure 25 represents the maximum quantity of CH_4 that can be contained in shale at 21 bar and varying oil yields; the lower curve represents the measured CH_4 emitted from the core sections under atmospheric conditions. The lower curve would
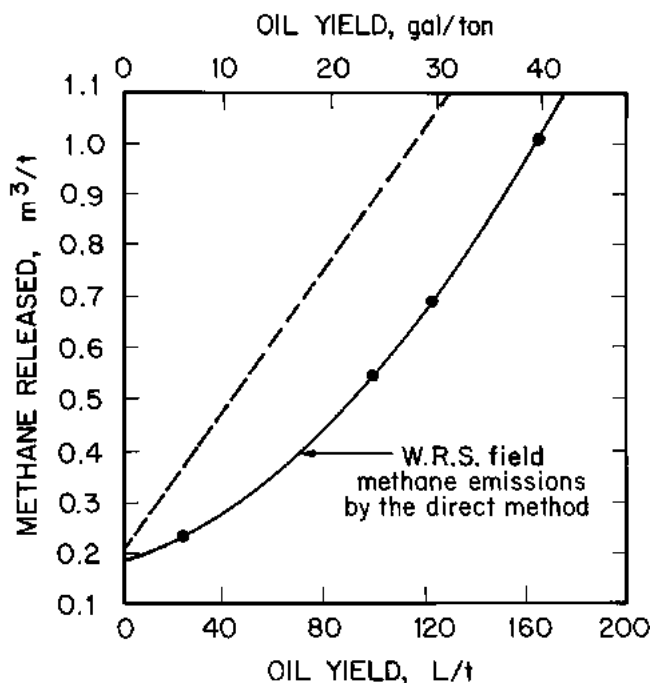
Figure 25.—Methane released from core sections taken near the White River shaft (lower curve). Dashed line shows maximum methane that could be absorbed by the cores.

more nearly represent the CH_4 released during blasting operations. The field CH_4 emissions averaged about 60 pct of the maximum CH_4 that could be absorbed in the core samples. Based on this 1976 core analysis (*50*), the CH_4 emissions from oil shale blasting operations should not have exceeded a release of 0.4 to 0.6 m^3/t for oil shale yielding 83 to 125 L/t (20 to 30 gal/st).

The 1984-85 studies at White River (*53*) measured an average CH_4 release of 0.4 m^3/t over a 3-h period following the

blasting operation. The core analysis work (*50*) predicted a CH_4 release of 0.6 m^3/t for 125 L/t (30 gal/st) oil shale. This variation in amount of CH_4 release may be due to the fact that the core samples were crushed to release all of the CH_4, whereas the blasting process often left large rocks and boulders. To date, very little is known about the initial CH_4 release versus powder factor used in the blast design. Conceptually, the larger the powder factor, the smaller the rubblization and the greater the initial CH_4 release followed by a more rapid dilution to background levels. However, no study has been undertaken in this area.

The CH_4 content of oil shales is one-twentieth or less than that for hvb coals at comparable depths (*53, 55*). However, the CH_4 is usually released continuously in coal mines, whereas it is suddenly released as a pulse during blasting in oil shale mines.

Additional samples were also taken in 1988 from the main exhaust at the idle workings of the White River Shale Oil Project. Based on the measured natural ventilation rate, the background CH_4 emissions ranged from 0.04 to 0.08 m^3/min (*54*). These values were substantially lower than the 0.3 m^3/min background emissions recorded following the mine's closure in 1986 (*53*). Samples collected and analyzed in 1992 revealed that the mine was liberating CH_4 at a rate ranging from 0.02 to 0.05 m^3/min. This continued reduction in the CH_4 yield was encouraging because as future mining operations increase in volume, the ventilation requirements to dilute and remove the CH_4 from the mine workings may not be as formidable.

This research into CH_4 emissions has shown that mining activity in deep oil shale formations will undoubtedly encounter potential gas hazards during the operations. Combining this hazard with the large quantities of dust generated near the face during blasting operations and the hot combustion products exiting from the unstemmed blast holes, a potentially dangerous condition existed for localized CH_4 and/or oil shale dust ignitions.[9]

DEVELOPMENT OF SAFER EXPLOSIVES FOR GASSY OIL SHALE MINES

As part of the research into the hazards associated with the mining and processing of oil shale, hundreds of experimental mine and gallery tests have been conducted to evaluate preferred combinations of explosives and inerting materials to prevent the ignition of oil shale dust and mixtures of oil shale dust and CH_4. This research was jointly sponsored by the USBM, the U.S. Department of Energy (DOE), and the Colorado Mining Association, along with MSHA support. Considerable technical contributions have also been provided by the following explosives manufacturers: Explosives Technologies International, Atlas Powder Co., ICI Explosives Canada, and IRECO, Inc.; and by the following mining companies: Unocal Corp., Exxon Co. U.S.A., and Occidental Oil Shale Corp.

This Disaster Prevention research focused on developing procedures and recommendations for acceptable underground oil shale blasting that maintain a high level of safety for underground workers when operating under gassy mine conditions while fulfilling operational requirements for efficiency. The research was aimed at providing new information, alternative

methods, and innovation in underground blasting procedures. The data directly impact regulatory standards for blasting under gassy mine conditions. MSHA awaited the results of this research program to promulgate blasting regulations at 30 CFR 57.22601 (*25*) for subcategory I-A mines, which include oil shale mines. These regulations, which state that all personnel must be removed to the surface before blasting underground in such mines, were stayed in 1988 and may remain stayed until commercial oil shale mining is resumed.

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[9]On December 7, 1995, a CH_4 ignition occurred in the 9.1-m-diam ventilation shaft of the White River Shale Project during mine sealing operations. One worker was killed and two others were injured as a result of this welding-induced CH_4 ignition.

INITIAL EXPLOSIVES

Four explosives were initially tested: (1) ANFO, (2) an experimental emulsion, (3) a 50:50 emulsion-ANFO blend, and (4) a granular explosive. The ANFO was the formulation commonly used in mining: 94 pct ammonium nitrate and 6 pct fuel oil. The Institute of Makers of Explosives defines an emulsion as an explosive material containing substantial amounts of oxidizers dissolved in water droplets surrounded by an immiscible fuel. The granular explosive—Atlas 8W—is a permissible explosive (25, 56) designed for use in underground coal mines where the presence of flammable gases or combustible dusts presents an abnormal blasting hazard. The key specifications of a permissible explosive are:

• The explosive composition must be within tolerance as determined by MSHA.
• It must pass a series of propagation tests and have an airgap sensitivity of at least 7.6 cm for a 3.2-cm-diam charge.
• The explosive must pass nonignition tests when fired unstemmed into a mixture of CH_4, air, and coal dust.
• It cannot produce over 70.8 L of toxic gases per 0.45 kg of explosive.
• It must exhibit insensitivity in the pendulum friction test.
• It must be used in a permissible manner, i.e., with proper and adequate stemming.

Details on these standard permissibility test methods can be found in reference 56. The permissible explosive was chosen as a reference to compare its relative incendivity with the nonpermissibles in preventing the ignition of predispersed explosible concentrations of oil shale and coal dust.

Tests were conducted to determine the detonation velocities (56) of each explosive. The permissible explosive was initiated with a No. 6 strength electric detonator; the ANFO, emulsion, and the 50:50 emulsion-ANFO blend were initiated with the No. 6 detonator and a PETN booster. These small, rubberlike boosters are a mixture of high explosive and an elastomeric binder that are highly resistent to detonation by mechanical impact. They detonate at approximately 7,315 m/s, generating almost equal priming energy in all directions.

The velocity measurements were made within a 3.7-m-diam sphere (56-57). Each 50-cm-diam charge configuration was 80-cm long with the detonator and booster inserted at one end. An enamel-coated twisted pair of copper wires was wrapped around the charge at 25 cm from the detonator; a second pair was wrapped around the charge at 75 cm from the detonator. Each twisted pair was attached to a counter to record the time required for the detonation wave to travel from the first pair to the second pair. As the detonation wave passed the first twisted pair, the wires were shorted and the counter started. The counter stopped when the detonation wave passed the second set. The detonation rate was then calculated from the length of the explosive column between the wire sets (50 cm) divided by the time measured by the counter. The detonation velocities for the explosives used in the BEM tests were as follows: 5,155 m/s for the emulsion, 3,240 m/s for the emulsion-ANFO blend, 1,000 to 2,000 m/s for the ANFO (confined in a steel cannon), and 2,700 m/s for the permissible explosive. These detonation velocities are considerably lower than the calculated ideal detonation velocities listed in the section

entitled "Low-Incendive Explosives" later in this report.

BRUCETON EXPERIMENTAL MINE TESTS

The purpose of our blasting research program was to determine the relative incendivity of various blasting agents in igniting fine-sized oil shale dust clouds, CH_4, and oil shale and CH_4 mixtures. These incendivity tests (58) on the oil shale dusts and CH_4 were conducted in the main entry of the BEM (figure 3). The tests were conducted near the closed end of the main entry, as shown in figure 26. The oil shale dust was predispersed throughout the test zone before detonation of the explosive. This was accomplished by loading the dust into the seven steel V-troughs. Pressurized air, exiting from holes in the bottom of the troughs, dispersed the dust before detonation of the explosive charge. The dust cloud was confined within the 9.1-m-long test zone by the plastic diaphragm. The mine entry was 1.8 m high by 2.7 m wide, giving a test zone of 44 m^3.

The explosive charges and stemming, when used, were loaded to the back of the steel cannon (57-mm-diam bore). The various charges were rear-primed with an electric detonator and booster (10 g). The cannon was located near the closed end of the entry (figure 26). If the detonated charge produced sufficient flame to ignite the predispersed cloud of oil shale dust and the flame traveled beyond the plastic diaphragm, the result was considered ignition and propagation. However, if the flame was quenched within the dusted zone (inby the diaphragm), the test was considered a nonignition. Typical ignition and nonignition static pressure traces are shown in figure 27. The initial pressure peaks were due to the detonating explosives and varied with the blasting agent. However, if the charge weight was sufficient to ignite the dust, a second pressure rise resulted from the propagating oil shale dust explosion, as shown in the bottom trace of figure 27. The magnitude of this second pressure rise depended on the type and amount of dust and explosive being tested. Instrumentation for these tests included flame sensors, pressure transducers, an infrared pyrometer, optical dust probes, and a high-speed camera.

Minimum Charge Weight for Ignition

The first series of tests (58) focused on determining the minimum amount of explosive (unstemmed) that would ignite Pittsburgh pulverized coal (PPC) dust and various grades of oil shale dust. The PPC, a reference dust, was tested strictly for comparison purposes. In these tests, an explosible concentration of each dust was predispersed from the air dispersion troughs before the detonation of the explosive charge. The blasting agents (charges) were detonated within the 57-mm-diam bore steel cannon, and the combustion products discharged from the cannon into the dust cloud. Three blasting agents (ANFO, the experimental emulsion, and the 50:50

PLAN VIEW OF EXPERIMENTAL MINE
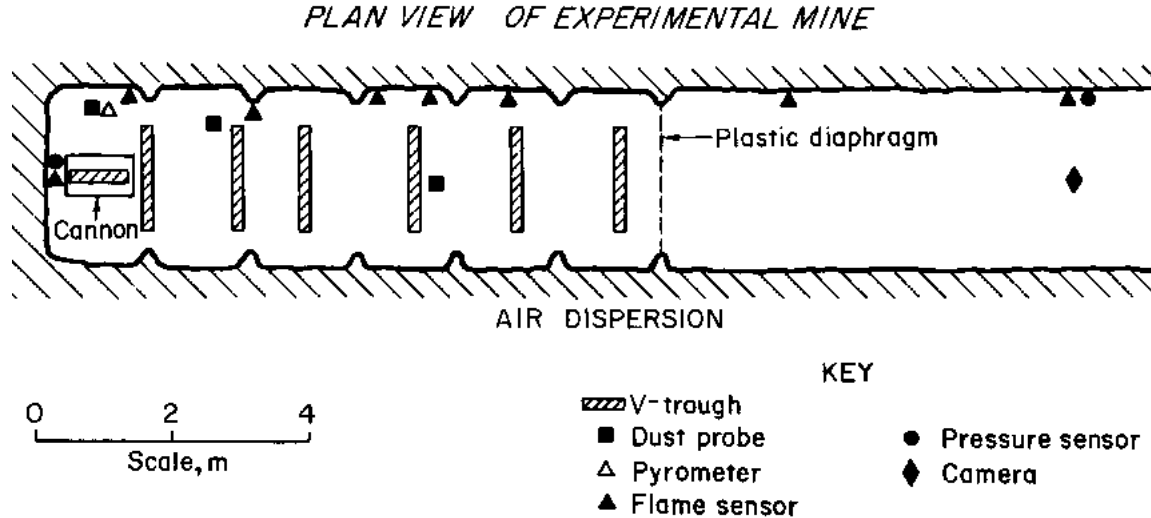
Figure 26.—Plan view of the face area in the main entry of the Bruceton Experimental Mine used for gas and predispersed dust tests.
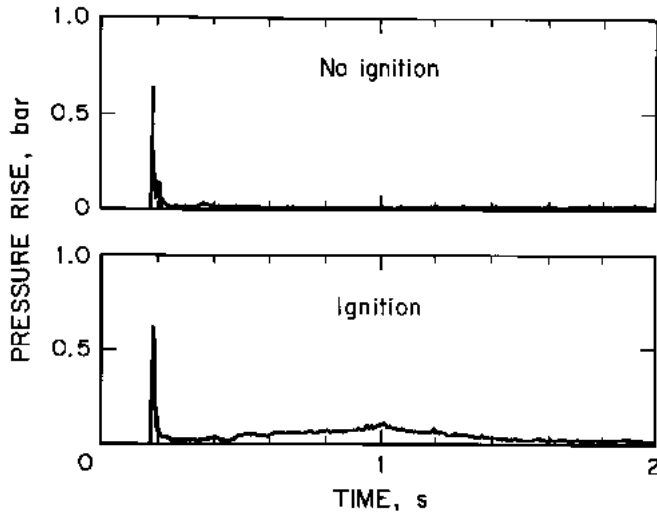
Figure 27.—Static pressure traces for typical mine dust explosions in the Bruceton Experimental Mine using oil shale.

emulsion-ANFO blend) were tested. The Atlas 8W permissible explosive was also tested for comparison data. The results of these tests are summarized in table 5. Note that all of the dusts (oil shale, PPC, and sulfide ore) for these tests were of finer particle size than that collected during production-scale blasting in the oil shale mines, as discussed in the earlier section entitled "Dust, Pressure, and Flame During Blasting in Oil Shale Mines."

In table 5, the ignition test results for various explosive charge weights are listed for the four dusts that were dispersed. For the 140-L/t (34-gal/st) oil shale dust, a 550-g charge of ANFO did not ignite the dust; however, a 600-g charge of ANFO ignited the dust. Neither a 1,050-g charge of the emulsion-ANFO blend nor a 1,000-g charge of the emulsion ignited the same concentration of oil shale dust. When 205-L/t (49-gal/st) oil shale dust was tested, only 400 g of the ANFO was necessary to ignite the dust cloud. However, 1,800-g of the emulsion-ANFO blend failed to

ignite the dust, as did a 1,300-g charge of the permissible explosive. A 1,000-g charge weight of the emulsion-ANFO blend also failed to initiate an explosion, even with 2.5- to 4.0-pct CH_4 dispersed with the 205-L/t oil shale dust. All attempts to ignite predispersed clouds of 205-L/t fine-sized oil shale with the emulsion blend were unsuccessful, even with the added CH_4. The tests were limited to a maximum charge weight of 2,000 g so as not to damage the steel cannon.

Additional tests (58) were conducted to determine the minimum amount of explosive required to ignite a predispersed cloud of PPC dust (table 5). A minimum charge weight of 250 g of ANFO was sufficient to initiate a coal dust explosion. This confirms that the coal dust is easier to ignite than even the rich 205-L/t (49-gal/st) oil shale. A 600-g charge of the permissible explosive ignited the PPC cloud, whereas a 1,200-g charge of the emulsion blend was required to ignite the coal dust. A maximum charge weight of 2,000 g (limit for cannon) of the emulsion did not ignite the PPC.

These tests show that the emulsion-ANFO blend is significantly less incendive than ANFO or even the permissible explosive. Relatively small amounts of ANFO readily ignited the oil shale dust. However, all attempts to ignite the fine-sized, rich oil shale dust clouds failed when using the emulsion-ANFO blend, even in the presence of up to 4.0 pct added CH_4.

Stemming

Another series of tests (58) was conducted in the BEM (figure 26) to determine the effectiveness of stemming materials in preventing a mixture of 8.5 pct CH_4 in air from being ignited by detonating explosives. The 4.6-m section of the main entry was sealed with the plastic diaphragm and injected with 1.7 m^3 of CH_4. When this gas was uniformly mixed within this zone,

as measured by infrared analysis, the explosive was detonated in the steel cannon located near the face (figure 26). Water, limestone rock dust, and ammonium phosphate powder were used as stemming agents. They were evaluated in terms of their ability to suppress the flame of the detonating charge and to prevent ignition of the CH_4. The water and limestone are thermal inhibitors; the ammonium phosphate may be a chemical inhibitor. The explosive charge weights were fixed at 500 g; the stemming lengths were varied to achieve suppression. The results of the stemming effectiveness tests are shown in table 6. Two explosives were tested: ANFO and the emulsion-ANFO blend. Table 6 lists each stemming agent, length and weight, and test result.

Water was evaluated in the form of a plastic tube of water and as a gelled water. A 10.2-cm-long water pack (a self-sealing 4.5-cm-diam plastic tube) was enough to prevent the ignition of the 8.5-pct CH_4-air mixture when detonating the 500-g emulsion-ANFO explosive charge. However, a 15.3- cm-long water pack was required to prevent ignition of the CH_4 by the ANFO. This confirms the results of the previous minimum explosive charge tests, which showed that the ANFO was more incendive. Additional BEM tests with the gelled water units showed that this stemming material was comparable, on a mass basis, with ordinary water packs. These gelled water units primarily consist of a guar gum and salt blended with water to form a gelatin inert material.

Two dry stemming agents were also evaluated. The limestone rock dust was the standard material used for rock dusting in mines. Compared with the water, about eight times (by weight) more rock dust was required to prevent ignition of the

CH_4-air. Ammonium phosphate powder (80 pct minus 200-mesh) was also tested because laboratory and large-scale minetests (59-61) have shown this material to be very effective against CH_4 explosions. The data in table 6 show that a stemming length of 20.3 cm (455 g) of ammonium phosphate powder was required to suppress the flame associated with the detonation of a 500-g charge of the ANFO-emulsion blend. A 30.5-cm length of ammonium phosphate powder was required to prevent ignition when using ANFO. The ammonium phosphate powder required significantly less material than the rock dust to prevent ignition; however, it was still not as effective as water.

Based on these limited results from the BEM, water was the most effective and economical stemming inhibitor for preventing the ignition of a flammable CH_4-air mixture. The BEM tests showed that over 30 pct less water stemming (pack) was needed to prevent ignitions by the emulsion-ANFO blend compared with ANFO. Due to its ease of handling (including its potential for pneumatic loading) and its superior performance on a borehole length basis (due to its increased density over plain water), gelled water was determined in the BEM tests to be the preferred stemming material.

Another type of water stemming device was then tested: a lightweight, yet durable, cylindrical polyethylene plug that was designed to be inserted into a borehole and then filled with water. The water plug was designed to provide an effective coupling to the bore. During each test, the plug was positioned in the 57-mm-diam cannon bore against the 500-g charge of ANFO, then inflated with water until the plug had reached an internal pressure of 1.4 bar. Initial testing of a 15-cm-long water

Table 5.—Minimum charge weights required for ignition in the Bruceton Experimental Mine

Predispersed dust cloud	Explosive charge weight, g, and test result			
	Emulsion	Emulsion-ANFO blend	ANFO	Atlas 8W permissible
Oil shale, 140 L/t			600 I	
@ 350 g/m^3	1,000 NI	1,050 NI	550 NI	NA
Oil shale, 205 L/t			400 I	
@ 250 g/m^3	NA	1,800 NI	350 NI	1,300 NI
PPC		1,200 I	250 I	600 I
@ 200 g/m^3	2,000 NI	1,000 NI	200 NI	500 NI
Sulfide ore				
@ 800 g/m^3	NA	1,000 I	1,000 I	NA

I ignition. NI Nonignition. NA No data available.

Table 6.—Effectiveness of various stemming materials in suppressing ignitions when detonating explosives (500-g charge) into a flammable gas zone

Stemming material	Emulsion-ANFO blend			ANFO		
	Length, cm	Weight, g	Result	Length, cm	Weight, g	Result
Water pack	7.6	120	I	10.2	160	I
	10.2	160	NI	15.3	240	NI
Gelled water	2.5	60	I	10.2	260	I
	5.1	140	NI	12.7	350	NI
Ammonium phosphate . . .	15.3	350	I	25.4	570	I
	20.3	455	NI	30.5	740	NI
Limestone rock dust	25.4	920	I	61.0	2,200	I
	30.5	1,105	NI	66.0	2,400	NI

I Ignition. NI Nonignition.

plug (filled with 350 g of water) showed it to be effective in

preventing the ignition of a flammable concentration of CH_4-air.

To determine the extent to which borehole coupling improves stemming efficiency, several shorter length plugs were tested in the same manner. A plug 11 cm long containing 165 g of water prevented the hot products from the detonating ANFO charge from igniting the flammable CH_4-air atmosphere. Therefore, the water plugs provided a greater degree of protection against the CH_4 ignitions than even the inert gelled water stemming material. The enhanced coupling to the bore effectively contained, for a longer period of time, the hot combustion products within the bore. The result was that these detonation products exited the bore at lower temperatures, thereby requiring less stemming to prevent the CH_4 ignition. Based on the effectiveness of the gelled water stemming materials in preventing the ignition of the CH_4 atmosphere during the experimental tests, several production-scale blasts were monitored at an oil shale mine to evaluate the performance of the gelled water stemming when used during a typical blast loaded with ANFO.

FULL-SCALE FIELD STUDIES WITH ANFO

In September 1988, four full-scale tests (*54*) were conducted at Unocal's Long Ridge oil shale mine in Colorado to compare the relative incendivity levels of different blasting procedures. These tests are listed as blasts 7 through 10 in table 3. The tests were conducted in an area of the mine similar to that shown in figure 11. In two of the tests, nonelectric initiation of unstemmed ANFO was used; in the other two blasts, gelled water was used to stem the ANFO, which was initiated with electric delay detonators. The tests were monitored for air blast overpressures, flame duration, and dust generation. These data were also compared with data collected from the previous unstemmed development rounds (*45-46*).

Instrumentation for these tests served mainly to measure and observe the flame associated with the ANFO face blasts. During the three previous full-face blasts in 1985 (*46*), large amounts of flame were ejected from the boreholes during round detonation. Also evident were large billowing clouds of oil shale dust generated from the face blasts. Previous dust sampling after the blasts (figure 18) had shown that concentrations were an order of magnitude below the experimentally determined lean limit concentrations (*5, 38*) for explosions of fine-sized oil shale dusts of similar grade. However, a small amount of CH_4 (well below the 5-pct lower flammability limit) has been shown to significantly reduce the lean limit dust concentration necessary to produce a localized ignition, which could cause extensive damage to equipment, ventilation ducts, and stoppings in the blast zone area.

During each of the 1988 tests, a flame sensor (silicon photodetector) was positioned approximately 30 m outby the face to measure the relative flame intensity and duration of the hot combustion products exiting from the blast holes. Pressure transducers were used to measure the air blast overpressures. One static pressure transducer was flush mounted to the rib about 10 m inby the last open crosscut (or about 50 m from the face). The second pressure transducer was located 100 to 120 m outby the blast face to measure the decay rate of the pressure pulse as a function of distance. The power supplies, recorder, and timer-delay initiation system were located in the second open crosscut from the face. Three 16-mm movie cameras were located in a direct line of sight 120 m from the face. The cameras filmed at rates of 32, 64, and 100 fr/s. A video camera was also installed adjacent to the other cameras to provide an immediate visual record of each test within minutes after blasting. In contrast to the earlier field tests, no additional illumination of the blast zone was required since the only concern was to determine the incendivity levels (flames) associated with each of the 1988 tests. The flash of the detonating cord and/or any exiting hot gas combustion products from the boreholes would provide the relative incendivity indication for each test.

The mine's routine 36-hole blast pattern for a 8.2-m-high by 12.2-m-wide heading was similar to that (figure 13) used in the previous blasts. The preshear and production holes were each 89 mm in diameter and were drilled to a depth of 7.3 m. Nonelectric time-delayed detonators were typically used at this mine. The preshear and center angled holes were detonated instantaneously to provide the expansion space necessary for the larger production holes. In this mine, preshearing resulted in fairly sound rib lines for long-term stability. Four production holes near the floor were the last in the blasting sequence and acted as lifters to produce a fairly uniform floor. The detonators used during these tests were noted for precise delay periods. The total time of 700 ms was required to complete the initiation of all of the blast holes in the face. Precision delay detonators provided improved fragmentation along with better vibration control by significantly reducing the occurrence of mistimed borehole detonations. The production holes were loaded with ANFO from the back of each hole to within 1.2 to 3 m of the collar. The preshear holes were loaded to within 1 m of the collar with a 45-mm-diam water-gel charge. When stemming was used for blasts 9 and 10, each preshear and production hole was stemmed with five inert water units. These prepackaged units, each 65 mm in diameter and 400 mm long, were slit and tamped into the holes, resulting in an effective cross-sectional stemming plug approximately 1.1 m long.

The relative incendivities associated with nonelectric versus electric initiation and with stemmed versus unstemmed oil shale face blasts using ANFO were compared in these four blasts. The flame sensors and high-speed movie cameras measured the relative flame intensity as a function of the time. For the unstemmed blasts 7 and 8 with nonelectric initiation, approximately 70 ms of flame radiation was detected with the photosensor before its field of view was obscured by the dust generated during the blast. This correlated with the detonation of the preshear and the initial angled holes. Flame existed over the entire blasting sequence, as documented by the cameras, but due to the limited field of view and the dust-blinding factor, only the first 70 ms of flame was recorded by the photosensor. From the 16-mm movie film, visible flame was evident along the lengths of the detonating cord trunklines and was particularly concentrated across the face wherever the cord and nonelectric shock tube connections were knotted. Large amounts of flame and dust were also observed ejecting from the preshear, centered angled, and initial production holes early in the blasting cycle. Over 160 ms of flame was recorded by the cameras compared with 70 ms for the flame sensors due to the camera's larger field of view, which enabled longer observation times before the dust obscured the blast face. Experiments in the BEM have shown that detonating small amounts of unstemmed ANFO or detonating cord (25 gr/ft) will

readily ignite a flammable CH_4-air mixture. In these mine tests, however, no CH_4 was present, and the oil shale dust generated during blasting was insufficient to develop into an explosion that would propagate down the heading. However, a localized dust ignition near the blast face area could occur because of the large volumes of dust generated during the blast. Although the dust sampling has shown that these dust ignitions would probably be limited to the blast zone, significant damage can still result to the mine's ventilation stoppings and other mining equipment.

For blasts 9 and 10 in table 3, no flame or illumination of any kind was detected by either the flame sensors or cameras during the ANFO tests that were stemmed with gelled water and initiated with an all-electric system. This showed much lower incendivity for these blasting procedures compared with blasts 7 and 8.

For the unstemmed blasts 7 and 8, the pressure transducers recorded peak air blast overpressures of approximately 0.3 bar at 50 m from the face (inby the first open crosscut). A transducer 120 m down the heading measured peak overpressures of about 0.15 bar. These values correspond to the instantaneous detonation of the charges in the preshear and initial center production holes and were similar to those measured during previous unstemmed ANFO face blasts (*45-46*). For the tests in which stemming was present, the peak air blast overpressures ranged from 0.15 to 0.21 bar at 50 m from the face (also inby the first open crosscut). At 110 m from the face, the maximum overpressure recorded was about 0.07 bar. The air blast pressure generated during the stemmed ANFO blast was about one-half that of a typical unstemmed ANFO test. A postblast qualitative inspection of the rubble pile showed no significant difference in fragmentation with or without stemming.

Dust samples were collected from measured rock surfaces at several locations within the rubble zone to determine the concentration of fine dust generated during blasting. The minus 20-mesh dust from each sample was weighed, and the data are plotted with data from previous blasts in figure 28. Laboratory and experimental mine tests (*38*) have shown that only the minus 20-mesh dust presents a potential explosion hazard. In figure 28, the floor dust loadings (left ordinate) in grams per square meter are plotted as a function of distance from the face. The nominal volumetric concentrations (right ordinate) in grams per cubic meter were calculated assuming a complete and uniform distribution of the dust throughout the entire cross section of the heading. The concentrations of dust from blasts 7 and 8 showed no significant differences from the earlier tests and were an order of magnitude below the concentrations required for a large-scale explosion. However, a small amount of CH_4 released during blasting in gassy mining conditions could significantly increase the hazard if nonelectric initiation and unstemmed ANFO are used.

Stemming may also serve as a means of further reducing the concentration of the dust clouds generated during the blasting sequence. Based on limited dust surveys after blasts 9 and 10, the dust generated during the blast was less for the stemmed tests (represented by open circles in figure 28) compared with the unstemmed tests (solid circles). Stemming appears to reduce dust generation compared with that normally ejected from the unstemmed boreholes.

LOW-INCENDIVE EXPLOSIVES

The Pittsburgh Research Center worked together with several explosive manufacturers in formulating new, low-incendive, high-energetic blasting agents for use in gassy mining conditions. Three types of prototype emulsion formulations containing different oxidizers were blended with ANFO and evaluated for incendivity (*62*). The oxidizers were ammonium nitrate (NH_4NO_3), calcium nitrate ($Ca(NO_3)_2$), and sodium nitrate ($NaNO_3$). For this discussion, they are represented as AN, CN, and SN, respectively. One type of emulsion contained an AN and SN oxidizer mixture in the base emulsion; the second type contained a mixture of AN and CN; and the third type consisted only of AN. All of the emulsion types were mixed with conventional ANFO. Within each type, the amount of fuel was varied to adjust the oxygen balance of each individual explosive blend in table 7. Properties of the various emulsion-ANFO blends, ICI Breakrite 2 water-gel explosive, and the Atlas 8W permissible are in table 7; properties of the various ANFO formulations are listed in table 8. The ideal hydrodynamic-thermodynamic equilibrium calculations of various detonation properties for each explosive tested are listed in tables 7 and 8. These calculated data were provided by the explosive manufacturers. All of the emulsion-ANFO blends in table 7 were 80 pct emulsion and 20 pct ANFO, except for B, which was a 50:50 blend. The B formulation is the same 50:50 explosive blend discussed earlier in this report in the "Initial Explosives" section. The 80:20 blends were more practical in that they could be conveniently pumped into the borehole; the 50:50 blend was too viscous. The ICI Breakrite 2 is a water-gel explosive that was being used as a low-incendive booster for detonating the emulsion blends. A water gel is defined as an explosive material containing substantial portions of water, oxidizers, and fuel, plus a cross-linking agent. The permissible explosive, Atlas 8W, was chosen as a reference material for comparison with the emulsion-ANFO blasting agents in preventing the ignition of predispersed flammable concentrations of oil shale and coal dust. Detailed explanations of the ideal detonation properties can be found in reference *63*. In tables 7 and 8, the oxygen balance denotes whether the formulation is oxygen-deficient (negative values) or oxygen-rich (positive values). The expansion work is one of the most realistic measures of explosive power because it approximates the amount of work that the gaseous products of the explosion can accomplish as
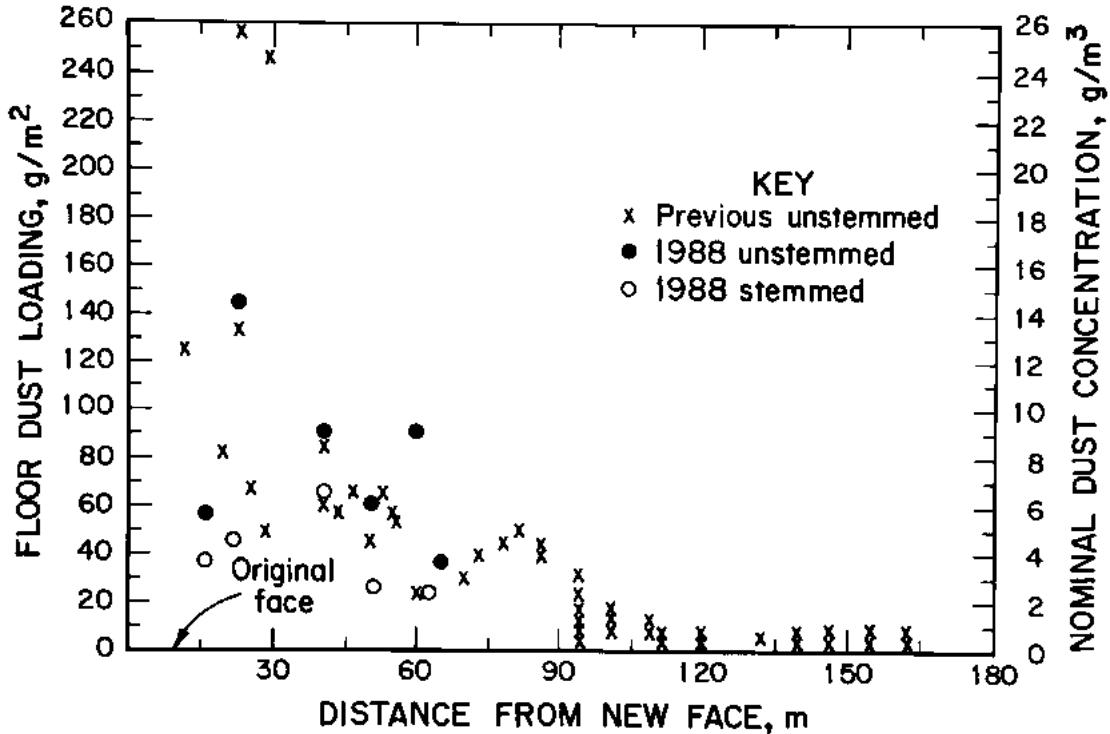
Figure 28.—Floor dust loadings (minus 20-mesh fraction) of unstemmed (solid circles) and stemmed (open circles) ANFO face blasts compared with pre-1988 unstemmed ANFO blast data.

Table 7.—Ideal detonation properties of various emulsion and other explosives

Ideal detonation property	Explosive product				
	A: AN Emul-ANFO	B: AN-SN Emul-ANFO	C: AN-SN Emul-ANFO	D: AN-SN Emul-ANFO	E: Breakrite 2 water gel
Oxygen balance, pct	−9.5	−2.3	−6.0	−2.0	−0.2
Density, g/m^3	1.34	1.28	1.34	1.34	1.13
Expansion work, cal/g	740	799	747	763	785
Reaction temperature, K	2,638	2,830	2,650	2,680	2,720
VOD, m/s	6,310	6,270	6,100	6,100	5,060
Chapman-Jouguet pressure, kbar	110	106	102	100	63
Gas, mol/kg	43.5	44.4	40.5	39.6	38.6
H_2, mol/kg	1.32	1.83	1.17	0.66	0.24
CO, mol/kg	0.64	0.87	0.54	0.29	0.10

Ideal detonation property	Explosive product			
	F: AN Emul-ANFO	G: AN-CN Emul-ANFO	H: AN-CN Emul-ANFO	I: Atlas 8W permissible
Oxygen balance, pct	−1.8	−2.5	−5.9	+7.8
Density, g/m^3	1.26	1.26	1.26	1.19
Expansion work, cal/g	874	612	581	658
Reaction temperature, K	2,080	2,041	2,000	2,043
VOD, m/s	NA	NA	NA	NA
Chapman-Jouguet pressure, kbar	38.7	32.5	32.3	31.7
Gas, mol/kg	44.5	38.9	38.9	37.3
H_2, mol/kg	0.53	0.62	0.87	NA
CO, mol/kg	0.22	0.42	0.64	NA

NA   No data available.

they expand from the initial detonation conditions to atmospheric conditions. The reaction temperature is the calculated maximum adiabatic temperature generated by the detonation of the explosive. The calculated velocity of detonation (VOD) is the speed at which a detonation wave travels through a column of explosive. Note that the calculated values of VOD in table 7 are

considerably higher than the experimental values listed earlier in the "Initial Explosives" section. The Chapman-Jouguet (C-J) pressure is the calculated pressure behind the rear boundary, or C-J plane, of the detonation front. Finally, tables 7 and 8 list the total gas produced per kilogram of explosive, the amount of hydrogen (H_2) produced, and the amount of carbon monoxide (CO) produced.

Table 8.—Ideal detonation properties of various ANFO formulations

	ANFO 96:4	ANFO 95:5	ANFO 94:6	ANFO 92:8
Oxygen balance, pct	+5.4	+1.8	-1.8	-9.1
Density, g/m^3	0.85	0.85	0.85	0.85
Expansion work, cal/g	785	875	906	850
Reaction temperature, K	2,820	3,060	3,140	2,990
VOD, m/s	4,420	4,620	4,760	4,770
Chapman-Jouguet pressure, kbar	42	46	48	49
Gas, mol/kg	43.4	43.3	43.8	45.7
H_2, mol/kg	<0.01	<0.01	0.75	3.63
CO, mol/kg	<0.01	<0.01	0.33	1.75

Two important factors to consider when formulating low-incendive explosives are the oxygen balance and the detonation reaction temperature. As might be expected, a correlation between the explosive's detonation reaction temperature and its incendivity was evident from these limited number of explosive products and will be discussed in the subsequent section entitled "Lake Lynn Cannon Gallery Tests." Some effects of the oxygen balance of the explosive formulation will be discussed here. Table 7 indicates that in the negative oxygen balance cases (excess fuel), there was not enough oxygen to oxidize all of the fuel and to convert all of the CO to carbon dioxide (CO_2); therefore, H_2 and CO were formed. The higher the negative oxygen balance, the more excess hot H_2 and CO would be discharged from the borehole. It is believed that this excess hot fuel, particularly the H_2 with its very high diffusivity, expands from the borehole, mixes with the air, and then burns. Experimentally, the lean flammability limit for H_2 at 25 °C is 4 pct for upward, 6 pct for horizontal, and 8 pct for downward propagation (64-65). Using the Burgess-Wheeler law (42, 64) to extrapolate to higher temperatures, the limits for upward propagation would be 1.9 pct at 500 °C and 0.6 pct at 800 °C. In addition, H_2-air self-ignites at temperatures above 500 °C (43).

The detonation of a stoichiometric ANFO mixture (94.4 pct AN and 5.6 pct FO) can be approximated by the following reaction by using CH_2 to approximate the fuel oil:

$$3NH_4NO_3 + CH_2 \rightarrow 3N_2 + 7H_2O + CO_2. \qquad (A)$$
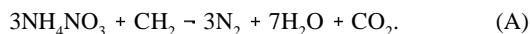
The amount and type of products will change as a function of the FO content. The ideal detonation properties calculated for AN:FO ratios of 96:4, 95:5, 94:6, and 92:8 are listed in table 8. When the amount of FO falls below 5.6 pct, the reaction is oxygen-positive and nitric oxides are produced. By contrast, a negative oxygen balance (excess fuel) tends to generate H_2 and CO (66).

To help understand this potential source of excess fuel (from the explosive) burning outside the cannon, experimental tests were conducted in the Crawshaw-Jones apparatus (56), which is used for determining explosive gas production. The test was performed by detonating 300 g of a given explosive in a steel cannon and collecting the expansion products in a 90-L evacuated chamber. The gases were analyzed by standard gas chromatography. Table 9 lists the gas analyses from the Crawshaw-Jones tests for the ANFO formulations, the emulsion-ANFO blends, and the other explosives. There is relative agreement between the experimental data for ANFO in table 9 and the calculations in table 8 in that the predicted H_2 and CO concentrations increase with increasing FO in both tables, even though the experimental data show higher gas values for each formulation. For all except one of the emulsion blends tested, the data in table 9 show that the blends produced less H_2 and CO than standard ANFO (94:6). The blend that produced more H_2 and CO than standard ANFO was the emulsion blend A, which had a high negative oxygen balance (-9.5). Generally, the more oxygen-balanced blends produced less H_2 and CO than the negative oxygen blends when comparing similar formulations. Likewise, the blends that contained only AN in the base emulsion generally produced more H_2 and CO than those containing SN or CN in the base emulsion when all other factors remained relatively constant. In general, relatively good agreement between the experimental Crawshaw-Jones data (table 9) and the equilibrium calculations (table 7) was shown for the emulsion blends.

Table 9.—Crawshaw-Jones data for various explosive products

Explosive	Gases produced, mol/kg		
	H_2	CO	NO_x
ANFO 96:4	0.13	0.23	0.47
ANFO 95:5	.81	.62	.31
ANFO 94:6	1.45	1.09	.14
Blend A	1.81	1.05	.02
Blend B	.15	.44	1.64
Blend D	.09	.20	.72
Water gel E ..	.78	.41	.04
Blend F	.67	.54	.07
Blend G	.36	.38	.60
Blend H	.80	.70	.26
Permissible I .	1.53	1.54	.04

To study the possible ignition of the excess fuel from the explosives, high-speed photographs were taken of the hot detonation products exiting from a steel cannon (57-mm-diam bore). Results indicated that the time duration of the flame outside the cannon increased with increasing fuel content. Flame durations ranged from 5 to 10 ms for the near oxygen-balanced emulsions to 30 to 40 ms for the fuel-rich formulations. It was believed that this longer lasting envelope of flame was contributing to the incendivity of these formulations. This flame envelope was believed to be associated with the hot combustible fuels, particularly due to the high diffusivity of H_2, which mixed with atmospheric oxygen and burned outside the borehole.

Because of the possibility that prilled ANFO (prill diameter range from 1,100 to 1,600 μm) was less able to sustain a steady detonation in the 57-mm-diam borehole compared with larger diameter holes and thus might be contributing to the incendivity

through incomplete detonation, tests were conducted with a ground ANFO (<840 μm) to obtain a higher bulk density in an attempt to approach more ideal detonation conditions. Incendivity tests (similar to those described earlier in the section entitled "Bruceton Experimental Mine Tests") showed about the same level of incendivity with either prill or ground AN for both 5 and 6 pct FO formulations of ANFO.

LAKE LYNN CANNON GALLERY TESTS

To facilitate the continuing research into the development of the low-incendivity explosives, the testing program was relocated from the BEM to a surface gallery at Lake Lynn Laboratory near Fairchance, Fayette County, PA. The use of the surface gallery eliminated the mandatory delays that were encountered during the mine tests for the removal of the detonation and combustion gases.

The Lake Lynn cannon gallery (LLCG) is a 6.7-m-long by 2.4-m-diam horizontal steel cylinder open at both ends. Mounted inside the LLCG near one end is a 1.5-m long by 0.6-m-diam steel cannon with a 57-mm-diam by 1.2-m-long bore. Figure 29 shows an example of flame exiting from the cannon after detonation of a blasting agent. In the standard LLCG tests, the flame and combustion products from a detonating explosive charge exit the cannon toward the geometric center of the gallery, where combustible dust has been predispersed or an atmosphere of flammable CH_4-air is present. An example of coal dust ignited by unstemmed ANFO in the LLCG is shown in figure 30.

Various explosives have been tested in the LLCG as possible replacements for ANFO, which is an inexpensive blasting agent that is used effectively in many noncoal mining operations. However, ANFO is also a highly incendive product when detonated and has resulted in numerous dust ignitions following blasting operations, particularly in sulfide ore mines in Alaska and Canada. Also, CH_4 is present in some deep oil shale formations (*4, 52-53*), and blasting with ANFO in these mines has resulted in CH_4 ignitions. For these reasons, research has been conducted to develop various replacement explosives and safer blasting procedures for use in these high-risk dust and gas areas. The research has focused on prototype emulsion and water-gel formulations as potential replacements for the ANFO that was typically used in oil shale blasting operations and the dynamite product that was commonly used in sulfide ore mines. The prototype explosive products generally exhibit good fragmentation characteristics and have relatively low incendivity levels compared with those of ANFO and dynamite. The results from these preliminary tests determine which explosives warrant further evaluation in actual full-scale mining operations.

One of the LLCG tests determined the amount of stemming required to prevent the hot combustion products (from the explosive) from igniting a flammable atmosphere of CH_4-air. For this incendivity test with CH_4, a fixed-weight 500-g explosive charge was loaded to the back of the steel cannon. Various amounts of inert gelled water stemming (in 50-g increments) were tamped against the charge. The CH_4 was then injected into

Figure 29.—Flame exiting from cannon after detonation of blasting agent. *(Photo by Kenneth L. Cashdollar, Pittsburgh Research Center.)*

the gallery and was contained between plastic diaphragms secured at both ends of the gallery. A small fan (with explosion-proof motor housing) located inside the gallery mixed the CH_4 with the air to provide a uniform flammable concentration. The CH_4 concentration was continuously monitored by sampling through a small tube extending from the gallery to an infrared CH_4 analyzer. The tests were made at a concentration of about 9.4 pct CH_4-in-air.

For the dust incendivity tests in the LLCG, a 5.0-m-long steel V-trough was positioned on the gallery floor in front of the cannon. Detonating cord was attached to the bottom of the V-trough, and PPC dust was loaded on top of the cord. The initiation of the detonating cord resulted in the dispersal of the dust. The unstemmed charge in the cannon was subsequently initiated after a 500-ms delay to allow sufficient time for the dust to disperse throughout the gallery. The objective of this test was to determine the minimum amount of unstemmed charge that would ignite the dust cloud (figure 30). The standard PPC dust was used during these tests due to its low flammability limit, the ease in which it dispersed, and its availability. This PPC dust has been used for many years as a comparison standard for flammability (*44*). The unstemmed explosive charge weight that would ignite the dust was only a relative indicator of the explosive's incendivity characteristics compared with actual blasting operations. The LLCG incendivity results were conservative because coal dust ignites more readily than oil shale or sulfide ore dusts and there was no borehole expansion work done in the cannon as would occur in rock. However, previous data (described earlier in the section entitled "Bruceton Experimental Mine Tests") had shown that the relative incendivity ranking of various explosive products that were tested against predispersed coal dust maintained their ranking compared with similar tests using predispersed oil shale dust clouds of varying grade or assay.

The explosives were received from the manufacturer in a prepackaged or bulk form. All of the explosives were tested in

Figure 30—Coal dust ignited by unstemmed ANFO at the Lake Lynn cannon gallery showing flame exiting from both ends. *(Photo by Kenneth L. Cashdollar, Pittsburgh Research Center.)*

approximately 50-mm-diam packages and were loaded and tamped firmly to the back of the steel cannon. With the prepackaged explosives, the manufacturer delivered the product already packaged in the 50-mm diameter. The bulk repumpable products were generally delivered in 11-kg units, which, before loading into the cannon, were repackaged in 50-mm-diam polyethylene lay-flat tubing. All of the cap-sensitive charges were initiated with a No. 8 strength copper detonator. In addition to the detonator, a PETN-based booster unit was used with the blasting agents. An incision was made along the length of the stemming package containing the inert gelled water material. The stemming unit was then loaded into the cannon bore and tamped firmly against the explosive charge, which resulted in an effective coupling of the stemming and explosive to the cannon bore. The ANFO was prepared on-site with an approximate loading density of 0.88 g/cm^3. The other explosives were supplied by the manufacturers with densities ranging from 1.25 to 1.34 g/cm^3 for the emulsion-ANFO blends, 1.12 to 1.20 g/cm^3 for the emulsions, 1.3 g/cm^3 for the water gels, and 0.95 g/cm^3 for the semigelatin dynamite product.

The LLCG tests were to be used as one of many relative indicators of an explosive's incendivity characteristics and were not part of any standardized testing schedule for evaluating explosives used during blasting operations in noncoal mines. However, a degree of uniformity and reproducibility was strived for during this testing program. Incendivity tests in the LLCG were conducted only when outside temperatures were above 7 °C. Likewise, reasonable attempts were made to maintain the temperature of the candidate explosive in the range of 7 to 15 °C. This was accomplished by storing the product in an underground explosive magazine. To maintain reliability of the data with limited testing, each data point corresponding to a nonignition

(for both the stemmed explosives detonating into CH_4 and the unstemmed explosives detonating into predispersed PPC dust) was verified by at least three LLCG tests.

Incendivity tests were conducted in the LLCG to determine preferred explosives that would reduce the probability of ignitions during blasting operations under gassy mine conditions. Experiments in the LLCG have shown that the detonation of unstemmed 500-g charges of standard ANFO (94:6) readily ignited flammable concentrations of CH_4 and/or fine oil shale dust. To study this incendivity level, tests were conducted with various ANFO formulations. A 500-g charge of standard ANFO required 350 g of a tamped, gelled water stemming material to prevent the hot combustion products from igniting the surrounding CH_4 mixture in the LLCG. When the FO content of the ANFO was increased to 8 pct, an increased amount of gelled water stemming (400 g total) was required to prevent the CH_4 ignition (table 10). Conversely, when the FO content was decreased to 4 pct, only 250 g of tamped gelled water was necessary to prevent the CH_4 ignition. However, as the incendivity characteristics of the ANFO were reduced by decreasing the FO content, the production of toxic nitric oxides was increased (table 9).

Incendivity tests were also conducted in the LLCG with new and reformulated emulsion-ANFO blasting agents as received from various explosive manufacturers. The pumpable blends consisted of 80 pct emulsion and 20 pct ANFO. Also tested was blend B, which was a 50:50 mix and too viscous for pumping. One technique designed to significant-
ly reduce an explosive's incendivity is through oxygen balancing of the formulation. Comparing the oxygen balance percentages found in table 7 for the emulsion-ANFO blend A (−9.5 pct) with the emulsion blend F (−1.8 pct), an increased amount of gelled water inert stemming was required with the emulsion blend A to

prevent the CH4-air ignition in the LLCG, as listed in table 10. Substitution of materials in the formulation is another technique that can reduce an explosive's incendivity. The blends that contained AN (blends A and F) in the base emulsion were more incendive (table 10) than those that contained CN (blends G and H) or SN (blends B, C, and D), thus requiring more inert stemming to prevent ignition of the CH4 zone under the LLCG test conditions. A 500-g charge of these lower incendive blends required only 100 to 150 g of tamped gelled water stemming to prevent the ignition of a 9.4 pct CH4-in-air zone, compared with 350 g for the standard ANFO. The lower incendive blends, as determined from the LLCG tests, were essentially oxygen-balanced and substituted AN with either SN or CN in the emulsion portion of the blend. In the LLCG tests, the emulsion-ANFO blends were generally the least incendive blasting agents tested because they required the least amount of stemming to prevent ignition of the CH4. Emulsion blend D exhibited the lowest incendivity characteristics of any of the blends and was subsequently field tested in several oil shale mines, as described later in this report.

Table 10.—Gelled water inert stemming weights to prevent methane-air ignitions in the Lake Lynn cannon gallery

	Maximum stemming, g, for ignition	Minimum stemming, g, for nonignition
ANFO 96:4	200	250
ANFO 95:5	300	350
ANFO 94:6	300	350
ANFO 92:8	350	400
Blend A	400	450
Blend B	60	140
Blend C	100	150
Blend D	50	100
Blend F	250	300
Blend G	100	150
Blend H	100	150
Water-gel A[1] . .	250	300
Water-gel B[2] . .	250	300
Emulsion A[3] . .	250	300
Emulsion B[4] . .	200	250
Dynamite[5]	100	150

[1]Cap-sensitive water gel.
[2]Water gel blasting agent.
[3]Emulsion blasting agent.
[4]Cap-sensitive emulsion.
[5]Cap-sensitive semigelatin dynamite.

NOTE.—Explosive charge for each test was 500 g.

Also listed in table 10 are two prototype water-gel-based explosives that were tested in the LLCG. These were formulated to have a low-detonation temperature and to produce a

minimum of hot particles after detonation. These products contained very little aluminum, which not only is a source of hot particles, but can also produce H_2 as a detonation product. Atmospheric air and H_2 can form a flammable mixture, which could extend the duration of flame from the detonating explosives. Table 10 lists the amount of inert gelled water stemming material required to be tamped in front of the explosive charge in the cannon bore to prevent ignition of the CH4 in the LLCG. Water-gel explosives A and B required 300 g of the inert stemming to prevent the CH4 ignition, compared with 100 to 150 g for the emulsion-ANFO blends C, D, G, and H. Standard ANFO (94:6) required 350 g of the gelled water material to prevent the CH4 ignition. The reaction temperatures, as determined from hydrodynamic-thermodynamic equilibrium calculations provided by the explosive manufacturers, were 2,000 to 2,680 K for the emulsion blends (C, D, G, and H) and 2,675 K for the water-gel explosives; these temperatures were significantly lower than the 3,140 K for ANFO. However, the water-gel explosives showed only marginal improvement relative to standard ANFO in preventing the CH4 ignitions, whereas the stemming requirements to prevent the CH4 ignition with the emulsion blends were much lower than those for ANFO. The water-gel explosives contained dissolved AN, whereas the emulsion blends replaced the AN in the 80-pct base emulsion with CN or SN. LLCG incendivity data have shown that the explosive products that contained AN ignited the CH4 more readily than those that consisted of other less incendive materials. The emulsions A and B and the dynamite listed in table 10 will be discussed below.

Table 11 lists the maximum unstemmed charge weight for nonignition and the minimum unstemmed charge weight for ignition of an explosible, predispersed concentration (200 g/m^3) of PPC dust in the LLCG. Coal dust was used during these tests due to its availability. The PPC dust is even more hazardous than oil shale or sulfide dust; it is more easily ignited and requires less dust to form an explosible concentration. Therefore, the LLCG data in table 11 are conservative and are only a relative indicator of an explosive product's incendivity. When detonating unstemmed charges from the steel cannon in the LLCG, all of the hot combustion products were ejected out of the bore and into the CH4 or dust atmosphere. When detonating in ore, much of the hot combustion products are rapidly cooled during the fragmentation process by the rock, which is a natural thermal sink. Standard ANFO detonated unstemmed from the 57-mm-diam bore readily ignited the coal dust cloud in charge weights as low as 100 g. It required 2,000 g of the unstemmed emulsion blend D to ignite the PPC dust; a 1,750-g charge did not ignite the dust. With the water-gel explosives A and B, a maximum cannon charge weight of 2,000 g did not ignite the PPC dust cloud with either explosive. Unstemmed 800-g charges of the emulsion A ignited the dust; 750-g charges did not.

**Table 11.—Relative incendivity of unstemmed explosives in a
flammable coal dust cloud at the Lake Lynn cannon gallery**

Explosive	Minimum charge, g, for ignition	Maximum charge, g, for nonignition
ANFO 96:4	100	NA
Blend D	2,000	1,750
Water gel A[1] . . .	>2,000	2,000
Water gel B[2] . . .	>2,000	2,000
Emulsion A[3] . . .	850	750
Emulsion B[4] . . .	500	400
Dynamite[5]	300	250

NA   No data available.
[1]Cap-sensitive water gel.
[2]Water-gel blasting agent.
[3]Emulsion blasting agent.
[4]Cap-sensitive emulsion.
[5]Cap-sensitive semigelatin dynamite.

NOTE.—Maximum cannon charge weight was 2,000 g.

The two water-gel explosives, the ANFO-emulsion blend D and the emulsion A, showed desirable low-incendive characteristics compared with those of ANFO when detonated unstemmed into an explosible concentration of PPC dust in the LLCG. The water gel A and the emulsion A explosives have also, to date, eliminated the dust ignition hazards associated with blasting operations in sulfide ore mines (as discussed later in the section entitled "Full-Scale Validation Tests in Sulfide Ore Mines"). It is presently unclear why the water-gel explosives exhibited lower incendivity characteristics relative to the coal dust than to the CH_4. It is also uncertain why the dynamite showed low incendivity against CH_4 in table 10, but high incendivity against coal dust in table 11.

In the LLCG tests, a correlation appeared to exist between an explosive's detonation reaction temperature and its incendivity. ANFO (94:6) was chosen as a reference explosive because it was used extensively for blasting operations in oil shale mines and was also used at times for blasting in sulfide ore mines. The ANFO (94:6) required 350 g of inert gelled water stemming to prevent the CH_4 ignition in the LLCG tests. Comparing the minimum stemming requirement (table 10) for each explosive with its ideal detonation reaction temperature (listed in tables 7 and 8), the following correlation can be drawn. Explosives (95:5 ANFO, 94:6 ANFO, 92:8 ANFO, and blend A) that required stemming of 350 g or more to prevent the CH_4 ignition had detonation reaction temperatures ranging from 2,638 to 3,140 K (average of 2,957 K). Explosives (blends B, C, D, F, G, and H; 96:4 ANFO; and water-gel A) that required 300 g or less of inert stemming to prevent the CH_4 ignition had detonation reaction temperatures ranging from 2,000 to 2,830 K (average of 2,478 K). Based on these limited data, an explosive's incendivity can be correlated generally with its reaction temperature. However, many factors, including the water content of the explosive product and the type of oxidizer used, influence this detonation reaction temperature. Of the four most incendive explosives in table 10, the one (blend A) with the lowest reaction temperature had a

large negative oxygen balance, which would make it more incendive than other explosives with similar reaction temperatures. Therefore, the final comparison with incendivity must include both reaction temperature and oxygen balance.

Tests were also conducted in the LLCG to evaluate the explosives' luminosity because the extent of flame outside the cannon was presumably related to the incendivity. In these tests, 500-g charges of various stemmed and unstemmed explosives were loaded into the cannon and detonated, thereby allowing the combustion products to discharge into the gallery. An optical flame sensor was mounted inside the gallery to measure, in terms of duration and intensity, the flame exiting from the cannon bore. With standard ANFO, the detonation of an unstemmed charge produced highly luminous byproducts that were observed for over 15 ms. The unstemmed water gel and ANFO-emulsion blend explosives each produced a 12-ms flame with a 50 pct lower relative luminosity than the ANFO. The explosives were then stemmed with 150 g of a tamped inert gelled water material to evaluate the effect of the stemming on the detonation byproduct luminosity. The gelled water stemming contained the detonation products within the bore for a longer period, resulting in a lowering of the temperatures of these products before exiting from the bore. The detonation of the stemmed ANFO charge still resulted in 15 ms of luminosity; however, the intensity was at a reduced level. The stemmed water gel and ANFO-emulsion blend explosive charges showed no evidence of any luminosity from the detonation byproducts exiting from the cannon bore.

In summary, the LLCG tests evidenced that an explosive's formulation can be altered to result in a product that exhibits low-incendive characteristics relative to its tendency to ignite combustible dusts and/or CH_4. The incendivity of an explosive product depends on many of the factors previously discussed, including oxygen balance, detonation reaction temperature, and type of oxidizer used. Field evaluations with the explosive products that exhibited lower incendivity characteristics were then conducted during production-scale oil shale and base metal mining operations.

FULL-SCALE FIELD TESTS WITH
A LOW-INCENDIVE BLEND

During the next phase of the program, full-scale validation tests (single-hole, multiple-hole, and full-face blasts) were conducted using the emulsion blend D with the inert gelled water stemming in two Colorado oil shale mines in the presence of shale dust and/or CH_4 (67).

Unocal's Long Ridge Mine Tests

Field tests (67) were conducted at Unocal's Long Ridge oil shale mine in August 1989 (blasts 11 through 14 in table 3) to compare the experimental, low-incendive emulsion blend with

the previous ANFO field data from 1988. Instrumentation was similar to that used during the 1988 tests as described in the previous section on "Full-Scale Field Studies With ANFO." These tests were designed to reduce the probability of hot detonation products exiting from the blast holes while still maintaining effective rubblization. The blasting agent used was the ANFO-emulsion blend D, which was 80 pct emulsion and 20 pct ANFO, as listed in tables 7, 10, and 11. Each 76-mm-diam by 7.3-m-deep production hole was loaded with the emulsion blend, stemmed with a 1-m plug of gelled water inert material (approximately 2.7-kg inert per hole), and initiated with an electric millisecond delay detonator. The holes were each primed with a 1.4-kg, 6.4-cm-diam, cap-sensitive water-gel product ("E" in table 7). This product provided for a high-energy, low-incendive booster while offering a much greater degree of protection against CH_4 and/or dust ignitions, particularly when driving crosscuts, than the commonly used, highly incendive PETN boosters.

Unocal's standard preshearing procedure was to use a high-velocity presplit powder loaded almost to the collar, as discussed in the previous section on "Full-Scale Field Studies With ANFO." This was replaced during the 1989 tests with the experimental emulsion-ANFO blend D using an air-cushioning technique. For these tests, each 76-mm-diam by 7.3-m-long preshear hole was primed with the water-gel E booster, and about 4.5 kg of the emulsion-ANFO blend D was then pumped in on top, followed up with a half a cartridge of the water-gel E product. The hole then was stemmed at the collar with three units of tamped, inert gelled water material (1-m plug), which provided a column (approximately 5.5 m long) of air between the charge and the stemming.

As with the stemmed ANFO shots in 1988, no flame was evident outside the boreholes in 1989 after the detonation of the stemmed emulsion blend-loaded rounds (also in the absence of CH_4). Results showed that the rock fragmentation of the emulsion-ANFO blend D in 76-mm-diam holes was at least equivalent to or better than that of the mine's standard ANFO round in 89-mm-diam holes. The air-cushioning technique also showed very encouraging results during these tests in that core borings of the rib line showed only minor fractures 0.3 to 0.5 m deep compared with cracks 3 to 4 m deep into the pillars during this mine's typical blasting operations. The stemming reduced the air blast overpressures in half and also reduced the quantity of dust generated during the blasting sequence.

For both the 1988 and 1989 full-face tests (*62, 67*), the drill patterns, column loadings, and sequencing of detonators were similar to that mine's typical blasting operation described by Baloo in reference *68* and shown in figure 13. The primary differences between the experimental face blasts in 1989 and the mine's typical face blasts were smaller borehole diameters, electric initiation, and the use of the low-incendive blasting agent and stemming in both the preshear and production holes.

In addition to the full-scale performance tests, single-hole charges were detonated and allowed to discharge into a 7-m^3

plastic chamber that contained a flammable CH_4-air atmosphere. These tests were conducted underground at Exxon's Colony Mine and are not listed in table 3. They were designed to evaluate the incendivity of two blasting agents (ANFO and the emulsion-ANFO blend) through the detonating of these charges in the presence of a flammable 10 pct CH_4-air environment. Video and 16-mm high-speed movie cameras were used to document the detonation of the various explosive products within these holes. No flame was observed with the detonation of the emulsion-ANFO blend D into the CH_4-air zone, even with a full, unstemmed column (worst-case scenario) loading in horizontal holes 76 mm in diameter and 7.3 m deep. However, ignition of the CH_4 zone was evident after the initiation of the unstemmed A N F O   c h a r g e   i n   t h e 76-mm-diam holes.

Even though the indications were that a properly stemmed ANFO round produced no flame generation outside the boreholes (based on the 1988 full-face blasts conducted in the absence of CH_4), the potential for a CH_4 and/or dust ignition is greater when using ANFO instead of the emulsion blend (or similar low-incendive blasting agent). If one or more boreholes in the pattern are inadvertently left unstemmed during the loading operations in a gassy mine condition, both the experimental (BEM and LLCG) test results (*57-58, 62*) and the oil shale mine single-hole results (*67*) showed that the unstemmed hole loaded with the emulsion blend should provide a much reduced probability for an ignition than an unstemmed hole loaded with ANFO.

In addition to the hazards related to using standard ANFO charges, a permissible system will be required to initiate the rounds when blasting in gassy mine conditions. BEM tests have shown that the detonating cord typically used to initiate the rounds in the current nongassy oil shale mines will readily ignite flammable concentrations of CH_4. Therefore, a permissible initiation system, such as the all-electric system used in coal mines, will be required when blasting in a gassy oil shale mine.

In June 1990, another test series (*69*) at Unocal's Long Ridge Mine was conducted in headings 5, 6, and 7 of production panel B, as shown in figure 31. These tests are listed as blasts 15 through 17 in table 3. The average cross-sectional area in these headings was about 110 m^2. The drill hole pattern used in each test was similar to Unocal's standard drill hole pattern. All of the holes in each of the three drill patterns were 89 mm in diameter by 7.3 m deep. As part of the drill pattern in headings 6 and 7, six equally spaced preshear holes were drilled along each rib; in heading 5, seven preshear holes were used (figure 32). Each pattern utilized 23 production holes, including three pairs of angled holes that were designed to provide a relief area for the rock during the blasting sequence. Each hole was initiated with a precision delay electric detonator and a cap-sensitive booster (w a t e r - g e l   E   p r o d u c t , 89 mm in diameter by 200 mm long). The emulsion blend D was loaded into the holes using a pneumatic pumping unit

(figure 33). Loading times were initially longer using the emulsion blend compared with those for ANFO; however, as the loading crew gained experience, the loading times improved significantly (2-h loading time for the third test compared with 1.5 h for ANFO loading). The powder factor for each round was adjusted slightly for each test to obtain optimal rubblization based on the geologic structures encountered in the headings. The average powder factor for the three tests was 1.1 kg/m^3 of realized shale. After loading the emulsion blasting agent, each hole was then stemmed with two gelled water units that were each 76 mm in diameter by 400 mm long. This stemming was tamped against the charge column. Typical nonelectric initiation was not utilized during these tests because the detonating cord used in this type of system can ignite a flammable CH_4-air atmosphere, and these tests are designed to evaluate potential blasting agents and procedures for blasting in gassy mine conditions. Therefore, an all-electric initiation system was used.

Instrumentation used to monitor these tests included pressure transducers and optical flame sensors. Both video and high-speed 16-mm cameras (64 and 250 fr/s) were used to document any evidence of flame outside of the boreholes. The cameras were positioned approximately 120 m outby the face. The static pressure transducers were located 45 and 120 m from the face. The optical flame sensor was positioned 25 m from the face. This sensor was located relatively close to the face in an attempt to record any occurrence of flame that may
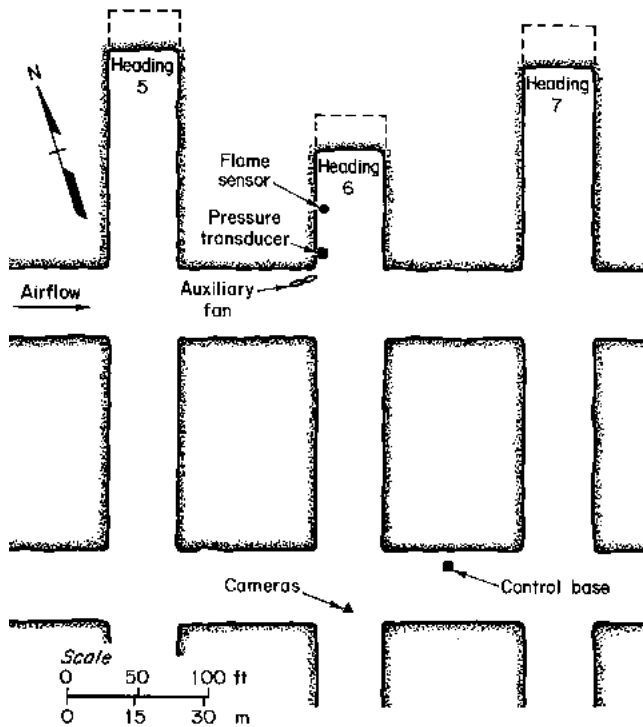
not be documented by the outby cameras due to dust obscuration. The rounds were again initiated by a relay system that was designed to allow the instruments to establish stable baselines and the cameras to achieve full operating speeds before initiation of the round. A portable PC was interfaced with this system for easier and more accurate data gathering and subsequent analyses.

For each of the three tests, the video and 16-mm movie film showed only short duration (<5 ms) detonation flashes associated with several holes. The optical flame sensor supported the camera data in that it recorded very little luminosity
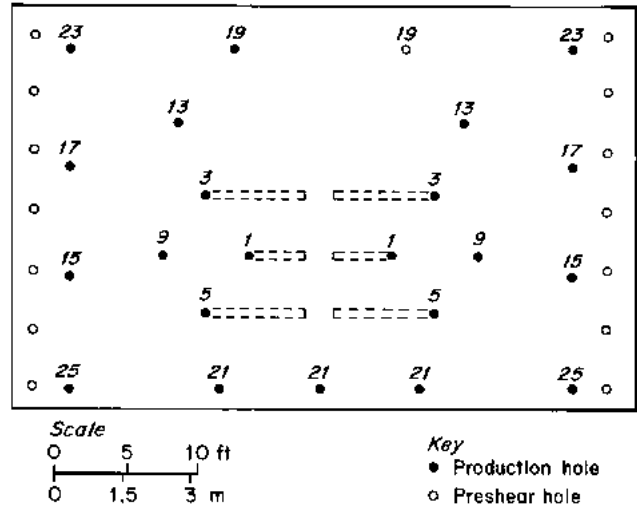

Figure 32.—Diagram of Long Ridge Mine face showing blasting pattern in heading 5 for 1990 tests.

Figure 31.—Map of Long Ridge Mine showing instrumentation locations for full-face blast in heading 6.

Figure 33.—Explosives being loaded into borehole at Long Ridge Mine.

have been generated during the later stages of the round but could

during each test, and that which it did show was very short-lived.

The performance of the emulsion blend in terms of fragmentation and incendivity characteristics during these test blasts was encouraging since there was uncertainty over the size-scaling effect in going to larger diameter holes. In similar full-face tests in 1989 using the emulsion blend D at Unocal, each hole was 76 mm in diameter and stemmed with three gelled water units. In these tests, no flame was observed during blasting. Therefore, in the 1990 tests, only two inert stemming units were used because the 1989 single-hole test data at Exxon's mine showed no evidence of flame even when detonating the emulsion blend D unstemmed into small containment zones of flammable CH_4-air. However, the 1990 tests at Long Ridge Mine and subsequent tests conducted the following week at Logan Wash Mine showed that two gelled water units were insufficient to totally eliminate flame outside the borehole.

The maximum static pressures recorded during the 1990 Long Ridge tests using the stemmed emulsion blend D were consistent with those from the 1989 tests and were only one-half the value measured during the earlier rounds (62) using unstemmed ANFO. One pressure transducer was located inby the last open crosscut approximately 45 m from the face. The pressures ranged from 0.1 to 0.2 bar at this location with the stemmed emulsion blend D compared with 0.3 bar for a typical unstemmed ANFO round. At about 120 m from the face (at the second outby crosscut), the pressure values for the three tests averaged about 0.02 bar. The use of stemming not only cooled the byproducts of detonation, but also allowed for more work on the rock by containing the gas pressures within the borehole for a longer period of time. Another contributing factor to these lower pressure values was the use of the air-cushioning technique on the preshear holes. The advantage of this method was that less explosive was used per hole and the explosive was loaded only at the back of the hole with stemming at the collar.

A postblast analysis was conducted following each round to evaluate the effectiveness of the emulsion blend D compared with the standard ANFO round. The three 1990 tests at Long Ridge Mine using the stemmed emulsion blend D resulted in fragmentation approximately equivalent to that of a typical unstemmed ANFO round loaded in a similar drill pattern. The dust generated during these tests appeared to be less than that of the standard unstemmed ANFO rounds and still at least an order of magnitude lower than that required for a large-scale dust explosion.

Occidental's Logan Wash Mine Test

A full-face test and a multiple-hole spacing test were conducted at Occidental's Logan Wash oil shale mine (15) located northeast of De Beque, Mesa County, CO. This mine consisted of three interconnecting levels and had been primarily used to develop and test an in situ retorting method (1-2). The 1990 tests (69) were conducted in a 25-m-long stub drift located off the main heading approximately 300 m inby the mine portal, as shown in figure 34. This stub drift had a cross-sectional area of about 32 m^2.

One purpose of the tests at Logan Wash was to evaluate the effect of a full-face blast into a flammable CH_4-air zone. To

accomplish this, a wooden framework was mounted around the perimeter of the drift approximately 2.4 m outby the face. A portable foaming unit was then used to seal the gaps between the framework and the mine surfaces to prevent leakage during the CH_4 injection period. Clear plastic sheeting was then attached as a diaphragm to the framework, resulting in a relatively airtight 77-m^3 containment zone. The CH_4 was remotely injected into the zone through two 12-mm-diam tubes from cylinders located about 150 m outby the face near the control base shown in figure 34. A remotely operated fan with a sealed motor housing was used to mix the incoming CH_4 with the air. During the gas injection, the CH_4 concentration was monitored by sampling through a 6-mm-diam tube located inside the containment zone. The sample was drawn through the tube by a vacuum pump to a gas analyzer located at the control base. When the required concentration was reached, the gas injection was stopped.

Instrumentation was similar to that used during the Unocal tests (pressure transducers, cameras, and flame sensors). A pressure transducer and flame sensor were located just inby the main heading and about 22 m from the CH_4 zone at the
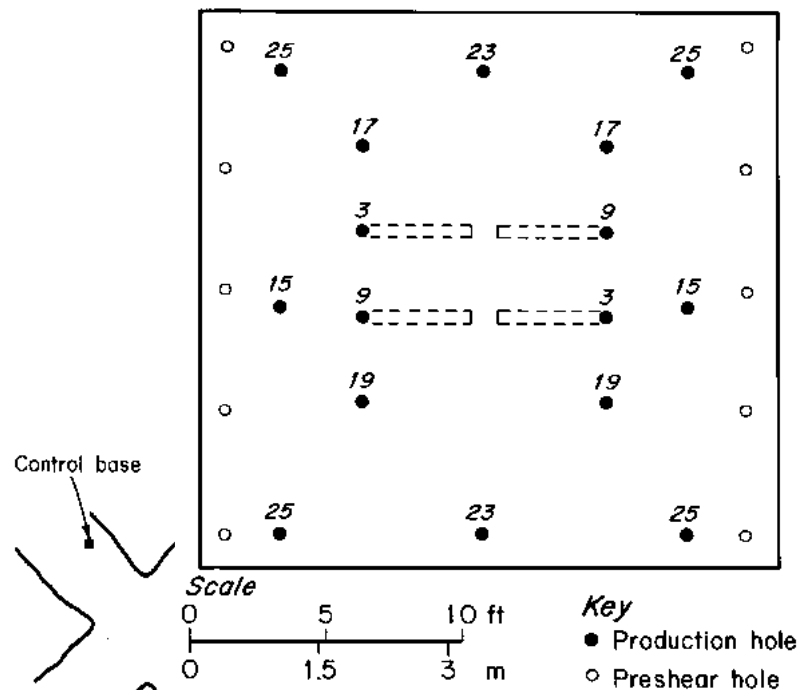

Figure 35.—Diagram showing drill pattern used during Logan Wash face blasts. (Numbers represent detonator delay periods.)

Figure 34.—Maps of Logan Wash Mine showing instrumentation locations for 1990 tests.

face. A video camera and two 16-mm cameras were positioned 28 m from the face. A second pressure transducer was located 150 m outby the face near the control base shown in figure 34. The same computer and shotfire initiation system used at Unocal was used during these tests.

In preparation for the detonation of a full-face round into a flammable CH_4-air zone, a baseline test was conducted with the ignition of the CH_4-air only. Following the CH_4 injection and gas analysis period, the shotfire system was armed and activated. For the baseline test, a small chemical ignitor was used to ignite the flammable CH_4-air mixture near the face. The static pressure transducer located near the face recorded a slow pressure rise that peaked at 0.04 bar overpressure. The transducer located 150 m outby the face recorded a maximum overpressure less than 0.01 bar. The video and 16-mm cameras observed a fireball that began from the lower center ignition point, expanded throughout the containment zone, then traveled outby the drift. The optical flame sensor located nearer to the face recorded about 1.25 s of flame associated with the CH_4 burning.

After the baseline test, a full-face round was detonated into a CH_4-air zone using the emulsion blend D in the boreholes. The drill pattern for the full-face test, shown in figure 35, consisted of twenty-six 89-mm-diam holes drilled to an average depth of 3.7 m. The face along each rib line had five equally spaced preshear holes. Two pairs of angled holes were drilled in the middle of the face to provide an expansion relief zone to facilitate the detonation of the later holes. Each hole was primed with the water-gel E booster and a precision delay electric detonator. The 16 production holes were then loaded with the emulsion blend D, and two stemming units were tamped into each hole against the charge column. The powder factor for the face test was 2.4 kg/m^3 of realized shale. The powder factor was substantially greater than the 1.1 kg/m^3 powder factor used for the Unocal headings. This high powder factor would present a higher CH_4 ignition risk. The air-cushioning technique was used for the 10 preshear holes, which were loaded at the back of each hole with 22 kg of the emulsion blend D. These holes were then stemmed at the collar with two units of the inert gelled water material.

After loading the explosives, another plastic sheet was attached to the wooden framework of the 77-m^3 gas containment zone. The CH_4 was then injected into the containment zone and mixed with the fan until the gas analyzer showed the concentration to be 10 pct CH_4-air, a worst-case scenario. Gas samples were also taken for subsequent laboratory analyses; the results were consistent with the on-line analyzer readings. The round was initiated with a permissible system consisting of electric detonators.

A maximum static overpressure of about 0.9 bar was recorded on the transducer located 22 m from the face and inby the main heading. The second transducer located 150 m outby the face recorded maximum overpressures less than 0.01 bar. Typical blasting overpressures with ANFO in this size drift from previous operations within the Logan Wash Mine were not available. However, the overpressures obtained during this test did not appear to be unreasonable considering the high powder factor and the small cross-sectional area of the drift. Previous tests (*45-46, 54*) at the Long Ridge and Colony Mines have generated overpressures ranging from 0.1 to 0.3 bar when the pressure transducers were located at similar distances from the face. However, the Long Ridge and Colony Mine headings had nearly four times the cross-sectional area of the Logan Wash Mine drift. Experimental results (*70*) have shown that doubling the area will reduce the overpressure values by one-half. This, coupled with the higher powder factor at Logan Wash, would account for the higher overpressures recorded during these tests. The high powder factor resulted in relatively uniform fragmentation of the shale in much reduced sizes than that obtained during the Long Ridge tests.

Data from the flame sensor and cameras indicated with reasonable certainty that an ignition of the CH_4-air mixture occurred as a result of the detonation of the full-face pattern stemmed with two inert gelled water units; at least, a bright flash of light occurred at the start of the round. However, because of the short distance from the face to the T-intersection at the main heading where the cameras and flame sensor were located, the overpressures and debris generated during the detonation of the round damaged the instruments and cameras shortly after initiation of the round. Several contributing factors may have enhanced the probability of an ignition. One factor would be the high powder factor, which was over twice that normally used in typical operations. Another change from past tests was the use of larger diameter holes. It would appear that the 89-mm-diam holes require at least as much or more stemming compared with the 76-mm-diam holes to prevent the hot detonation gases from exiting from the hole and igniting the CH_4. The spacings of the drill holes in the face pattern may also have been a major contributor to the CH_4 ignition. An explosive loaded in a hole can be desensitized by the detonation of an adjacent hole should that hole be too close. The result is that the desensitized charge will not detonate properly as evidenced by its lower velocities and may deflagrate rather than detonate. Separate tests to evaluate the spacing of blast holes are discussed below.

A multiple-hole test (*69*) was conducted at Logan Wash Mine after the full-face test to measure the detonation velocities of loaded holes spaced at various increments from one another. As part of the field evaluation with the experimental, low-incendive emulsion blend D, this test was designed to evaluate the performance of the explosive and determine the minimum spacings required between blast holes. The minimum spacing requirement is critical in preventing potential misfires because the detonation of the adjacent short delay rounds can desensitize the explosive in the longer delay holes. Desensitization can occur when the sensitizing agent in the explosive, typically glass microbubbles, is crushed from the shock waves produced from the detonation of adjacent holes. The concentration of micro-bubbles in a formulation determines the explosive's sensitivity and the strength of the initiator required to detonate the explosive.

The spacing test entailed eight holes drilled horizontally into two opposite oil shale ribs (four holes per rib). There was a distance of 2.1, 1.5, and 1.0 m between the holes in each set; each hole was 6 m deep and 89 mm in diameter. The

emulsion blend D was bulk loaded in the back 4.6 m of each hole and stemmed with a gelled water inert material to maximize the shock pressures to the surrounding strata. A velocity-of-detonation recorder (VODR) was used to measure the detonation velocities in each hole. This method of recording the detonation velocity within an explosive column can also be referred to as the resistance wire method (*71*). Constant current is supplied to two insulated conductors, such as the center wire and shield of the coaxial cable used during these experiments. A continuous length of coaxial cable was looped into each of the eight holes and then connected to the VODR. At the far end of the coaxial cable, which was located at the back of the first hole to be detonated, the two conductors were shorted. As the detonation progressed through the explosive column, the coaxial cable conductors were forced into contact as the cable was crushed. This shorting reduced the circuit voltage since this action reduced the length of the electrical path and therefore the resistance of the probe. Recording the change in voltage over time provided the detonation rate. A sequential blaster machine initiated the holes at 10-ms intervals. Results from this first test indicated that a spacing of 2.1 m between holes had no detrimental effect on the performance of the emulsion blend in the oil shale formation. However, the holes on 1.5-m or closer spacings did not detonate properly, as evidenced by the lower velocities. Many of the holes in the Logan Wash full-face drill pattern (figure 35) were spaced less than 2 m apart. It was quite possible that the CH_4 ignition during the full-face test may have been the result of the improper performance of the explosive in some of the holes. Additional spacing tests need to be conducted in oil shale to verify these preliminary findings. Future full-face tests using the emulsion blend D should be conducted with hole spacings of at least 2 m.

EXPLOSION HAZARDS OF SULFIDE ORE DUSTS

The Pittsburgh Research Center, at the request of the MSHA, expanded its research program on developing safer explosives and blasting procedures for gassy oil shale mine conditions to include sulfide ore mines. Working in cooperation with several explosives manufacturers, the Pittsburgh Research Center continued to test various explosive formulations, including water-gel and emulsion products, in the LLCG to evaluate their relative incendivity compared with standard ANFO and other explosives.

There were some very early reports of sulfide mine dust explosions in the 1920's, as described in references *72* and *73*. In recent decades, there has been a significant increase in the number and severity of blast-induced sulfide dust ignitions in the underground workings of massive sulfide deposits in Europe, the Republic of South Africa, Australia, Canada, and, more recently, the United States. These recent sulfide dust explosions have been summarized in a literature survey by Job (*74*) and in several papers by Enright (*75-77*). Another literature survey on spontaneous combustion and sulfide mine fires was presented by Ninteman (*78*). The explosives normally used in sulfide ore mining were dynamites and/or ANFO. Most of these dust ignitions have occurred in open stopes where large-diameter drill holes were used with bench blasting or vertical retreat mining.

However, a significant number of dust ignitions have been associated with development headings and the secondary blasting of ore at draw points. There are numerous documented accidents relating to the various hazards associated with the mining and processing of these ores (*79-82*). Fortunately, the number of fatalities associated with these ignitions has been minimal due to the practice of evacuating mines before stope blasts. However, many operators report extensive damage during many of the dust ignitions. Even with minor dust ignitions, there was appreciable lost production while a mine was checked for toxic levels of sulfur dioxide and hydrogen sulfide and the damage to the ventilation system was repaired to safely purge the mine of these toxic gases. However, dust ignitions have caused extensive damage to ventilation ducts and stoppings and also to the mining equipment, thereby creating potentially hazardous situations for the miners (*81*). The intense heat and flame associated with these explosions can ignite timber sets and other combustible materials in the area. The destructive pressures generated from a sulfide dust ignition are generally limited to the blast face area and, in most cases, do not extend significant distances into other areas of the mine. This is due primarily to the significant decrease in the concentration of entrained sulfide ore dust outby the immediate

blast area. Reddish dust or reddish stains of iron oxide (Fe_2O_3) on rock or equipment can be used to track the extent of flame propagation. Magnetite (Fe_3O_4), although black, is also sometimes produced during sulfide dust ignitions and is also useful in detecting the extent of the explosion.

In 1986, Wheeland and McKinnon summarized the sulfide dust explosion problem and current mining practices at the time (*83*). One critical factor affecting sulfide dust ignitions is the sulfur content of the ore; the ore with the higher content is more explosible in that it requires less mass suspended in the air to propagate an explosion (*76-77, 83-86*). In sulfide ores, higher sulfur content is often associated with higher pyrite (FeS_2) content because pyrite is the most common host sulfide and has the highest sulfur content (53 pct) of the common sulfide ores (*84*). Pyrrhotite (~FeS) is also a reactive host sulfide even though it has a lower sulfur content than pyrite.

LABORATORY DATA

The laboratory explosibility data for the sulfide ore dusts were measured in the 20-L chamber (*34-36*) shown in figure 5. The test procedures were identical to those used for the oil shale dust explosibility measurements described in the "Explosion Hazards of Oil Shale Dusts" section earlier in this report.

The chemical assays and size analyses of the sulfide ores are listed in table 12. The sulfide ores are identified by date (month/year) received. One sulfide ore was identified by the mine as being high in pyrite (FeS_2) content and is so noted. The table then lists the location of the sulfide ore mine from which each sample was obtained. The first four samples were from Brunswick Mining and Smelting Corp. (a division of Noranda) in Bathurst, New Brunswick, Canada. The 1983 samples were received as rocks and were crushed and pulverized at the Pittsburgh Research Center. The 1985 and 1987 samples were from a dust collector at the processing plant in New Brunswick, Canada. In the table, the two 1989 samples from Greens Creek Mine in Alaska are differentiated by an "s" for the standard sulfide ore and a "p" for the sample that was higher in pyrite content. The sulfur content and metals analyses are then listed in the table. These sulfide ores were mined mainly for zinc and lead content. The heating value was measured in a bomb calorimeter. Note that the ores that have the higher sulfur contents have the higher heating values. The particle size analyses shown in table 12 are in a form similar to those used in table 1. The 1983 dusts are significantly larger in size than those from 1985, 1987, and 1989. The dusts were stored under nitrogen.

Table 12.—Chemical and size data for sulfide ores

Data	Sulfide ore dusts					
	12/83-py[1]	12/83	7/85	4/87	8/89-p	8/89-s
Location	New Brunswick, Canada	New Brunswick, Canada	New Brunswick, Canada	New Brunswick, Canada	Alaska	Alaska
S, pct	~40.0	33.5	27.0	28.0	~34.5	24.5
Si, pct	2.6	4.9	7.4	7.6	0.6	1.0
Ca, pct	0.5	2.3	3.8	3.4	1.8	2.6
Fe, pct	39.0	24.0	23.1	23.5	25.5	4.8
Cu, pct	0.5	0.3	0.3	0.5	0.3	0.3
Zn, pct	1.3	13.8	10.8	11.0	18.6	32.5
Pb, pct	0.4	4.7	3.8	3.6	3.8	24.4
Heating value, cal/g	1,300	1,000	790	840	1,100	770
Minus 200-mesh, pct . . .	79	82	96	99	99	98
D_{med}, μm	34	23	13	13	12	12
$\bar{D}_W$, μm	44	36	23	17	19	20
$\bar{D}_S$, μm	20	16	10	10	9	10
$\bar{D}_S$(%T), μm	16-50	15-35	8-18	8-20	6-11	8-12

[1]Identified by mine as mainly pyrite.

The laboratory data for the explosibility of the sulfide ore dusts are shown in figures 36-38 in forms similar to those of the oil shales depicted in figures 6-7. The data were obtained in the 20-L chamber using the 5,000-J chemical ignitors. The data show that some sulfide ore dusts can propagate an explosion if the particle size is fine enough and the ignition source is strong enough. The data in figure 36 show that the pyritic sample ignites at lower dust concentrations and produces higher pressures and rates of pressure rise than the regular sulfide ore sample from 1983. The two samples in figure 37 show somewhat similar characteristics, as expected, because they both came from a dust collector at the processing plant. The 7/85 sample was tested twice. The first time was shortly after it was received, and the data points are denoted by the filled circles. The second time was 2 years later (open circle data points), and the tests produced only a slight pressure rise that would not be considered a propagation. This lower explosibility of the sulfide ore may be due to some possible oxidation of the sample over time. Although there was an attempt to keep the sample under nitrogen, there may have been an air leak into the storage jar.
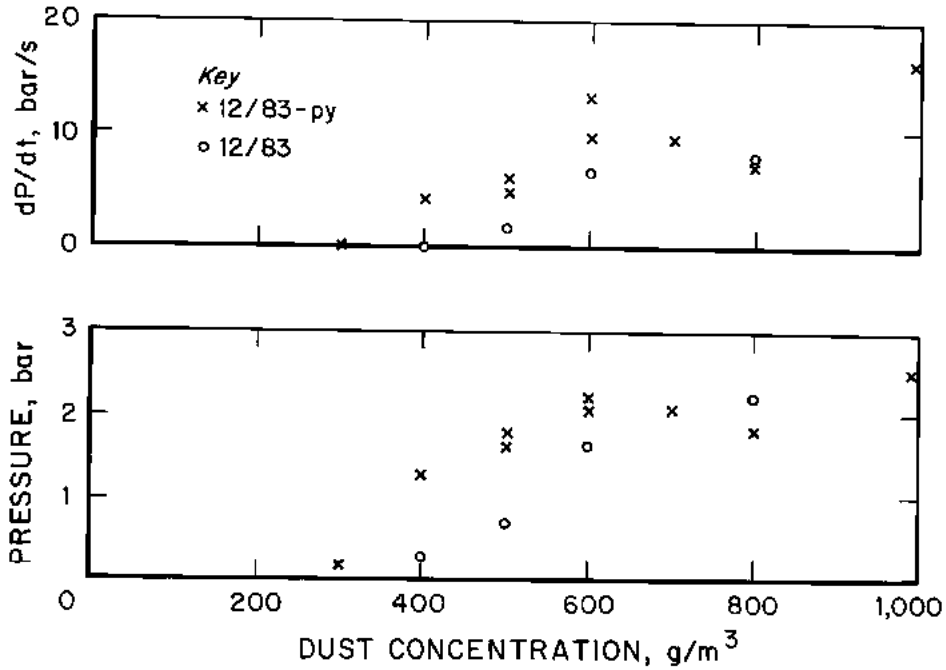

Figure 36.—Explosibility data for 1983 pyritic and sulfide ore dusts from the 20-L chamber.
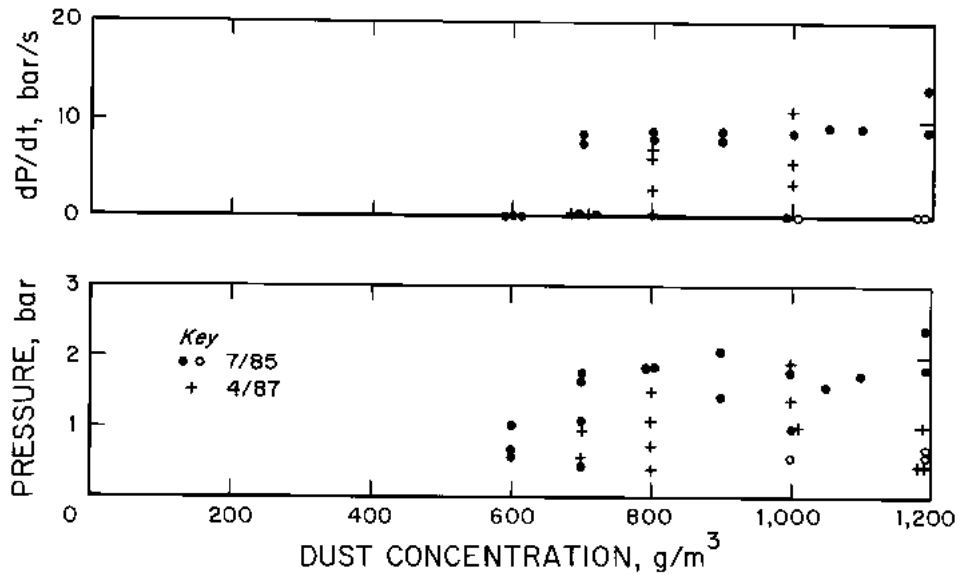

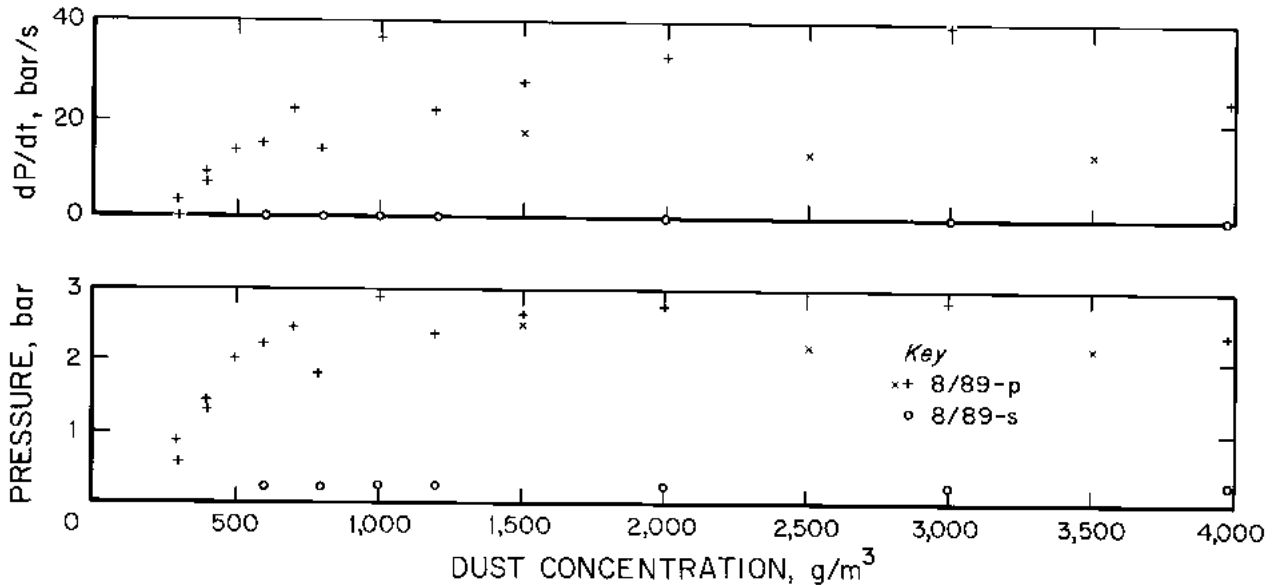**Figure 37.—Explosibility data for 1985 and 1987 sulfide ore dusts from the 20-L chamber.**

Figure 38.—Explosibility data for 1989 pyritic and sulfide ore dusts from the 20-L chamber.

The pyritic sample from 1989 (figure 38) showed the highest pressures and rates of pressure rise of all of the sulfide ore samples. It was significantly finer in size than the other two ores that were high in sulfur content (12/83 and 12/83-py). It also had a higher sulfur content than the other fine-sized dusts from 1985, 1987, and 1989. This combination of finer size and higher sulfur content probably caused its higher explosibility. The standard sulfide ore (8/89-s) from Alaska shown in figure 38 could not be ignited, even with the 5,000-J ignitors. This sulfide ore had the lowest sulfur content of all of the ores tested.

The sulfide ore explosibility and ignitability data are compared with the coal dusts in table 13 in a form similar to that for the oil shales in table 2. Even the five sulfide ore samples that ignited in the 20-L tests had much higher MEC values and lower P_{max} and $(dP/dt)_{max}$ values than those of the two bituminous coals. The explosibility data are closest to the 82- and 95-L/t (19- and 23-gal/st) oil shale data in table 2. The 8/89-s sulfide ore could not be ignited by the 5,000-J ignitor, similar to the anthracite coal. The 8/89-s sulfide ore produced an explosion in a single test at 4,500 g/m³ using a 10,000-J ignitor, although it is uncertain whether that ignitor would overdrive the 20-L chamber. Based on the data in the table, one can conclude that the sulfide ore is a weak explosion hazard, but that it can be ignited if the particle size is fine enough, the sulfur content is high enough, and the ignition energy is strong enough.

The maximum pressure data for the sulfide ores in table 13 are consistent with those of other researchers. Enright (76-77) measured somewhat higher pressures, but he used a much stronger ignition source (1.5 g of nitrocellulose or 15 kJ) in a 36-L chamber. The particle size of his sulfide dust samples

was not listed. Mintz and Dainty (84) measured pressures of 2.2 to 2.6 bar for pyrite and pyrrhotite in a 20-L chamber using a 5,000-J ignitor, similar to the values for the higher sulfur content ores in table 13. Liu (85) and Liu and Katsabanis (86) conducted experiments in a 1-m³ (1,000-L) chamber using an explosive charge as the ignitor. One of the samples (listed as type I from Brunswick Mining and Smelting Corp.) that they tested was a sulfide ore similar in size to the 7/85 and 4/87 ores shown in tables 12 and 13, but with slightly higher sulfur content. Their type I sample had P_{max} = 3.0 bar. This is higher than the P_{max} value for the 7/85 and 4/87 ores, but they used a much stronger ignition source (15 g of Detasheet explosive, or 72 kJ). The other sample (type II) that they tested was pyrite ore with P_{max} = 3.7 bar. Soundararajan (87) and Soundararajan, Amyotte, and Pegg (88) measured the explosibility of sulfides in a 20-L chamber using a 5-kJ ignitor. They measured P_{max} = 3.1 bar for a pyrite dust with 46 pct sulfur. The dP/dt values from the various researchers cannot be compared directly with the data in table 13 because of possibly different turbulence levels in the various chambers.

Enright (76-77) and Liu and Katsabanis (86) concluded that ~20 pct was the minimum sulfur content for sulfide ore explosibility. In the Pittsburgh Research Center 20-L laboratory tests, >26 pct sulfur content was required for an explosion using 5,000-J ignitors. The difference is probably related to the stronger ignition sources used by Enright and by Liu and Katsabanis. For sulfur contents between 20 and 40 pct, Liu and Katsabanis reported that explosion violence increased significantly with sulfur content. There is a general correlation in the Pittsburgh Research Center data (table 13), but it is complicated by the variation in particle size of the dusts tested (table 12).

Table 13.—Laboratory explosibility data for sulfide ores and comparison dusts

Explosibility data	Sulfide ore dusts						Coal dusts		
	12/83-py	12/83	7/85	4/87	8/89-p	8/89-s	hvb	lvb	anthra
MEC, kg/m^3									
@5 kJ	~0.4	~0.6	~0.7	~0.8	~0.4	NI	0.06	0.09	NI
P_{max}, bar	~2.3	~2	2.0	~1.4	2.8	0.3	5.6	5.0	0.4
$(dP/dt)_{max}$, bar/s . . .	~13	~7	10	~7	36	<1	145	77	<1
MAIT, °C	~490	~520	—	~530	—	—	530	~610	~670

NI Nonignitable.

NOTE.—A dash means not analyzed.

The MAIT values (490 to 530 °C) for three of the sulfide ores are listed in table 13. These values are comparable with those of the hvb coal and the oil shales (table 2), but are lower than those of the lvb coal and the anthracite. Mintz and Dainty (*84*) measured a similar MAIT of 510 °C for pyrite. Soundararajan (*87*) measured a MAIT of 480 °C for his finest sized pyrite dust and higher MAIT values for larger sizes.

In the bituminous coal mining industry, limestone rock dust is added to the deposited float coal dust to make it nonexplosible. Bags of rock dust suspended near the face have also been considered as a means to prevent sulfide ore dust ignitions during a blast. Figure 39 shows data from the 20-L chamber for a mixture of 20 pct limestone rock dust and 80 pct of the 7/85 sulfide ore. The slight pressure rises shown in the lower part of the figure were only due to a small amount of burning in and around the ignitor flame. The rate of pressure rise data show that there was no sustained propagation beyond the ignitor, and therefore these mixtures would be considered nonignitable. For Pittsburgh coal dust of similar size, about 75 pct rock dust in the coal-rock mixture was required to inert for the same test conditions (*35-36*). In similar tests, Enright (*76*) found that more than 40 pct limestone was required to inert pyrite ore.

LARGE-SCALE EXPERIMENTAL MINE DATA

The large-scale flammability tests on the sulfide ore dust were conducted in the main entry of the BEM (figure 3). The predispersed tests were conducted near the face (figures 4 and 26) in a manner similar to the predispersed oil shale tests. The test zone was 9.1 m long and had a volume of 44 m^3. For each test in this limited series, the sulfide ore dust (7/85 and 4/87 from table 12) was loaded into seven V-troughs and/or on three suspended roof shelves. Pressurized air, exiting from holes in the bottom of the V-troughs, dispersed the dust within the 44-m^3 test zone. The dust on the shelves would be dispersed by the pressure pulse from the detonating ANFO in the cannon. The ignition source for the sulfide ore tests consisted of a 644-g charge of 2.5 pct aluminized ANFO. This charge was detonated from the cannon using an electric blasting cap and two tetryl boosters (3 g each). The criterion for ignition and propagation of the sulfide dust was that flame be observed beyond the test zone, i.e., 9 m from the face or 8 m from the cannon.

A series of six tests was performed in the BEM to determine the minimum concentration for propagating explosions of the sulfide ore dust. The test series was ended when the limited quantity of sulfide dust was exhausted. The results of these tests are listed in table 14. The table indicates the BEM test number, followed by the nominal dispersed dust concentration, the loading distribution of the dust between the V-troughs and the shelves, the flame travel distance (measured from the cannon), and maximum pressure rise. The pressure was measured after the decay (>100 ms) of the pressure peak from the detonating ANFO. Therefore, this listed pressure would be that due to any propagating sulfide dust explosion.

In a baseline experiment where only the cannon ignitor charge was detonated, the maximum static pressure was approximately 800 mbar with a flame extension of about 3 m from the cannon bore or 4 m from the face. In the initial test (No. 4202) with the sulfide dust, a nominal concentration of 800 g/m^3 was uniformly loaded into the seven V-troughs. The dust was then dispersed with the compressed air system, and the ANFO was detonated about 1 s later. Flame was observed out to 7 m from the cannon, but it did not go beyond the dusted zone. Upon inspection of the zone after the test, significant amounts of sulfide dust were beneath the troughs, indicating inadequate dispersion. However, an iron oxide residue covered the entire test zone, which evidenced considerable burning of the sulfide dust. In the second test (No. 4204) with an 800 g/m^3 concentration, 38 pct of the dust was suspended above the troughs on three roof shelves. The remainder of the dust was loaded into the air dispersion troughs. In this test, the flame traveled 14 m from the cannon (6 m beyond the diaphragm) with a pressure rise of 90 mbar. This test was considered to be a propagating sulfide ore dust explosion, even though the pressure was rather low. A more uniform entrainment of the dust occurred when the roof shelves were used and the dust loading in the air troughs did not exceed 500 g/m^3. This was determined by the concentration measurements of the three optical dust probes in the test zone and from a visual inspection of the zone after the test. In test No. 4203, all of the dust was loaded in the V-troughs, but the flame only went 3 m beyond the cannon; this test showed evidence of poor dispersion.

Additional tests were conducted with dust both on the shelves and in the troughs to determine the lowest concentration that would propagate an explosion. In two tests
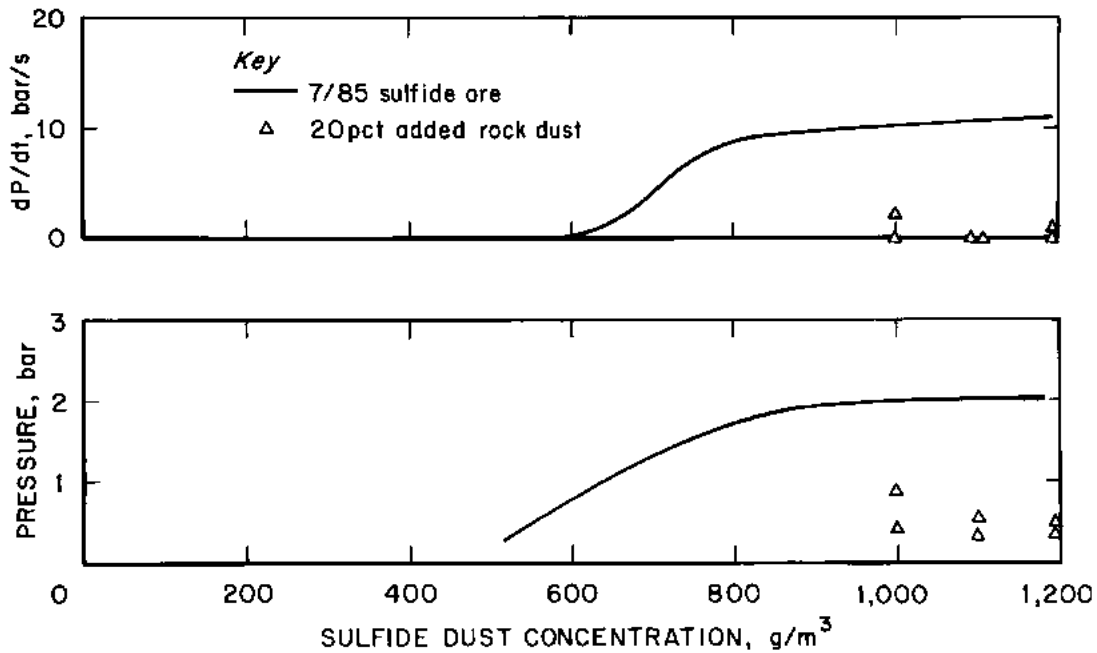
Figure 39.—Explosibility data from the 20-L chamber for a mixture of 80-pct sulfide ore and 20-pct rock dust compared with data (solid curves) for sulfide ore.

Table 14.—Predispersed sulfide ore dust tests in the Bruceton Experimental Mine

Test No.	Concentration, g/m^3	Dust loading,[1] pct	Flame travel, m	Pressure rise, mbar
4211	0	0	3	0
4207	630	100 (S)	10	21
4205	650	54 (T), 46 (S)	3	0
4206	725	59 (T), 41 (S)	3	0
4202	800	100 (T)	7	NA
4204	800	62 (T), 38 (S)	14	90
4203	1,000	100 (T)	3	0

NA No data available.

[1]T refers to V-troughs; S refers to shelves.

with 650 and 725 g/m^3 loadings, the flame extended only 3 m from the cannon, which was attributed solely to the ignitor flame travel. Very little iron oxide residue was evident after these tests. In a final test with the remainder of the sulfide ore dust, a 630-g/m^3 nominal loading was deposited entirely on the three roof shelves. The dust was dispersed by the shock wave of the detonating ANFO charge. The flame extended slightly outby the plastic diaphragm. Both the laboratory and experimental mine tests show that a high concentration of dust

and a strong ignition source are necessary to generate a propagating sulfide dust explosion.

The Pittsburgh Research Center large-scale tests of the explosibility of sulfide ores are consistent with other tests conducted at Russian sulfide ore mines (*89*). For these experimental blasts, the normal precautions of stemming and wetting the face were not taken. The Russian tests showed that sulfide ore with high pyrite content (40 to 50 pct sulfur content) could propagate slow explosions to distances up to 76 m.

FULL-SCALE VALIDATION TESTS IN SULFIDE ORE MINES

Until recently, control of sulfide dust ignitions in mines was based primarily on limestone inerting of the dust generated during blasting and/or quenching the ignition process with water sprays. The most common method was to disperse crushed limestone dust into the face area at the beginning of the

detonation sequence. Water sprays or mists were also used to quench the flame and reduce the dispersibility of the previously deposited dust.

Several low-incendive explosive product formulations for sulfide mines were evaluated in the LLCG, as discussed earlier

in section entitled "Lake Lynn Cannon Gallery Tests." Several of these explosives have undergone field testing and have been extremely successful in preventing dust ignitions in sulfide ore mines in areas most susceptible to these ignitions (*69*).

In July 1990, Kennecott Minerals Co., working together with the Pittsburgh Research Center and two explosives manufacturers, began an extensive testing program with a low-incentive water-gel explosive at its Greens Creek Mine, located on Admiralty Island 29 km southwest of Juneau, AK. The ore types that comprise the mineralized horizon at the Greens Creek Mine are basically divided into two types (table 12): a massive sulfide ore that is composed of 80 to 90 pct pyrite, and a massive sulfide ore with a high base metal content and a low pyrite (10 to 15 pct) content. Historically, the fine-grained, high-sulfur content areas have created the greatest difficulty in controlling sulfide dust ignitions. The dust generated during the mining cycle consisted primarily of pyritic and lead-zinc sulfide. The mining zones containing the various bodies of ore varied greatly in all respects; dips ranged from flat to vertical, and thicknesses ranged from 0.6 to 9.1 m. The dip of the ore body dictated the type of mining method used. The three most frequently employed mining methods were drift and fill with back stoping (used in steeply dipping sections), drift and fill (used in structures dipping between 20° and 50° from horizontal), and room-and-pillar with backfill (used in areas of flat-lying ore). In general, drifting rounds with burn cuts comprised 50 to 60 pct of mine production. A typical production drift had dimensions of 3.7 by 3.7 m. Blasting was conducted using nonelectric detonators, high-velocity detonation cord, and a fuse-detonator initiation system. The cross-sectional area of the drifts ranged from 9 to 28 m^2. The holes in the pattern were 41 mm in diameter and were drilled 2.4 to 3 m deep.

The first sulfide dust ignition at the Greens Creek Mine occurred in 1989 while using the mine's standard explosive: semigelatin dynamite. Others occurred later, with minor injuries to employees primarily associated with the inhalation of combustion products. No major damage was sustained, but production delays resulted in losses averaging about $40,000 per incident. When the Greens Creek Mine first began to experience problems with sulfide dust ignitions, strong preventive measures were instituted. Management designated areas of the mine that presented a potential hazard for a sulfide dust ignition. Once an area had been so designated, the procedures below were followed.

1. The face, ribs, and back within 3 m of the face were thoroughly washed down before loading.
2. All loaded holes were stemmed with one filled water bag.

3. The remainder of the drift within 30 m of the face was thoroughly washed down 45 min before blasting. Any ventilation tube or bag in the drift was also thoroughly washed down.
4. A water-mist spray system was hung from the rib no more than 23 m from the face and directed toward the face. This was turned on at least 15 min before blasting.

These procedures were labor-intensive and, although reducing the number of incidents, did not eliminate them. In fact, the use of water caused other problems. The mine strata are composed of a complex package of strongly altered volcanic rock that degraded quickly when exposed to water. The preventive measures designed to limit the sulfide dust ignitions seriously degraded haulage roads, thereby affecting production.

Production-scale evaluations of the low-incentive water-gel A explosive product (tables 10 and 11) at the Greens Creek Mine began in July 1990. This water-gel explosive comes in prepackaged cartridges rather than in bulk form. Approximately 125 kg of the 38-mm-diam by 400-mm-long water-gel explosive cartridges was used during each face blast. The water-gel explosive A is cap-sensitive and does not require a booster. Initially, the testing was limited to high-sulfur-bearing areas in the mine that were prone to dust ignitions. The standard precautions already in use (outlined above) were employed with the initial testing. Nearly 4,750 kg (approximately 38 average rounds) of the low-incentive water-gel explosive was consumed using the full precautions. No sulfide dust ignitions occurred. As testing progressed, the preventive control procedures were gradually decreased. The water spray was first eliminated while maintaining the washing down of the drift and the use of water stemming. After successfully using an additional 8,430 kg (approximately 68 rounds) of the low-incentive water-gel A explosive with no dust ignitions, the use of water stemming was then dropped; wetting down of the area before blasting was continued. To date, Greens Creek Mine has consumed well over 136,000 kg (approximately 1,100 rounds) of the low-incentive water-gel product and has experienced no sulfide dust ignitions.

The blasting efficiency of the water-gel product at Greens Creek was very high. The water-gel explosive cartridges tamped well, and compaction ratios of 90 pct or greater were the norm. This high degree of coupling improved the brisance (shattering power) and effective work characteristics of the explosive. This kept undetonated particles to a minimum and enhanced the product's ability to control ignition of the sulfide dust. The powder factor ranged from 3 to 3.8 kg/m^3 and averaged 3.6 kg/m^3. The holes were initiated with long-period nonelectric detonators.

In February 1991, a temporary gap in the supply of the water-gel explosive caused the Greens Creek Mine to return to the use of the mine's usual semigelatin dynamite explosive.

As a result, the mine experienced three sulfide dust ignitions in a 2-week period, even while the mine returned to the full list of preventive measures. Past experience had shown that extreme caution was warranted when blasting in the stope where these ignitions occurred. Two days following the last ignition, the supply of the water-gel product was reestablished. The first new round detonated in this stope using the water-gel product used the full precautions. No sulfide dust ignition occurred, and the precautions were reduced to wetting down of the blast area and beyond, which also resulted in no dust ignitions. The Greens Creek personnel by then had developed confidence in the water-gel product and undertook a project to test the product under extreme conditions. A stope had been mined to a point where a slash-while-retreating method was being employed. During the occurrence of the dust ignitions while using the gelatin dynamite, only one slash, no greater than 3.7 m high by 3.1 m long by 3.1 m deep, was taken. This procedure was in effect to minimize dust generation and, hopefully, to eliminate dust ignitions. During the experiment to evaluate the ability of the low-incendive water-gel product to eliminate ignitions, slashes on each side of the initial drive were detonated at the same time with substantial delay. Four such experimental blasts were conducted with this configuration, and no sulfide dust ignitions occurred. Again, the only precaution taken was wetting down of the blast area and beyond. All slash sizes were approximately 3.7 m by 3.1 m by 1.8 m. After over 2 years of testing and full production use, no sulfide dust ignitions have occurred at the Greens Creek Mine when using the water-gel product.

Greens Creek then began full-scale blasting evaluations with a bulk product (emulsion A in tables 10 and 11) that had been tested in the LLCG and shown to be another preferred lower incendive product. No sulfide dust ignitions have occurred since the mine began using the bulk product. Greens Creek has since installed a 455-kg hopper and two more pumping systems for face loading of the bulk product in the development rounds. Overall blasting costs were significantly reduced when using the bulk product compared with the water-gel cartridges. Pumping reduced the labor involved in the loading process by at least one-half compared with the loading times with the prepackaged cartridges. Preblast precautions have been even further reduced to only occasionally

washing down the drift to remove fine sulfide dust deposits. Another benefit attributed to these low-incendive explosives was improved haulage roads because the use of water to control the sulfide dust ignitions was greatly reduced.

Initial incendivity studies at the Greens Creek Mine with the water-gel cartridges and bulk emulsion product were conducted in 41-mm-diam holes and resulted in no sulfide dust ignitions. However, to optimize explosive performance and rock fragmentation, the blast holes were increased slightly to 44 mm in diameter. Both the prepackaged and the bulk products continued to provided excellent results in terms of low incendivity and good fragmentation characteristics. Again, no sulfide dust ignitions occurred when using the lower incendive products in the 44-mm-diam holes.

The lower incendive water-gel explosives have also been used with success in two Canadian base metal mines to reduce the probability of sulfide dust ignitions following blasting operations. These mines are the Westmin Resources Ltd. H-W Mine on Vancouver Island, British Columbia, and the Hudson Bay Mining and Smelting Co. Ltd. Ruttan Operation near Leaf Rapids, Manitoba. The Ruttan Operation had been averaging 2.7 sulfide dust ignitions per month over a 16-month period. During that time, the sulfide dust ignitions accounted for 14 pct of their blast delays, which averaged 2.6 h per occurrence. Any blast that had produced sulfur dioxide (SO_2) was considered to be a sulfide dust ignition. Mine reentry following blasting operations was restricted until the ventilation system had removed and/or diluted the toxic gases to a safe level. The mine's policy was to send mining personnel home if these toxic gases were not cleared within 4 h, which then resulted in lost production time. This occurred one to two times monthly. Preliminary data show no sulfide dust ignitions have occurred when using the low-incendive water-gel product at these two Canadian mines.

Because of the encouraging results in both base metal and oil shale mines, the Colorado Mining Association, DOE, the Canadian Institute of Mining and Metallurgy, the Canadian base metal industry, MSHA, and various explosive manufacturers continue to support Pittsburgh Research Center research on explosive incendivity, stemming, and initiation systems as a means of reducing the dust and/or gas ignition probability during normal development blasting.

RECOMMENDATIONS

In 1984, an ad hoc committee representing oil shale producers, MSHA, the Pittsburgh Research Center, and explosive manufacturers produced a report (*90*) outlining the need for research into the development of safe, effective blasting procedures for the oil shale industry. Many of the committee's recommended research directives have been accomplished by the Pittsburgh Research Center, as summarized in this document. In addition to this research, the authors believe that research should continue on the development of safer explosives and procedures for blasting in the presence of combustible dusts and CH_4 gas in noncoal mines. Intermediate-scale research in the LLCG has been effective in determining the incendivity of blasting agents. A

standard test method could be developed to evaluate the incendivity characteristics of new and/or existing explosive formulations that may be used during blasting operations under hazardous mining conditions, such as in the oil shale and sulfide ore mines. More experimental data on the temperatures and extent of afterburning may lead to a more complete understanding of how best to prevent the ignition of flammable gas or dust atmospheres.

Should the oil shale industry again become active, additional full-scale validation field tests in operating mines need to be conducted to further evaluate the incendivity and performance of the low-incendive explosive products and alternative blasting

procedures before formal recommendations are made. Most importantly for oil shale mines, these tests need to be conducted in the presence of CH_4 to determine and confirm the amount of stemming material required in each hole to prevent the gas ignition. Additional studies also need to be conducted within the target strata to determine the minimum spacing requirement between blast holes with the various explosive products to avoid shock-induced desensitization of the explosives in the adjacent blast holes. This may then require modifying drill patterns and powder factors to optimize the effectiveness of the explosive product while reducing the probability of combustible dust and/or CH_4 ignitions.

The mine tests described in this report used an all-electric initiation system because the current nonelectric systems will ignite flammable gas. However, it would be more beneficial to develop a nonelectric initiation system for use in gassy atmospheres. The majority of noncoal mine blasting operations today rely on nonelectric methods for initiation of the explosives mainly because of the immunity of nonelectric systems to stray electricity. This use of nonelectric initiation eliminates one of the major causes of unwanted detonations in blasting, thus improving safety. However, nonelectric initiation systems, as they exist now, cannot be used in flammable atmospheres due to the inherent potential for igniting that atmosphere. A nonelectric initiation system that would not present a hazard in gassy mines would greatly improve the safety of blasting operations. Any proposed, new nonelectric initiation systems would have to be evaluated for safe use in gassy mines.

There is also a need to develop a toxic fumes standard for approval of various low-incendive blasting agents. Reduction in the amounts of toxic fumes generated in noncoal blasting operations would improve safety, as well as increase efficiency by allowing work to commence sooner in the shot area. Several areas affecting toxic fumes should be examined. First, a standard method needs to be developed to determine the toxic fumes generated by blasting agents. Tests exist for cap-sensitive explosives, but not for blasting agents. Second, the toxicity of blasting fumes should be better defined. Currently, guidelines exist that only limit the total quantity for as many as seven types of fumes. Because the toxicity of some of these types of fumes is greater than that of others, the guidelines should follow some type of weighted-average technique. Finally, the fume characteristics of the explosives need to be determined. This would entail determining which types of explosives generate less, or more, toxic fumes than others and which parameters affect the detonation of the explosives.

CONCLUSIONS

Our research has contributed to a better understanding of the fire and explosion hazards of blasting in commercial-scale underground oil shale and sulfide ore mining operations. The data were obtained through laboratory, gallery, and experimental mine tests and through full-scale blasts in operating oil shale and sulfide ore mines.

The experimental mine and laboratory tests have shown that oil shale and sulfide ore dust can be ignited given the proper predispersed concentrations, particle size, and kerogen or sulfur content. In the Pittsburgh Research Center evaluations, explosions did not occur in oil shale dusts with oil assays less than ~85 L/t (~20 gal/st). For the sulfide ores, explosions did not occur with sulfur contents less than about 20 to 25 pct, depending on the particle size and ignitor strength. The oil shale dusts with the higher kerogen content (higher assays) released more volatiles when heated and thus required a lower entrained dust concentration to ignite and propagate down the heading. These tests have also shown that the oil shale dusts with finer particle sizes required less entrained dust to ignite because the smaller particles were pyrolyzed at a faster rate by the ignition zone flame front, thereby emitting the volatile gases at a faster rate. Even though oil shale dust has been shown to be explosible in the experimental mine tests, sampling of dust depositions following blasting in oil shale mines has shown that the dust generated during blasting was an order of magnitude below the concentrations required to propagate an explosion in the large headings of typical oil shale operations. However, cameras monitoring these same full-scale tests have recorded high-concentration dust clouds generated by the detonation of the blast holes. The ignition of these dust clouds by the hot detonation products could result in an unwanted explosion near the blast face. Inert gelled water stemming has been shown in full-scale oil shale field tests to reduce the dust generated from blasting and to also reduce the air blast overpressures in half. Sulfide ore dusts also exhibit a higher degree of explosibility as the particle size becomes finer and the sulfur content increases. Numerous documented sulfide dust ignitions have occurred following blasting operations and have resulted in personnel injuries and production and equipment losses.

The presence of CH_4 in many of the deep oil shale formations poses a significant hazard to underground blasting operations. A gas sampling system installed at the White River Shale Project in Utah sampled the CH_4 emission rates following blasting operations. The average total CH_4 emission rate was 0.4 m^3/t as sampled over a 3-h period following the blast. Long-term background emission rates have been monitored at two deep oil shale mines. At Horse Draw Mine, CH_4 was continuing to be liberated at a rate of 0.74 m^3/min over 10 years after mine closure. At the White River Shale Project, on the other hand, background CH_4 emissions decreased from 0.3 m^3/min following mine closure in 1986 to about 0.06 m^3/min 2 years later. Both mines have experienced a CH_4

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ignition following blasting. Experimental mine tests have shown that even a small amount of CH_4 can significantly reduce the lower explosible concentration for predispersed oil shale dusts.

Our research also focused on developing safer explosives for blasting in gassy oil shale mines. Tests conducted at the LLCG provided a means of evaluating the relative incendivity characteristics of an explosive product. Two explosives, a pumpable emulsion-ANFO blend and a prepackaged water gel, were determined, through these tests, to exhibit low-incendive qualities compared with other more highly incendive products, such as ANFO. These products were then evaluated in operating mines.

Full-scale field testing of experimental low-incendive explosives was conducted at three oil shale mines in Colorado. Based on data and observations collected during numerous oil shale blasts utilizing drill patterns with hole diameters up to 89 mm, the emulsion-ANFO blend appeared to be considerably less incendive than the conventional blasting agent ANFO while still providing effective fragmentation.

As a direct result of this research, the Canadian base metal industry and MSHA requested that the research on development of safer explosives and blasting procedures for oil shale mines be applied to base metal mines in an attempt to reduce the frequency of dust ignitions when blasting in high-sulfur-bearing ore. Many of these mines had been experiencing several ignitions monthly, resulting in significant production delays. Working in conjunction with explosive manufacturers, the Pittsburgh Research Center has been involved in the recent development and testing of a prototype water-gel explosive and an emulsion blasting agent that have been shown through LLCG testing to exhibit low-incendive characteristics. Full-scale validation tests were conducted at a base metal mine in Alaska and several mines in Canada. After several years of testing and full-production use, no sulfide dust ignitions have occurred when using these low-incendive explosive products.

Based on positive experimental and field testing results, low-incendive explosives show promise as a means of greatly reducing the occurrences of dust and/or gas ignitions following blasting operations in oil shale and sulfide ore mines.

ACKNOWLEDGMENTS

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APPENDIX.—ABBREVIATIONS USED IN THIS REPORT

ANFO	ammonium nitrate-fuel oil		MEC	minimum explosible concentration
BEM	Bruceton Experimental Mine		MSHA	Mine Safety and Health Administration
CFR	Code of Federal Regulations		PETN	pentaerythritol tetranitrate
DOE	U.S. Department of Energy		PPC	Pittsburgh pulverized coal
hvb	high-volatile bituminous		SEM	scanning electron microscope
LFL	lean flammable limit		USBM	U.S. Bureau of Mines
LLCG	Lake Lynn cannon gallery		VOD	velocity of detonation
lvb	low-volatile bituminous		VODR	velocity-of-detonation recorder
MAIT	minimum autoignition temperature			

Cover photo: Logan Wash oil shale mine at outcrop near De Beque, Mesa County, CO. (Photo by Kenneth L. Cashdollar, Pittsburgh Research Center.)