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Coal dust explosions and their prevention

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COAL DUST EXPLOSIONS AND THEIR PREVENTION

BY

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THESIS

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ABSTRACT

The purpose of this study is to review the various causes of coal dust explosions and to summarize the preventive measures through which it may be possible to minimize the hazards of coal dust explosions. The properties of coal dust which have an important influence on its explosibility are: fineness, purity, percentage of volatile matter, dryness, age and degree of oxidation, lifting velocities and dispersability, and its static electrification.

Explosibility of coal dust has been studied in laboratories, in explosion galleries and in an experimental coal mine. The data and results obtained from the tests showed that finely pulverized coal dust has a lower explosive limit of 0.035 to 0.08 ounces per cubic foot.

Although formation of coal dust is not possible to prevent, its dispersion can, however, be considerably reduced if proper preventive measures are adopted.

Application of water during the various phases of mining is an effective means in many instances for reducing the quantity of fine coal dust dispersed during mining operations.

One of the most practical measures is a generous application of rock dust in the mine workings. The requirement of 65 percent of incombustible material in coal mine dust provides a comfortable factor of safety against coal dust explosions.

Other major preventive measures which can be adopted include rock dust barriers, concrete stoppings, and pressure relief vents in their various forms and modifications.

ACKNOWLEDGEMENT

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CHAPTER I

INTRODUCTION

Explosions in coal mines have caused death and injury to miners and destruction of workings in all countries where coal is mined underground. In the United States, the first reported explosion occurred in 1810 and others are still adding to its devastation. Mine explosions are caused by a combination of factors, including concentration of methane in air, formation of dust clouds, and the presence of a suitable source of ignition. These factors vary with the changes of mining methods and practices over the years.

During the last century it was recognized that finely divided coal played a role in many mine explosions. However, the importance of coal dust was not realized until the Courrieres mine explosion in France in 1907 in which 1100 miners were killed. As a result of this disaster, it later became known that, in wide-spread explosions, coal dust is the principal and probably only flame propagating combustible substance of mine explosions. During the first decade of the present century, many serious mine explosions occurred in this country and abroad. In the United States alone, between 1902 and 1911, nearly 350 people were killed annually by major explosions in coal mines; during 1942 to 1951 the average number killed was about 75 per year; and during the last five years there were three major coal mine explosions which resulted in 27 fatalities. Figure 1 shows the number of fatalities in coal mines during the period between 1902 and 1951.

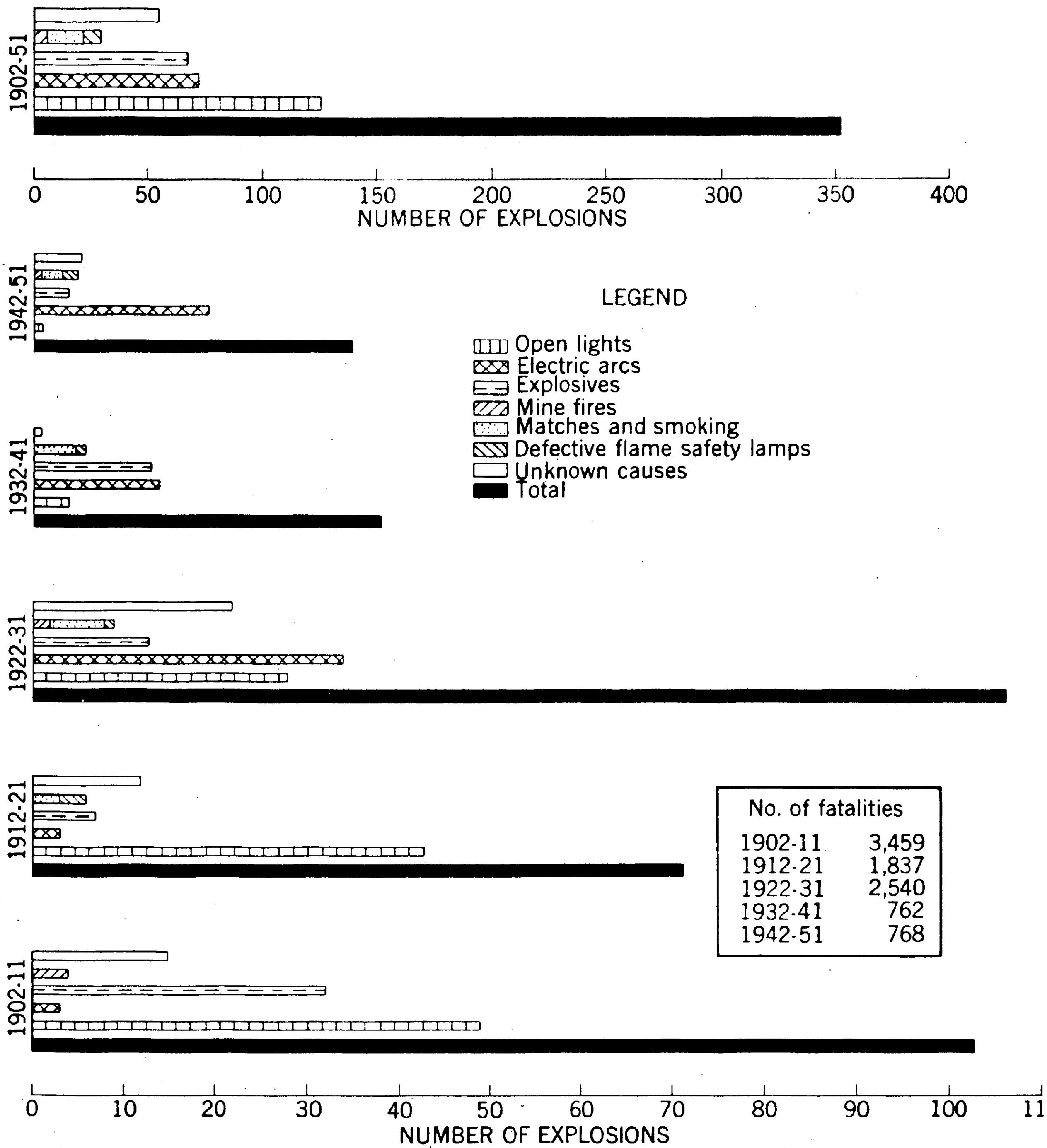


FIGURE 1

From United States Bureau of Mines I.C. 7785, 1957.

In spite of this improving record, further intensive studies of the coal dust problem are being pursued continually. Coal dust explosions are still occurring, however, and many unsolved problems on this subject still remain. This is due, in part, to their complexity and to the many parameters that affect the ignition sensitivity and the explosibility of coal dust. Furthermore, new dust problems are being generated and old ones accentuated by changes in mining conditions and techniques. Modern mechanized mining with high speed machines, multiple-entry systems, multiple blasting, high velocity air currents, and rapid haulage have resulted in greatly increased production which, in turn, produced increased dissemination of fine coal dust and a greater probability of incendive spark formation by frictional contact between moving metals and the mineral impurities of the coal seam.

Significant explosion studies had been conducted in surface galleries in England and other European countries for many years but the opening of an experimental coal mine at Bruceton, Pennsylvania, provided the first underground testing laboratory in which carefully controlled, large-scale tests could be made under simulated mining conditions. The experimental mine has, to date, been subjected to more than 2500 explosion tests which have been directed toward a study of the ignition of coal dust and the initiation of explosions; to observe the development of explosions and their propagation through the mine entry; to study the general physical and chemical phenomena of explosions; and develop measures for preventing or controlling coal mine explosions. Many important data have resulted from these underground studies and, it is believed, that additional discoveries, by these and other similar investigations, will continue to bolster the common effort toward diminishing the coal dust explosion hazard.

CHAPTER II

COAL DUST

Physical Properties

According to the empiric standard stipulated by the United States Bureau of Mines, coal dust means particles which will pass a 20-mesh sieve, produced in mining and handling of coal, or artificially by grinding. To obtain dust of uniform composition and size for the purpose of testing, coarse coal is ground in ball or roller mills, or other pulverizers. As the inflammability of coal dust increases with fineness, samples are rated by the percentage passing through a 200-mesh sieve, all having passed through 20-mesh.

From the point of view of explosibility, the properties of coal dust which require careful study are: (a) fineness; (b) purity; (c) percentage of volatile matter; (d) dryness; (e) age of the dust; (f) lifting velocities and dispersibility; and (g) static electrification.

Fineness

As stated above, the finer the dust, the more readily ignitible it will be. In the tests carried out at the Experimental Mine of the United States Bureau of Mines and in other European countries, the standard of fineness was that 80 percent of the dust should pass a sieve having 200 meshes to the linear inch and all passing through 20-mesh sieve. The dust found in mine galleries varies greatly in fineness; in roadway dust, from 10 to 20 percent of the particles passes through a 200-mesh screen, while in dust taken from the timbers and sides of the roadways about 40 percent passes through a 200-mesh screen.

Purity of the Dust

The susceptibility to ignition and propagation depends also upon the purity of the coal dust. If it contains a large percentage of ash or becomes mixed with a large percentage of incombustible material, it will be less readily ignitible than if it contained no impurities.

Percentage of Volatile Matter

In the explosion tests carried out at the Experimental Mine of the United States Bureau of Mines, it was found that the explosibility definitely varied with the ratio of volatile combustible matter in the coal. Bituminous coals, as a rule, have a ratio of volatile combustible to total combustible ranging between 30 percent and 40 percent. The dusts from such coal are readily ignitible and may be considered as being highly explosive. It does not follow, however, that coal with less than 30 percent of volatile matter are not readily explosive.

Dryness

The drier the coal dust, the more readily ignitible it is and the more readily it is raised into the air; also the drier the atmosphere of a mine, the more readily inflammable will be the dust produced in the mine. It can be shown that in a deep mine at certain periods of the year, the air in the mine is drier than at other periods and, other things being equal, there is more likelihood for an explosion to occur during that period than at any other time. While this is so, a condition of dampness does not by any means form a safeguard against the possibility of an explosion. No condition of wetness less than that in which the dust collecting on the floor, roof, and sides has been converted into mud can be considered an adequate safeguard.

Age of the Dust

If coal-dust is exposed to the atmosphere it gives up some of the contained volatile matter. A coal-dust which contains a large proportion of volatile matter, which is freely evolved from the dust by a high temperature, is more readily ignited and consequently more dangerous from the point of view of explosibility than other kind of dusts which are not so abundant in volatile matter and which do not give off their volatiles so readily.

Lifting Velocities and Dispersibility

When a solid or liquid is broken up into finely divided particles and is dispersed in the air, two important changes take place: (a) the surface area is greatly increased, and (b) the space occupied by the dispersed material is expanded many times over the volume of the original mass. The effect of these changes is to intensify the chemical and physical activity of the material. The rate of oxidation is increased so much that coal dust burns in air with explosive violence. Rates of evaporation and solubility are also increased, and the phenomena of adsorption and electrostatic activity are intensified.

Resistance to travel of a particle through air is dependent upon such factors as: (1) size, shape, and specific gravity of the particles to be lifted; (2) the velocity and turbulence of the air; and (3) the direction of dispersion of the dust, whether horizontally or vertically.

The general equation (10, p. 5) for the resistance to travel may be expressed as:

$$R = \frac{C\rho Au^2}{2}$$

where R = resistance

C = coefficient of resistance

A = projected area of particle

u = particle velocity relative to air

ρ = air density

The drag coefficient, C, is not constant for all conditions of motion but varies systematically with the dimensionless Reynolds number $R_e = ud\rho/\mu$, μ being the viscosity of the air and d the particle diameter. C also varies with the shape of the particle, the effect being different for each value of R_e.

The relationship $C = f(R_e)$ is shown for spheres in Figure 2 for a range of R_e from 10^{-4} to $> 10^5$. The relationship has been divided into three zones according to the nature of dependence of C upon R_e. For high values of R_e, i.e. $R_e > 10^3$, C is reasonably constant. For spheres, C has an average value of 0.44. This is the zone of turbulent motion where the viscosity of the air has no effect. For this region, resistance varies with the squares of particle diameter and velocity:

$$R \doteq k\rho d^2u^2$$

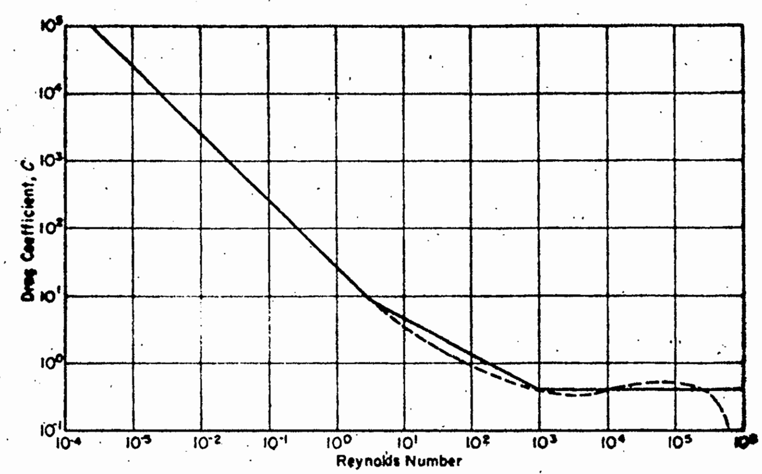


FIGURE 2

From Industrial Dust, McGraw-Hill Book Company, 1954.

For spheres, $k = 0.44 \times \pi/8$; when $R_e < 3.0$, and C varies inversely with R_e:

$$C = \frac{24}{R_e} \quad \text{and} \quad R = k\mu du$$

For spheres, $k = 3\pi$. Here resistance varies directly with particle diameter and velocity and with viscosity of the air, but is independent of air density. It is the zone of streamline motion for which Stokes developed the above equation. (10, p. 6)

Any body falling through a resisting medium such as air or water will accelerate from zero velocity until it reaches some terminal velocity which is dependent upon the densities of the particle and the medium, and the size of the particles. When a particle has attained its terminal velocity, the air resistance is just balanced by gravitational attraction and for spheres,

$$R = F_g = \frac{\pi}{6} d^3(\sigma-\rho)g$$

where σ is the density of the particle and g the gravitational constant. The terminal velocities for spherical particles in the three zones of motion are derived by equating resistance R to the particle weight, neglecting the buoyancy of air:

1. Streamline motion: $U_t = \frac{\sigma g d^2}{18\mu}$

2. Intermediate motion: $U_t = \frac{20(\sigma g)^{2/3} d}{(\rho\mu)^{1/3}}$

3. Turbulent motion: $U_t = \left(\frac{3\sigma g d}{\rho}\right)^{\frac{1}{2}}$

Simplifying these equations for spherical particles falling in air at ordinary temperature, $\rho = 1.2 \times 10^{-3}$, $\mu = 1.8 \times 10^{-4}$, we get, for velocity in centimeters per second and size expressed in microns,

1. Streamline motion: $U_t = 0.003\sigma d_m^{\,2}$ $\quad (d_m < \frac{115}{\sigma^{1/3}})$

2. Intermediate motion: $U_t = 0.34\sigma^{2/3} d_m$ $\quad (\frac{115}{\sigma^{1/3}} < d_m < \frac{2130}{\sigma^{1/3}})$

3. Turbulent motion: $U_t = 16\sigma^{\frac{1}{2}} d_m^{\,\frac{1}{2}}$ $\quad (d_m > \frac{2130}{\sigma^{1/3}})$

For irregular particles obeying Stokes' law, it is convenient to roughly take the terminal velocity for a 10-μ particle, as about 1 fpm (½ cm/sec) and for other sizes is given by the ratio $V \text{ fpm} = \frac{d_m^{\,2}}{100}$, where d_m is in microns.

The two important functions of moving air with which we are concerned in coal mine openings are: (1) the air must be moving with sufficient velocity to dislodge the dust particles from their positions; and (2) the velocity of the air must be sufficient to raise the particles in order to disperse them into a dust cloud.

For dust particles which are lying free on a given surface, the velocity required to dislodge them will be considerably less than that required to remove those which may be attached to the surface. Furthermore, the shape of the particles plays an important part in their ease of removal. Considering the effect of an air current upon a spherical, a rectangular, and a wedge shaped particle, it may be predicted that a wedge shaped particle would be the easier to raise from its indicated position; a spherical particle would be disturbed also; while a rectangular particle would be the most difficult to move.

In mines, we are encountered with turbulent flow. The velocities are much higher and the character of air flow for higher velocities over rough mine surfaces is different from that in smooth-walled ducts. The creation of eddies and swirls in the air in mines is a very important factor in dust dispersal.

Static Electrification of Coal Dust

Several coal mine accidents have been caused by the discharge of static electricity developed by dust laden compressed air. Vivid sparking has been noticed from various belts due to the rubbing of belts against materials and idler pulleys.

The air in metal auxiliary ventilating ducts travels at high speeds and carries with it dust particles produced by mining operations. The particles and the ducts become statically charged and thereby develop the possibility that sparks can pass from the duct to some adjacent body and ignite a firedamp-air mixture. This is especially true where auxiliary fans are used and a good ground is difficult to obtain. The quantity of electricity generated in such a process increases with the speed of the air current, the weight of the dust carried, and with increased particle fineness.

Experiments were conducted in the development of static electricity on solid particles carried by a blast of compressed air through a pipe line. The charge on these particles was collected by placing an insulated iron disc, about 500 sq. cm. in area, opposite the nozzle of a feed pipe. With an air pressure of 3 kg., sparks of 5 mm. to 8 mm. in length could be drawn from the disc. As the charge developed on the particles, an equal charge of opposite sign developed on the pipe walls; but this would normally be discharged over the entire length of the pipe. Charges of about 6,000 volts were developed on various obstacles with the natural air encountered in a compressed air line.

R. A. Dale (9, p. 333) cites a case wherein a deputy found a small accumulation of gas at a ripping lip in a conveyor gate and decided to remove the gas by compressed air. Compressed air for driving two face conveyors was supplied by a seven inch main terminating 20 feet back from the ripping lip which was itself about 15 feet from the face. The deputy coupled a 30 foot length of 7/8 inch hose to the main and "blew it out" on the floor of the roadway. He wrapped the loose end of the hose, in which was inserted a coupling, around the last girder, which was two to three feet from the ripping lip. He stood on the scaffold in front of the ripping lip and instructed his helper to "turn on the air steadily." Immediately the air was turned on there was an ignition of firedamp which injured the deputy.

Tests were conducted in various countries to determine whether detonators can be prematurely initiated during pneumatic stemming by an electrostatic phenomen similar in principle to the ignition of firedamp by dust-laden compressed air. A shot hole is stemmed by grains of sand projected into it by compressed air. The sand-laden compressed air passes through the tube of the stemmer; the tube becomes electrified and the grains of sand acquire an equal and opposite charge.

Static charges on conveyor belts in mines are usually small. (9, p. 335)

The maximum charge detected was only 35 volts. This charge was found near the point where the belt left the idle pulley; one foot away from this point no charge could be detected. The belt used was of normal construction and was moving at a speed of 220 feet per minute. There are certain circumstances under which dangerous sparks can be produced by mine conveyor belts. In one case the rubbing of belt-fasteners against guide rollers gave a spark 3/8 inch wide.

CHAPTER III

COAL DUST EXPLOSIBILITY

Historical

Prior to the first experiment on underground coal mine explosions, which was conducted in the Experimental Mine at Bruceton, Pittsburgh, by the U. S. Bureau of Mines in the year 1910, no conclusive evidence could be obtained regarding the explosibility of coal dust.

Investigations were started as early as 1844 (49, p. 23-40 plus 25-51) by Faraday and have been carried on at the Lievin Gallery and the Pas de Calais coal field in France; at testing galleries at Altofts, Eskmeal, and Buxton, Great Britain; and the U. S. Bureau of Mines at Pittsburgh. International cooperation on mine safety research, begun by an agreement with Great Britain and referring especially to mine explosions, standardization of samples and instruments, and interchange of information, was extended in 1931 to other European nations.

The first experimental explosion was made on October 30, 1911. The mine involved is shown in Figure 3 and consists of a pair of entries driven in the Pittsburgh bed. These entries are nearly parallel with the "butt" joints of the coal and at right angles to the "faces". The coal bed rises on a slight grade from the outcrop. At the time of the experimental explosion, the main entry was 713 feet long, with 669 feet of parallel airway, and with three crosscuts between them. The west one is called the main entry and is the chief explosion passage. A third passage, 198 feet long, enters the airway from the east at an angle of

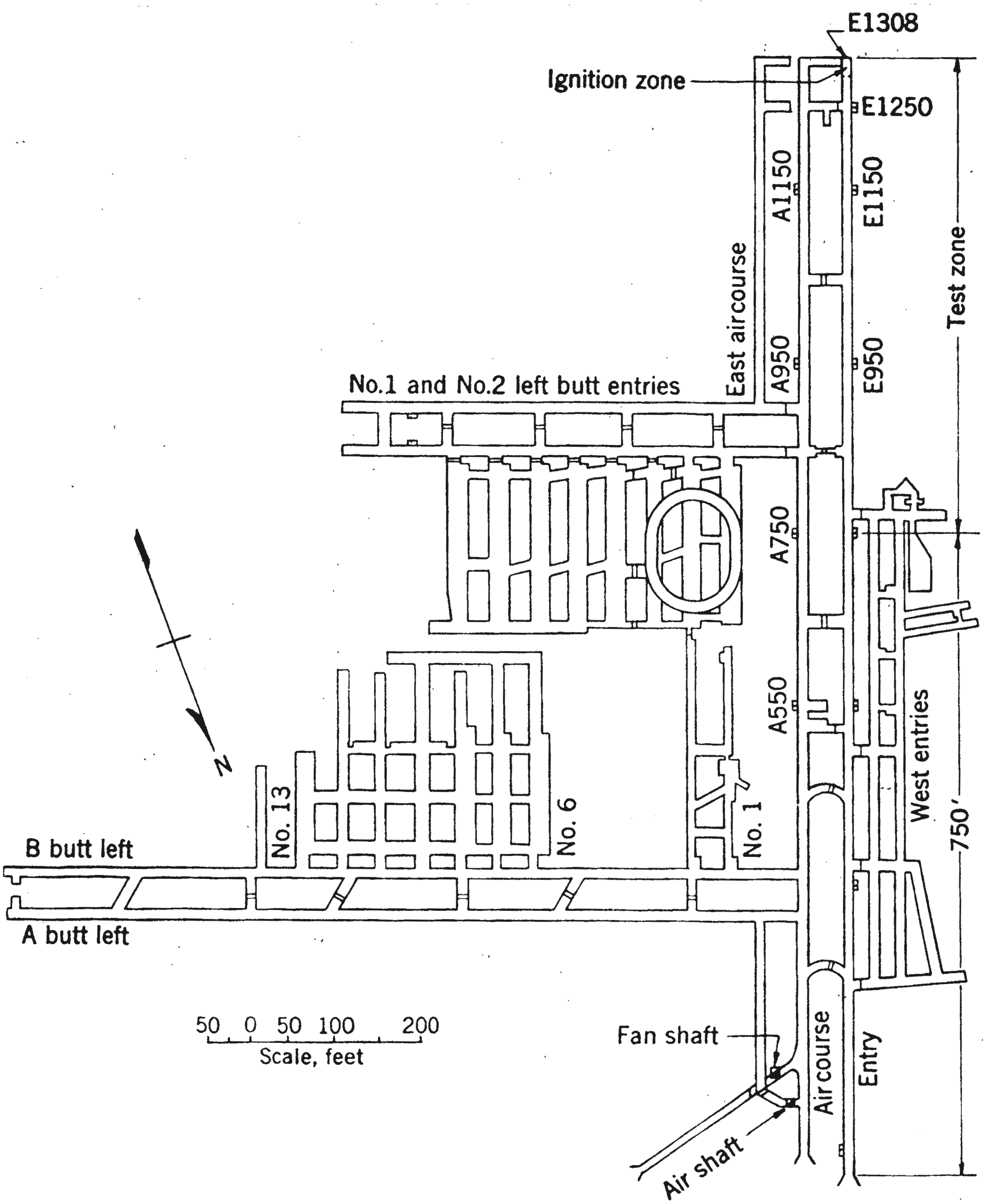


FIGURE 3

From United States Bureau of Mines I.C. 7785, 1957.

55 degrees. It has a small shaft on its east side, 62 feet from the portal for ventilation purposes during development.

Along with the tests in the experimental mine, laboratory studies of coal dust inflammability were also conducted by the Bureau of Mines. This first series of tests proved conclusively that bituminous coal dust is violently explosive and extremely dangerous under certain favorable conditions. The laboratory studies and the results obtained will be dealth with in a separate chapter.

As a result of the above studies, the U. S. Bureau of Mines lists the following factors as effecting the explosibility of coal dust:

1. The composition of the dust, its volatile, moisture, and ash contents.
2. The particle size of the coal dust and inert dust admixed.
3. The presence of inflammable dust in the atmosphere.
4. The quantity of dust raised to form the cloud.
5. The distribution of the dust in the various parts of the mine.
6. The strength of the source of ignition.
7. Surrounding conditions which affect the rate at which energy is taken from the ignited dust, either by direct absorption of heat or by release of pressure.
8. The age of dust.

Chemical and physical properties of dust include the chemical composition, heat of combustion, rate of oxidation, oxygen required for complete combustion, specific heat, particle size, shape, and lifting velocities.

Concentration and uniformity of distribution of dust cloud has a marked effect on explosibility. To be explosive, a dust-air mixture must possess a dust concentration, depending upon particle size, which must be above the lower explosive limit and below the upper explosive limit.

Important properties of the atmosphere are the oxygen content, inflammable gas and vapor present, humidity, temperature, pressure, specific heats of constituents, and heat conductivity.

Industrial dust explosions have been initiated by: (1) electric sparks and arcs, (2) static electric discharges, (3) frictional or metallic sparks, (4) open lights, (5) overheated machinery and other hot surfaces, (6) glowing particles, (7) spontaneous combustion of coal, (8) chemical reaction between constituents of dust mixtures, (9) gas explosion, (10) improper use of explosives.

The size, shape, and construction of the enclosure in which a dust explosion occurs have important influence on the propagation of explosion and the violence which may be caused. The presence of pressure release vents are of special importance.

Other characteristics that influence the explosibility of dusts are the ease of dispersion which depends upon the size, shape, specific gravity, the moisture absorbing quality of the dust, the accumulation of electrical charge of dust particles, tendency of the particles to agglomerate, and the absorption of oxygen or other gases by the dust.

In general, the conditions required for a coal dust explosion are:

1. Formation of coal dust in sufficient quantity.
2. Dispersion of the dusts to form a dust cloud.
3. A suitable source of ignition.

The mathematical probability that the right combination of conditions may occur for both the ignition and propagation of an explosion is small under the operating conditions in most coal mines. But once a small ignition starts, and it produces heat more rapidly than it is dissipated to the surroundings, the adjacent dust will be ignited and the flame will be propagated throughout the mixture. This will cause a very rapid rise in pressure and temperature, which in turn produces shock waves to proceed throughout the mine openings. Coal dust which is present on the floor and ribs, will be lifted up and dispersed into the atmosphere by the rush of air and the turbulence caused by the explosion and thus permit the explosion to be further propagated.

Initiation of Coal Dust Explosions by Gas Explosions

Investigations by the Bureau of Mines have shown that the greatest single cause of ignition of coal dust in the mines of the United States is by an initial explosion of an accumulation of methane which raises and ignites a dust cloud. (55, p. 1)

To initiate a coal-dust explosion, the gas explosion must raise the coal-dust from its resting place, form the dust cloud, and ignite it. The force developed by a gas explosion depends upon various factors.

It is influenced by such factors as the percentage of methane present, total volume of mixture, thoroughness of mixing, the igniting source, and the confinement to which it is subjected.

The lower explosive limit. When there is less than this amount of methane present, the mixture will burn in contact with a flame or other sufficient source of ignition so long as the source is maintained, but combustion ceases when the source is removed. The lower explosive limit has been determined many times and under closely controlled laboratory conditions, as 5.3 percent for a quiet atmosphere. A slightly lower value is obtained if there is turbulence present in the atmosphere and for mining conditions the lower explosive limit may be taken as five percent.

The upper explosive limit. The speed at which an explosive mixture burns depends upon the percentage of methane present. Mixtures just above the lower explosive limit burn very slowly, but there is a rapid increase in the speed of burning as the percentage of methane increases, and a maximum is reached at about 10 percent. With further increase in the percentage of methane, when the upper explosive limit is reached at 14.8 percent of methane, a mixture is obtained which will not give self-supporting combustion. It burns readily, however, when in contact with additional air or oxygen.

Tests in the Experimental Mine

In order to find out the effect of gas explosion on the initiation of coal dust explosion, experiments were conducted in the Experimental Mine. The dead-end of the main entry of the experimental mine was selected as the best place for the tests, and the arrangements are shown in Figure 4. This entry extends 50 feet beyond the last crosscut and is 10 feet wide. The height is about 6 feet at the crosscut and the roof slopes downward a little towards the face. Pittsburgh coal-dust ground so that 95 percent would pass through a 200-mesh sieve was used. One-third of the dust was distributed on the regular 3-inch side shelves attached to the ribs, and two-thirds was placed on cross-shelves near the roof. These shelves were at intervals of 5 feet and were balanced on a narrow support so that a small force would upset them and spill the dust.

Experiments were made with uniform mixtures of natural gas and air. The properties of the natural gas were quite similar to those of methane. For proper control of test conditions, it was necessary to prevent escape of the mixture used, and this was done by erecting a paper diaphragm six feet from the face of the entry.

The tests may be divided into groups according to the volume of gas mixture used:

Group 1. In these tests, 300 cubic feet of an explosive gas-air mixture occupied the entire volume behind the diaphragm. The space was roughly 10 feet wide, 6 feet long, and 5 feet high. With no dust present, a nine percent mixture projected flame about 30 feet from the face of the entry. It was found that 75 pounds of coal-dust distributed on the shelves gave the greatest extension of flame when ignited. The length of the dust-laden zone was 50 feet. Both nine and seven percent mixtures ignited this dust, and flame of its explosion was projected 25 feet beyond

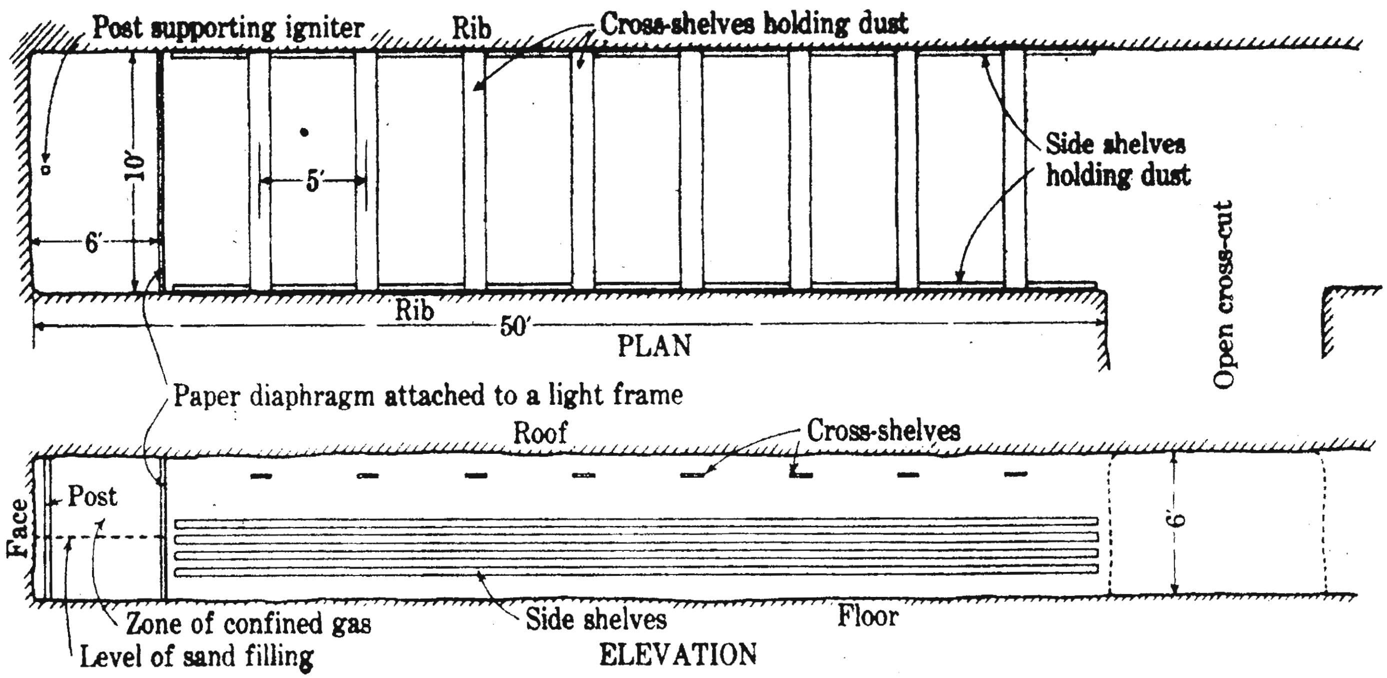


FIGURE 4

From United States Bureau of Mines R.I. 3016, 1930.

the end of the dust loading.

Group 2. These tests were made with 225 cubic feet of an explosive gas-air mixture. Dust conditions were the same as in Group 1. The dust was ignited by a nine percent mixture and the flame of its explosion extended 25 feet beyond the end of the dust zone.

Group 3. One hundred and fifth cubic feet of an explosive gas-air mixture was used. The dimensions of the gas body were then approximately 10 by 6 by 2½ feet. Dust conditions were the same as in Group 1. The dust was ignited by 9 to 10 percent mixtures, and the flame was projected 25 feet beyond the dust zone.

From the results of the tests it may be concluded that the explosion of as small a quantity as 150 cubic feet of a uniform mixture of natural gas and air is sufficient to initiate an explosion of fine Pittsburgh coal dust when so distributed that it is readily thrown into a cloud. These tests emphasize the need of adequate ventilation. The conditions of the above tests required that the coal dust be brought into suspension by the force accompanying the burning of the gas. If coal dust was already thrown into suspension by blasting or mechanical agencies, there is no question that ignition of the dust would occur with far smaller quantities of gas than those used in the tests.

Testing the Explosibility of Coal Dust

Study and research for the explosibility of dusts can be categorized under three general heads:

1. Small scale laboratory testing.
2. Tests in explosion galleries.

3. Explosibility tests of coal dust in a full scale experimental mine.

Laboratory tests are made to determine the fundamental principles involved in dust explosion and to establish the explosibility limits of coal-rock dust mixtures as a criteria for their behavior under actual mining conditions. Full scale tests are employed largely to verify and establish explosibility characteristics of dust and gas in actual size openings. Both laboratory, as well as full scale tests, are vitally important from both a theoretical and a practical point of view. The laboratory work has been extended to all types of industrial dusts, whereas, in the experimental mines, studies have been made only on coal dust. However, any pertinent facts obtained through the studies of other industrial dusts will be considered along with coal dust explosion problems.

In general, laboratory tests were designed to make determinations of the following characteristics of coal and other dusts:

1. The temperature required to ignite dust clouds and undispersed dust layers.
2. The total amount of inert dust required to be mixed with or contained in a dust to prevent its ignition by a suitable igniting source. This property is known as relative inflammability.
3. Energy of ignition. That is, the amount of electrical discharges from a condenser required to ignite a dust cloud or dust layer.
4. Lower limit of explosibility of dust concentration in air and other atmospheres.

5. Maximum pressure developed by explosion of dust clouds of various concentrations and the rate of pressure rise.
6. Effect of oxygen deficient atmospheres on ignition of dusts.
7. Effect of particle size and shape on explosibility characteristics.

The United States Bureau of Mines used various types of apparatus to perform the laboratory tests. Some of them are described below:

The apparatus that was used by Frazer, Hoffman, and Scholl (59, p. 5) is shown in Figure 5. It consists essentially of the explosion flask (a), the platinum coil (i), and devices for putting the dust in suspension and for measuring the pressure developed in (a).

The flask (a), which has a capacity of 1,600 c.c., is provided with large tubulures at its top and bottom; the ends of these tubulures are ground true on a glass plate with emery powder. The brass plate (k), which rests on the end of the top tubulure, carries the platinum coil (i) and the brass tube (m). The brass plate (c), on which (a) rests, carries the small glass funnel (b), which is cemented gas-tight into (c). The contacts between the ends of (a) and the brass plates (k) and (c) is made gas-tight by wide rubber bands. The contraction of rubber draws the projecting portion of the rubber down on the ends, forming a rubber cushion between the ends of the tubulures and the brass plates and by screwing down the nuts above the steel piece (1), the joints at these points are made tight. The platinum coil (i) is suspended near the center of (a) by the two stout nickel leads (j, j), which pass through fiber plugs in (k). The coil (i) is made of about 100 cm. of No. 26 platinum

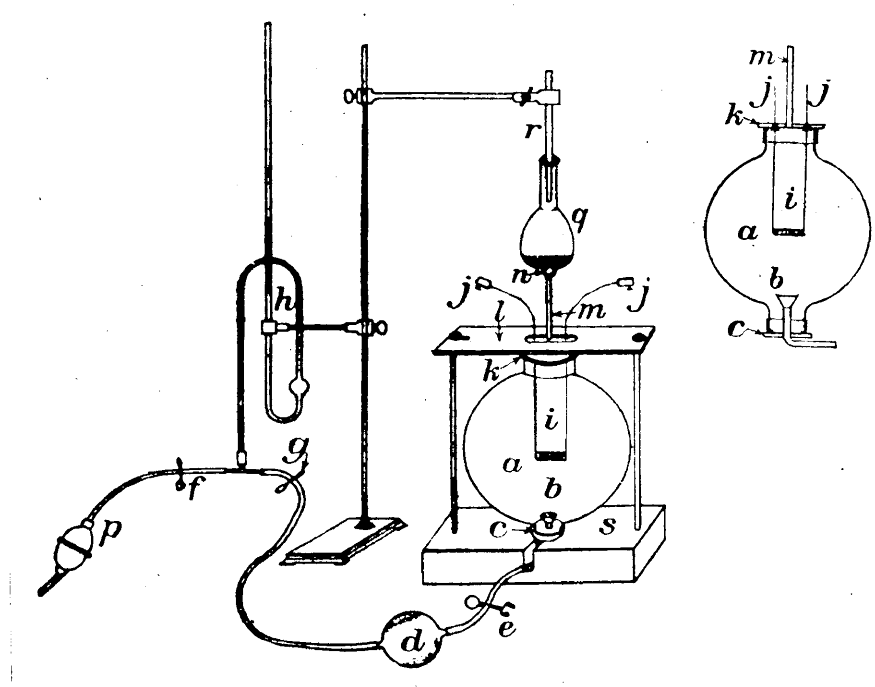


FIGURE 5

From United States Bureau of Mines Bulletin 389, 1935.

wire, wound on a quartz-glass frame, which is attached to the leads (j,j). The steel ball (n) is ground to fit practically gas tight on top of (m), which is soldered to (k) and communicates with (a).

The dust to be investigated is weighed into the glass funnel (b) and at each trial is brought to about the same position in the stem of the funnel, which is then connected by means of a short rubber tube to the 150-c.c. glass bulb (d). By means of the compression bulb (p), the air in (d) is compressed until a pressure of 150 mm. of mercury is indicated by the manometer (h). At the proper instant the dust in (b) is ejected and put in suspension in (a) by suddenly opening the pinch cock (e). In order to ensure a more uniform dissemination of the dust in (a), (b) is covered with a small piece of 18-mesh copper gauze. The pressure developed in (a) is determined by ascertaining by several trials the smallest weight that must be placed on (n) to prevent its being lifted from (m).

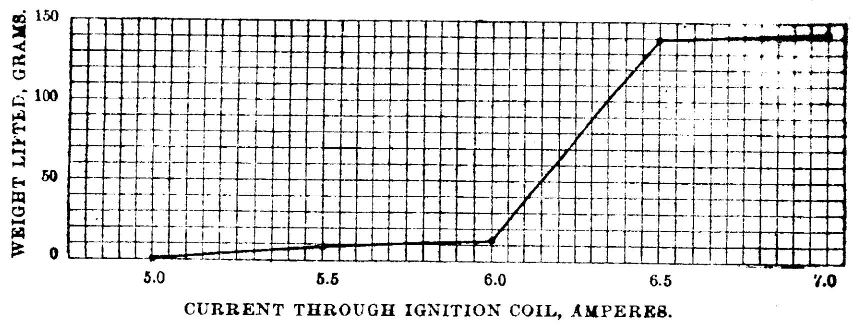
With the apparatus connected as shown in Figure 5, the desired current is passed through the platinum coil for exactly three minutes and during this interval the expanding air in (a) is released at intervals of 1, 2, and 2-3/4 minutes after the instant the current is first passed through the coil. At the end of exactly three minutes, (e) is quickly opened and the dust in (b) is ejected. The experiment is repeated several times; the weight on (n) being varied each time until it is found that the pressure developed in (a) lies between two values differing by five grams. The weight, which is a small flask or other glass vessel containing mercury, is easily varied. In every experiment, 0.05 gram of dust was used for ignition, as with this quantity it has been found that the most inflammable coals are still able to exert the maximum explosive force.

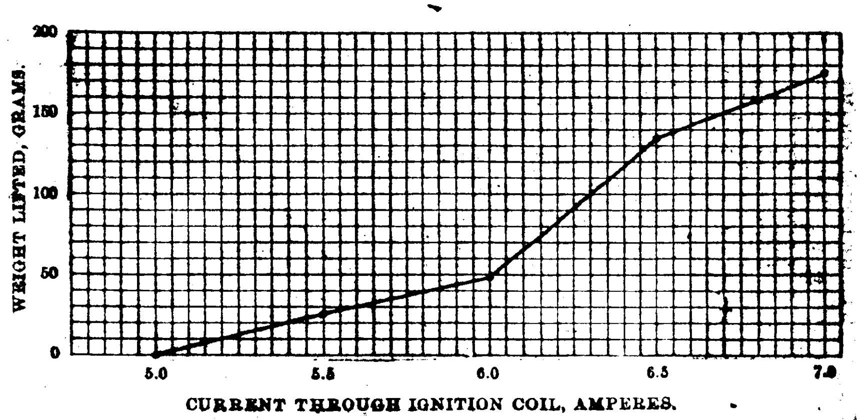
Experiments were made on each dust with current strengths of 5, 5.5, 6, 6.5, and 7 amperes. In all the tests, each sample was air dried to constant weight and then ground fine enough to pass through a 200-mesh sieve; the fineness of all the samples being as nearly as possible the same. Results are plotted with the strength of the current on the coil as abscissas and the pressures in the flask as ordinates; the maximum weight in grams lifted in a test is taken as the pressure corresponding to the strength of current in that test. Since the diameter of the brass tube (m) is 7 mm., the pressure per unit area can be calculated if desired. Six curves are given in Figures 6 and 7 to illustrate the results obtained.

Laboratory Inflammability Apparatus

Figure 8 shows the apparatus (13, p. 4) with the furnace, A, mounted on a tripod. The heated central tube is cylindrical, of refractory material with a 1¼-inch inside diameter by 8 inches long. Heat is supplied through a nichrome winding placed on a groove on the outside of the tube. The temperature gradient along the tube is reduced by concentrating the winding towards the ends. From each end to the center, each successive inch of the tube carries seven, six, five, and four turns of wire respectively.

This tube is enclosed in a sheet-metal shell, 5½ inches in diameter by 9 inches long, provided with ends of transite bored to hold the tube in place. The space between the tube and shell is packed with diatomaceous earth (kieselguhr) to reduce heat losses. A power circuit is connected to the nichrome winding through terminals mounted on the top





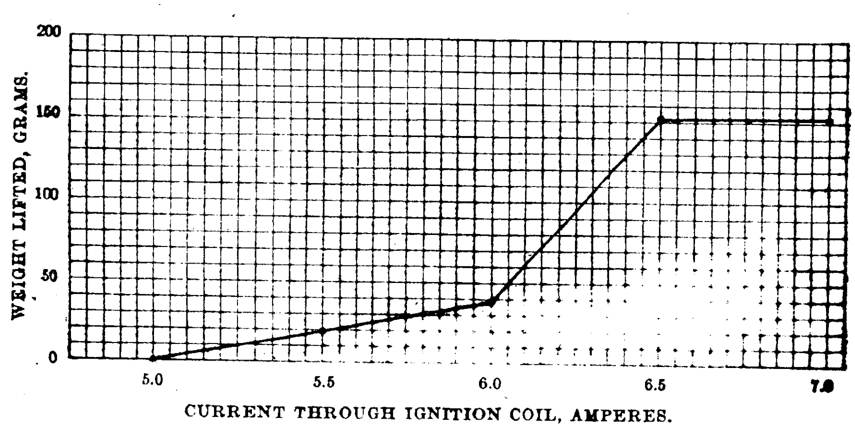


FIGURE 6

From United States Bureau of Mines Bulletin 389, 1935.

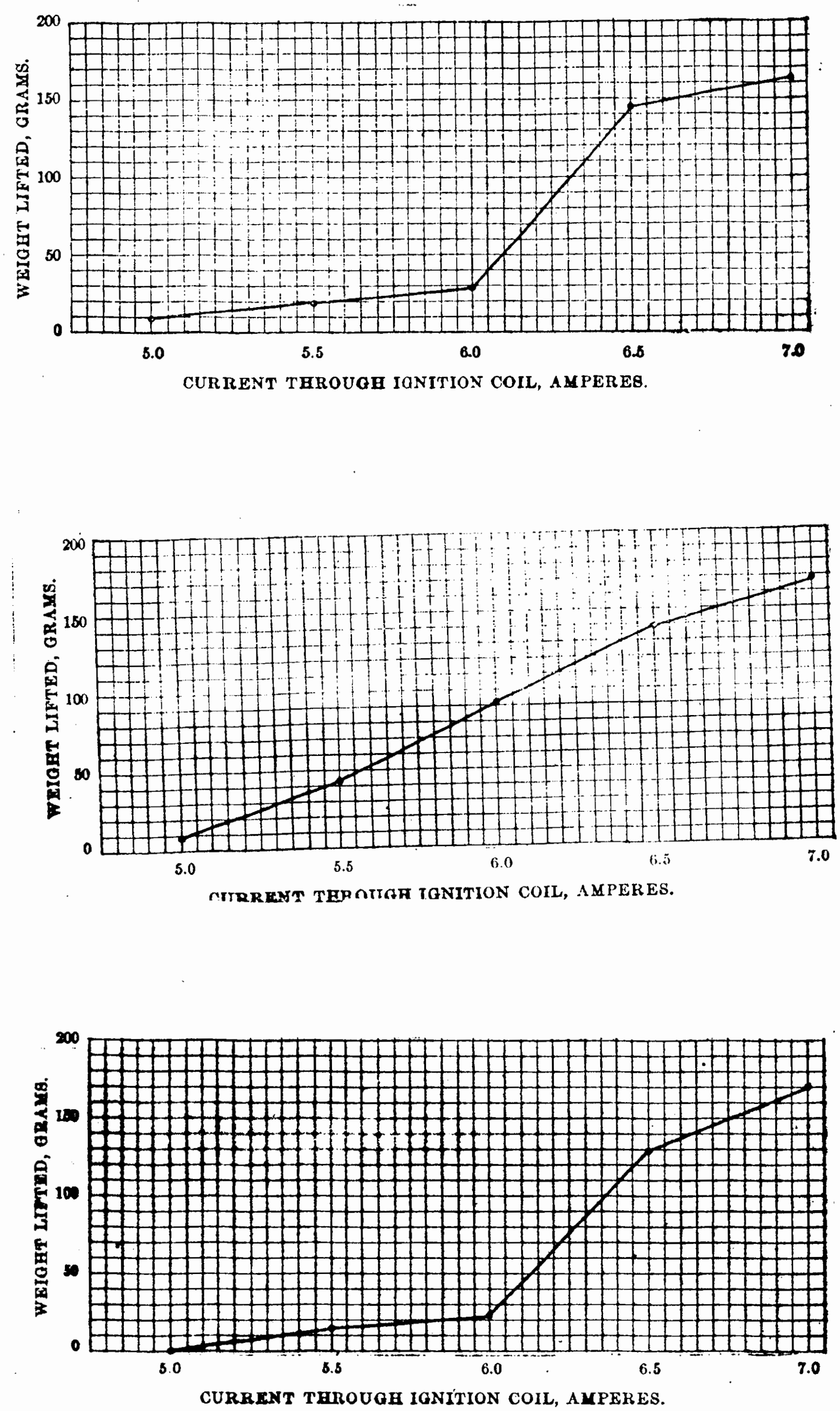


FIGURE 7

From United States Bureau of Mines Bulletin 389, 1935.

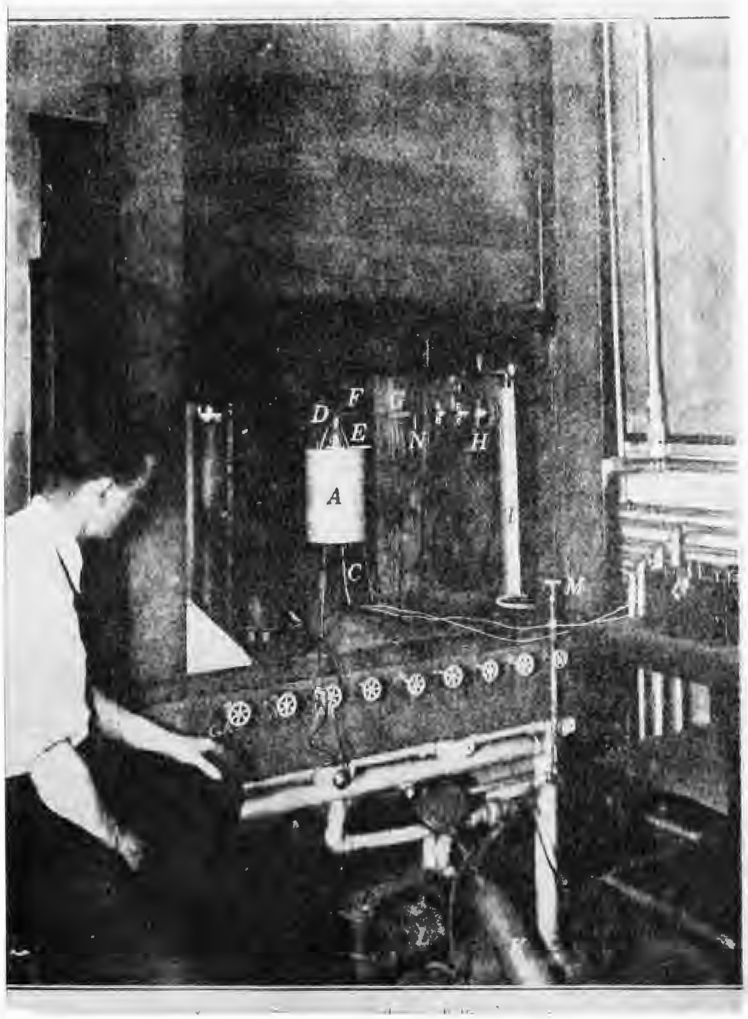


FIGURE 8

From United States Bureau of Mines Bulletin 44, 1912.

and bottom transite heads; a rheostat and an ammeter are included in the circuit for proper control.

The temperature is measured by a base-metal thermocouple, C, inserted in the bottom of the tube. Exploration with this thermocouple showed a temperature gradient from the walls to the axis of the tube and from the center to the ends thereof. This gradient is shown in Figure 9 with values obtained with a midpoint wall temperature of 720 degrees Centigrade.

The dust to be tested is placed in an open-end aluminum boat inserted at (F) in the horizontal part of the dust tube (D), through a ground-glass joint between (F) and (G). The inside of the vertical part of the tube (D) is aligned with the furnace bore, and the tube is kept in position by a short brass extension (E), screwed to the top of the furnace. This extension also holds a coarse-wire gauze that assists in the dispersion of the dust as it enters the furnace. Air for injecting the dust is contained in reservoir (H), a glass globe four inches in diameter and having a side connection to manometer (I). On the floor is a compressor (L) and a high-pressure reservoir (K), from which (H) is filled through needle valve (M). It is necessary to keep the air blast from reservoir (H) constant in different tests and uniformity of release is obtained by a spring operated pinch-cock (N), which is released by a trigger.

Determination of Inflammability

The furnace is heated to the correct temperature; a quantity of the coal dust or a mixture of coal and inert dusts to be tested is weighed

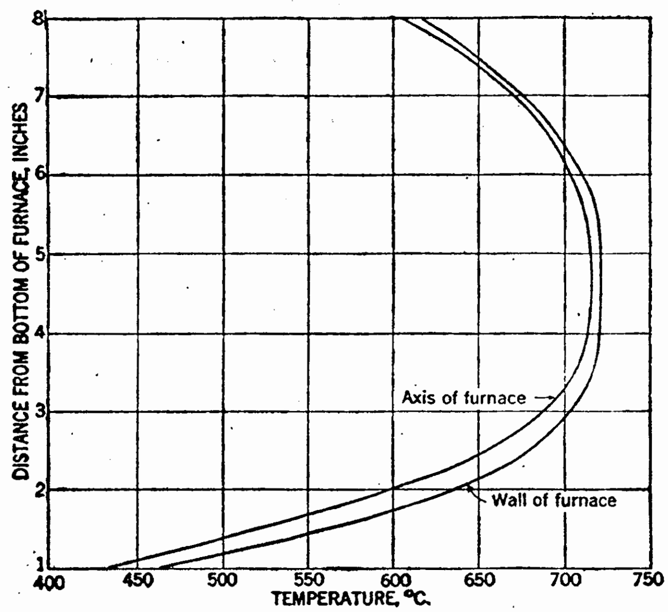


FIGURE 9

From United States Bureau of Mines Bulletin 44, 1912.

into the boat which is then placed in the horizontal arm of the dust tube, after which the ground-glass joint is fitted in place. Automatic release (N) is closed and reservoir (H) is filled with air to the correct pressure. The trigger of the release is pressed and the dust is blown completely through the furnace in the rush of air thus liberated. The cloud appearance is observed as it emerges from the bottom of the furnace. If a volume of flame appears, the dust is termed "inflammable"; if no flame appears the dust is termed "non-inflammable". The test is then repeated to confirm the first observation.

The quantity of dust used and the composition of the mixture are varied in successive tests until two mixtures are found which differ in the proportion of inert dust by 2.5 percent, of which the one containing the least inert dust definitely inflames in the furnace tube at some weight of dust used and the other does not inflame at any of the weights tested. Inflammability of the coal dust is taken as the mean of the contents of total incombustible material in the two critical mixtures. Total incombustible is the sum of the inert dust and the moisture and ash of the coal dust mixed with it.

It was found that a mixture close to the limit of inflammability might inflame at some given quantity of dust and not at others. Tests were made with total weights of dust ranging from 0.5 to 1.5 grams in steps of 0.2 grams.

Large-scale tests of fine dusts prepared from British coals were conducted by Mason and Wheeler (13, p. 10) in a steel gallery at Buxton, England, and from the results obtained, the following mathematical relationship was deducted: $S = 100-(1,250/V)$, in which S is the incombustible content of any limiting mixture and V is the volatile ratio of coal expressed in percent.

Godbert-Greenwald furnace. This appartus was developed by the U. S. Bureau of Mines for the study of inflammability of coal dust and its use has been extended to other types of industrial dusts. Figure 10 shows the apparatus with its test chamber consisting of an electrically heated cylindrical tube furnace. The dust to be tested is placed in a small boat which is connected to the top of the furnace by a tubing. A weighed amount of dust is placed in the boat and the dust is conveyed pneumatically through the tube to the furnace, the temperature of which is controlled by an automatic thermostat. This apparatus can be employed to determine: (1) the minimum temperature of ignition, and (2) the relative inflammability, both by cut-and-try methods.

For minimum temperature of ignition studies, the temperature of the furnace is varied and four trials are made at the temperature at which ignition does not occur. The lowest temperature at which ignition does occur, as indicated by the appearance of a flame at the lower mouth of the furnace, is recorded as the ignition temperature of the dust.

Relative inflammability is determined with the temperature of the furnace held constant at 700 degrees Centigrade (1292 degrees Fahrenheit). As an initial step, pure dust is blown into the furnace. If it gives out a flame, increasing quantities of Fuller's earth are added to increase the content of inert material. This is continued until the minimum percentage is found which will prevent ignition of the powder in four successive trials.

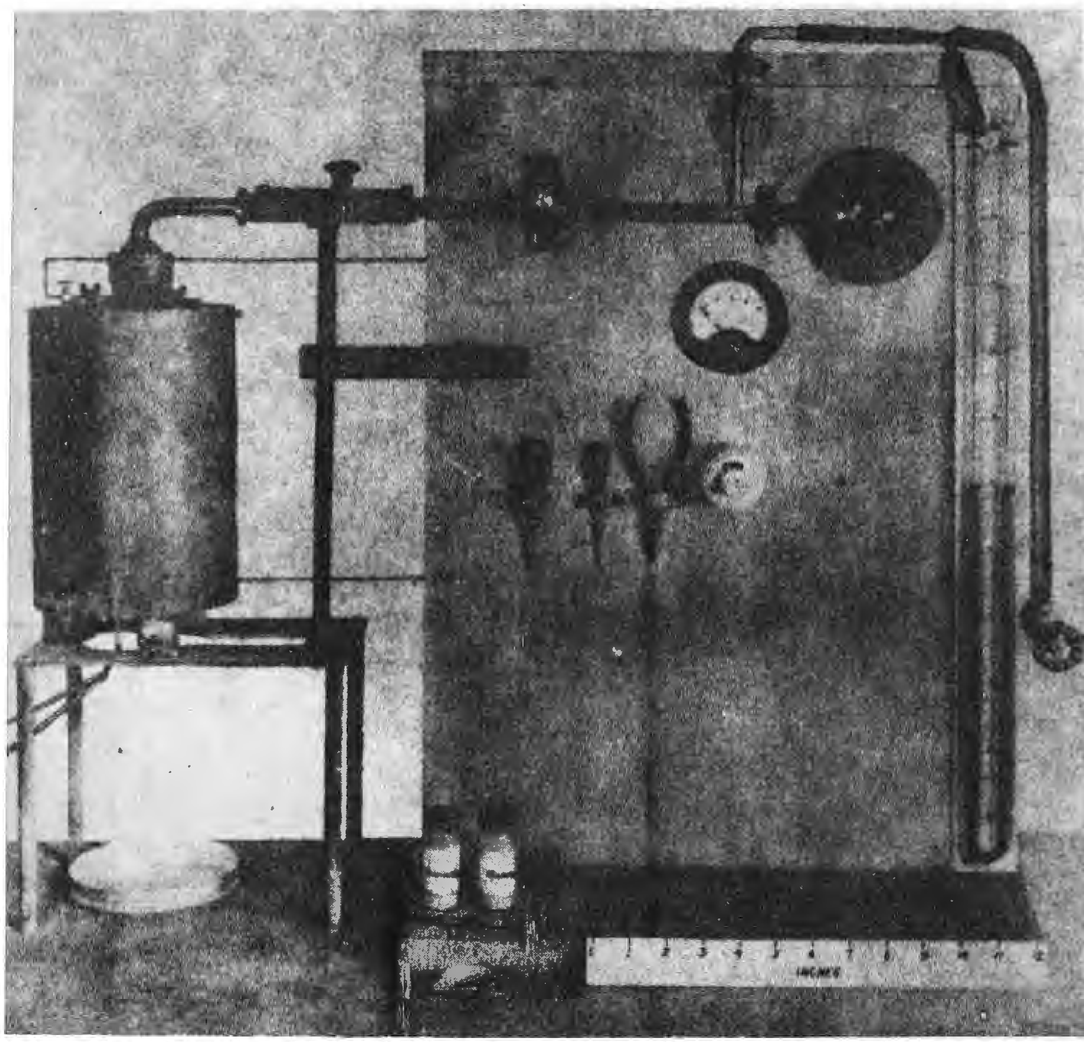


FIGURE 10

From United States Bureau of Mines R.I. 3132, 1931.

Open-spark apparatus. The appartus used is shown in Figure 11. It differs from the Godbert-Greenwald appartus only in that an electric spark is substituted for the furnace as a source of ignition. It consists of a high-voltage continuous spark having an average power of 20 watts which is passed through a gap between electrodes in a pyrex tube. The test sample is blown through the tube so that the spark is completely immersed in the moving dust cloud. The apparatus is flexible so that the atmosphere in the tube, which carries the dust, may be composed of any gas mixture desired.

Hartmann apparatus. This apparatus was designed; (1) to determine the maximum pressure and the rate of pressure rise during explosions of dust clouds, (2) to measure the lower concentration limit of explosibility, and (3) the minimum energy required in a static spark to ignite dust clouds. A sketch of the apparatus is shown in Figure 12. The explosion chamber is a cylindrical lucite tube 12 inches long by 2.75 inches in internal diameter, with a 0.25 inch thick wall and enclosed in a split-copper sleeve to prevent rupture by high pressures. The copper sleeve has observation ports and the lucite tube has inserts at two inch intervals for placing electrodes. The bottom of the lucite tube is closed by a brass cap which supports a specially designed cup for holding the test sample which is dispersed upward in the tube by air or gas, depending upon the test being made. The top of the tube is connected to a manometer for pressure measurement or it may be closed by a paper diaphragm if pressure measurements are not desired.

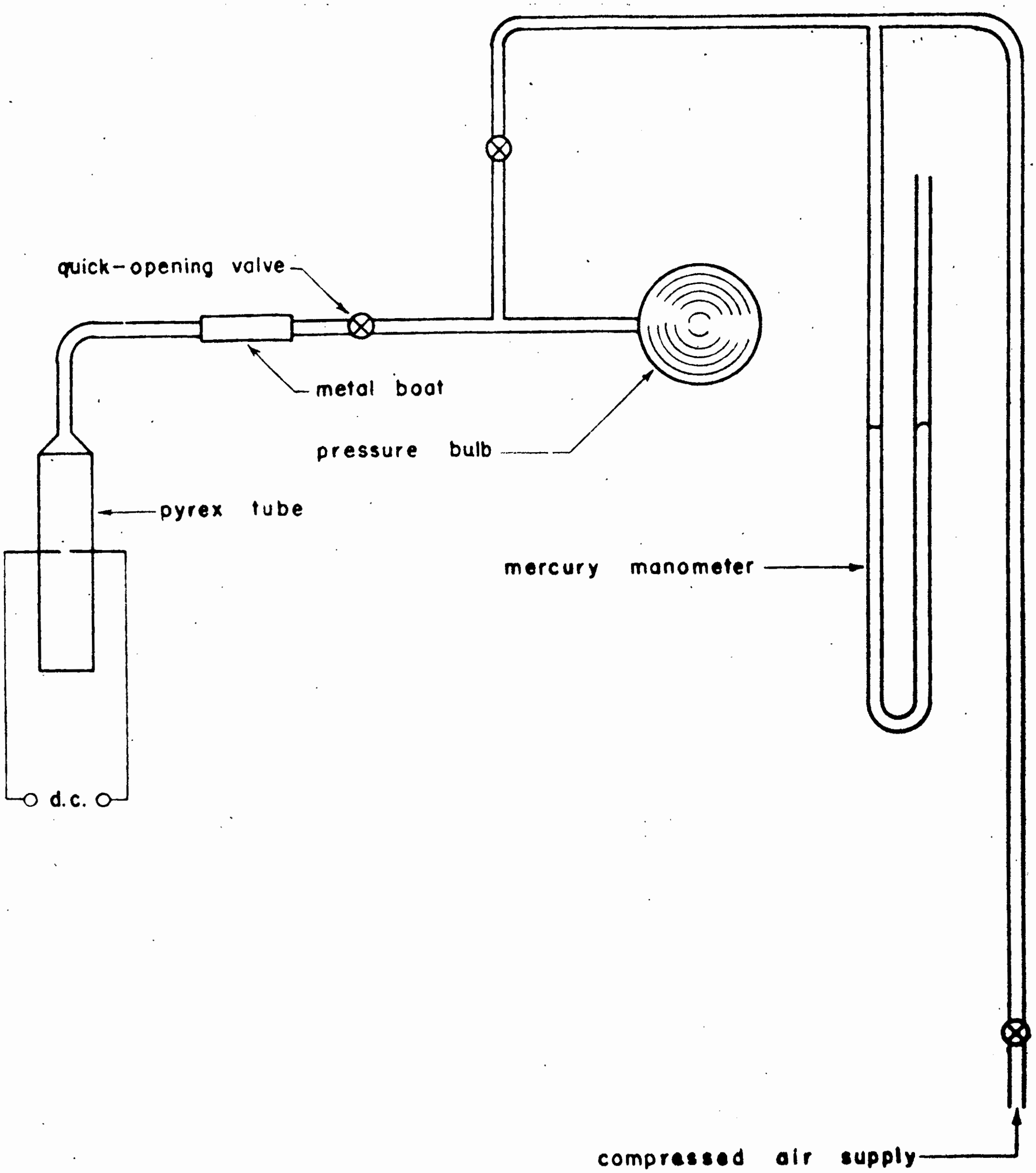


FIGURE 11

From United States Bureau of Mines R.I. 3132, 1931.

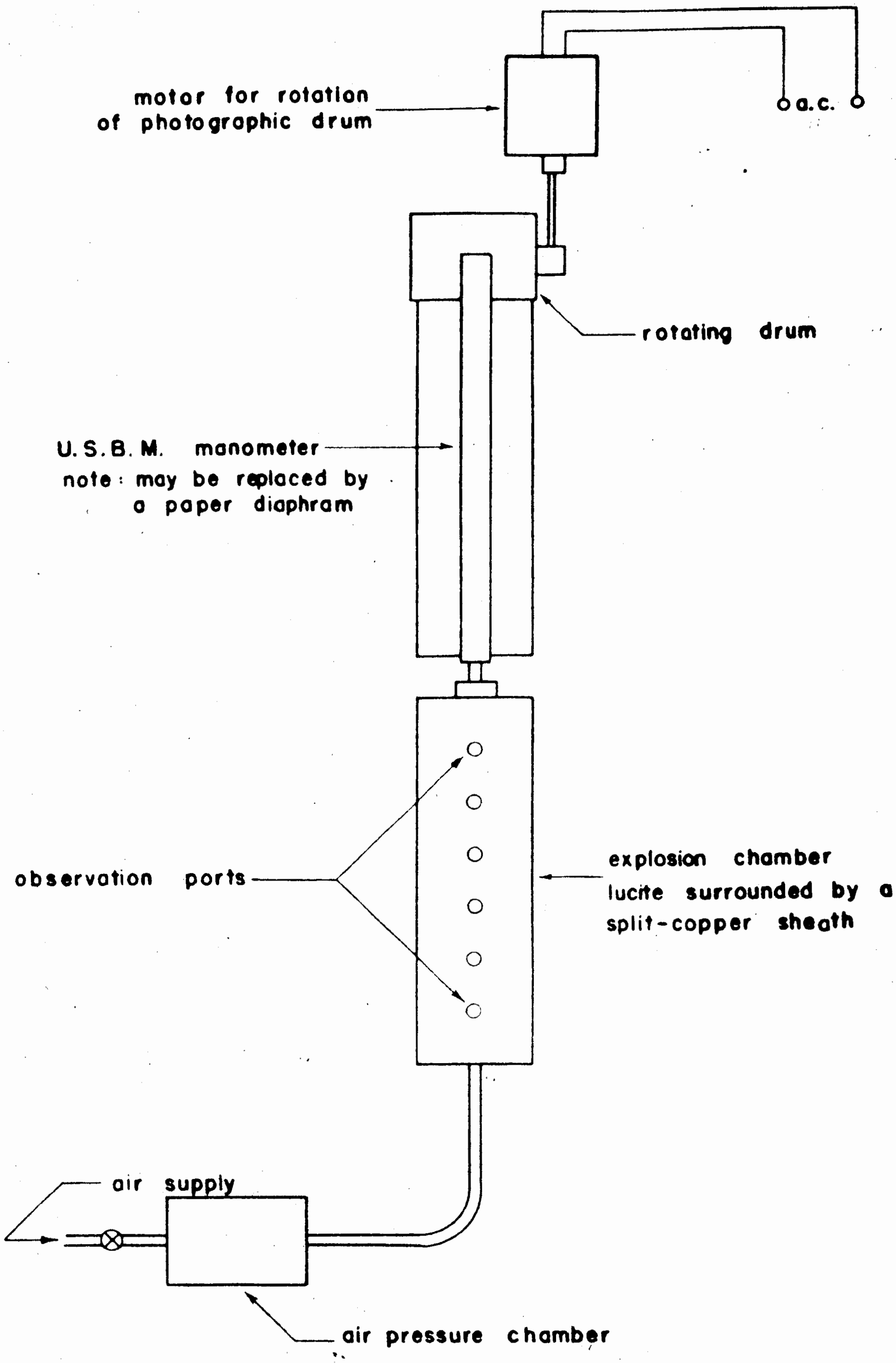


FIGURE 12

From United States Bureau of Mines R.I. 3132, 1931.

The source of ignition is an electric spark with an average power of 24 watts. Five levels or vertically displaced positions for the electrodes are provided to assist in obtaining lower inflammability limits. The top of the explosion cylinder is covered with a paper diaphragm and the minimum quantity of dust required to ignite at each electrode level is found by successive trials. The higher the electrodes, the greater the quantity of dust which must be dispersed into the chamber. This shows that the dust cloud is not uniform but is denser in the lower portions of the tube. With this equipment, however, the vertical spacing of the electrodes permits a more accurate estimation of the lower limit than can be obtained from data taken with the electrodes at one level only.

The manometer used for measuring pressures is of the diaphragm type with magnification of diaphragm movement through an optical lever for recording on photographic film mounted on a rotating drum. The Hartmann apparatus is also employed to determine relative inflammability and the electrical energy necessary to ignite dust clouds. This latter measurement is made through the knowledge of the quantity of electricity discharged across the spark gap from a charged condenser.

Tests in Explosion Galleries

Explosion galleries used by the U. S. Bureau of Mines vary in size and shape with most of the experiments being carried out in one to six foot cubical enclosures. Elongated galleries have also been used for experimentation.

A four by four foot gallery is of typical construction and made of 2-inch tongue-and-groove pine planking reinforced with steel angles along the outer edges and lined on the inside with galvanized steel sheet. There are four adjustable openings or vents in the walls with one on top and one on each of three sides. Openings may be adjusted from 18 by 30 inches to smaller sizes. The fourth side has a door on vertical hinges. This gallery is anchored to the floor of a shed which is 12 by 16 feet by 10 to 12 feet high with large doors and removable panels.

Pressures are measured by two types of manometers; one for higher and the other for lower pressures. Two or three manometers were used in each test with care being taken to prevent heat effects from deflecting the diaphragm of the manometers.

Ignition was initiated either by an electric spark or a guncotton flame. The factors affecting combustion and the pressures developed are; (1) dispersion of the dust, (2) effects on the location of the dust cups, (3) location of the ignition sources, (4) timing of ignition relative to dust cloud formation, (5) the effect of dust concentration upon the severity of the explosion, and (6) the size of vents and material used to cover them.

<u>Lower limit of explosibility</u>. (6, p. 44) Particle size has a marked effect on explosive limit. For the purpose of studies, dusts used are such that 50 percent or more pass through a 200-mesh screen. The lower explosive limit also depends upon the source of ignition. A weak electric arc would not ignite dispersions of an atomized aluminum

powder in air if the concentration was below 0.05 ounce per cubic foot but a rapid timed flame of a small quantity of gun cotton could start explosions in concentration as low as 0.025 ounce per cubic foot. The corresponding ignition limits for finely pulverized coal dust were found to be 0.035 and 0.005 ounce per cubic foot.

If heat losses are neglected, the heat of combustion produced by a unit volume of dust cloud must be equal to the heat required to raise the temperature of a similar volume to ignition level. This is demonstrated by the expression;

$$pq = p\ (T-t)\ c_2 + (T-t)\ sc_1$$

wherein,

$$p = \frac{(T-t)\ sc_1}{q - (T-t)\ c_2} ,$$

p = lower explosive limit of dust concentration,

q = heat of combustion of a unit weight of dust,

T = ignition temperatures of dust,

t = initial temperature in a dust cloud,

s = density of air at initial temperature and pressure,

c_1 = specific heat of air at constant volume, and

c_2 = specific heat of dust.

This expression is only an approximation because it neglects heat losses, dissociation, incomplete combustion, and pressure effects in front of the flame or combustion zone; and does not take into account the particle size effects. It is useful for comparative purposes only.

The above formula may be modified by introducing a factor q' which is found by correcting q for the errors stated above. Then;

$$q' = q - k\,(T-t) - d(T) - f(s)$$

where:

k = a heat loss factor of the apparatus,

$d(T)$ = loss of heat due to dissociation, and

$f(s)$ = loss of heat due to particle size effects.

The expression for explosibility limit would then be:

$$p = \frac{(T-t)\ sc_1}{q - (k + c_2)\ (T-t) - d(T) - f(s)}$$

The three new correction factors have to be determined by experimentation and, at present, very few such data are available.

Tests in the Experimental Mine

A brief description of the experimental mine has been given on page 15 and in Figure 3. The seam consists of five feet of nearly clean coal, one to two feet of friable clay, and one to two feet of "roof coal", which is composed of alternate layers of coal and shale. The upper coal layers make an excellent roof, usually requiring no timber support and being capable of withstanding the shock of explosions in most places.

Stations have been cut along the main entry for experimental purposes and the cut-throughs have been blockaded with stoppings in order that the main entry can be used separately for many of the tests. The floor of the main entry was concreted and the walls and roof were covered with gunite to prevent contamination of the dust samples and to strengthen them as well as to facilitate cleaning up after an explosion.

The mine is equipped with wiring systems so that appropriate measurements can be taken of temperatures, pressures, flame velocities, and any other quantities desired. Automatic gas samplers are used to sample before, during, and after an explosion.

Test 1. Pulverized Pittsburgh-bed coal was loaded at rates of 0.032, 0.04, and 0.08 ounces per cubic foot and subjected to ignition "source B", consisting of a blown-out shot of four pounds of FFF black blasting powder tamped with three pounds of clay. This was fired from a steel cannon resting on the floor at the face of the main entry into 100 pounds of pulverized Pittsburgh-bed coal-dust distributed over the 50-foot dead end inby the last crosscut. The dust under test is distributed in the crosscut and outby on both entries for a distance of 300 feet. The blown-out shot develops an explosion in the pulverized dust which is projected into the dust under test.

In some tests the dust was distributed on overhead cross shelves, which are fixed at 10-foot intervals. An explosion was obtained with 0.08 ounces of dust per cubic foot but was not obtained with 0.04 ounces. A second trial was made with the dust placed on special shelves so arranged that they would be thrown down and the dust scattered in the air. With this arrangement, propagation was again obtained with 0.08 ounces of dust per cubic foot and was not obtained with 0.032 ounces. That the latter method of forming the dust cloud was more effective was shown by the fact that the explosion with the 0.08 ounce loading was much more rapid and violent than with the former method.

Test 2. In this test, pulverized Pittsburgh-bed coal-dust (92 percent through 200 mesh) was loaded at the rate of 0.032 ounces per cubic foot. The dust was equally divided between the overhead cross shelves, longitudinal side-shelves, and the floor; a method of loading that has been standard in most of the work. The source of ignition was a gas explosion, called "source C". The dead end of the main entry, 50 feet long, is closed by a paper diaphragm and a uniform mixture of natural gas and air in proportions to obtain maximum explosibility is prepared in it. In the test there was 9.3 percent of gas in the mixture which was ignited eight feet from the face by firing a black-powder igniter into it. This gas mixture gives a violent explosion, pressures are three to four times as great as obtained from source B, and there is a greater projection of flame from this explosion zone in the entry head and into the dust testing zone beyond. The dust raising power is greatly enhanced by the higher pressure. An idea of the violence may be obtained from the fact that some of the three by eight inches oak cross shelves are broken frequently.

Propagation was not obtained in this test, as the flame died out after traversing about half of the coal dust loading, It is evident that there was not sufficient dust raised in the path of the flame to give a self-sustained explosion even though the dust cloud was fairly uniform.

A similar test was made with dust passing 95 percent through 20-mesh and 20 percent through a 200-mesh sieve, and loaded at the rate of 0.04 ounces per cubic foot. This loading also failed to give propagation.

The flame traveled through about 80 percent of the dust and, therefore, it was closer to the dividing line of the density required than the previous test with pulverized dust.

Tests in the Experimental Mine have shown that the lower limit of inflammability of minus 200-mesh Pittsburgh coal dust lies between 0.032 and 0.08 ounces per cubic foot. This is a very small amout and is equivalent to three ounces per linear foot in an entry of 60 square feet in cross section. Such a small amount, when distributed over all the inner mine surfaces, could be detected only by careful examination. On this basis, it is evident that no coal mine can be so clean that propagation of a coal dust explosion is impossible.

Electricity and Coal Dust Explosion

With the increased use of electricity in present day mining, direct ignition of coal dust by electricity requires careful consideration. Laboratory investigations started in the early 19th century and the investigators agreed that for direct ignition of coal dust, a dust cloud of sufficient density must be brought in contact with an electric spark or arc.

Demonstrations in the Experimental Mine

Demonstrations of the ignition of coal-dust cloud by an electric arc are carried out in the entry pit mouth, which is concrete lined, has a width of eight feet and a height at the center of the arch of a little more than eight feet. A dust cloud is formed by blowing coal-dust from a shelf near the roof by means of compressed air. The arrangement is shown in Figure 13. An arc lamp is suspended six to seven feet in front of the shelf with the arc about 18 inches below. All protecting glasses of the lamp are removed so that the arc is fully exposed. When the lamp is in operation, the current in the arc is 6.5 amperes and the potential across the carbons is 65 volts. The arc used has much less intensity than one that might occur by the grounding of a trolley or power line or of an extended lighting circuit.

When air is admitted to the pipe on the shelf, about 35 pounds of the dust is blown off and falls over the arc in a well-dispersed cloud. The coal dust thus blown from the shelf obscures the arc momentarily while a flame starts from it and spreads through the cloud with a rush to the portal and into the open air beyond. The size of this arc is small when compared with those normally met in mining practice whose current drawn in a direct short circuit may be hundreds of amperes and at voltages of 250 to 500. Such arcs would have enormously greater igniting power.

<u>Experiments at the face of the main entry</u>. The demonstration described above gives no quantitative data on the ignition variables involved. This particular test was made in order to determine the amount of dust required to initiate a self-sustaining explosion with a given arc, how the danger varies with the composition of the coal, and how both of these vary with the voltage and current of the arc.

The face of the main entry is 50 feet in advance of the last open cross cut, its size is about 6 by 10 feet and with a dead-end volume of about 3,000 cubic feet. Figure 14 shows the longitudinal section of the

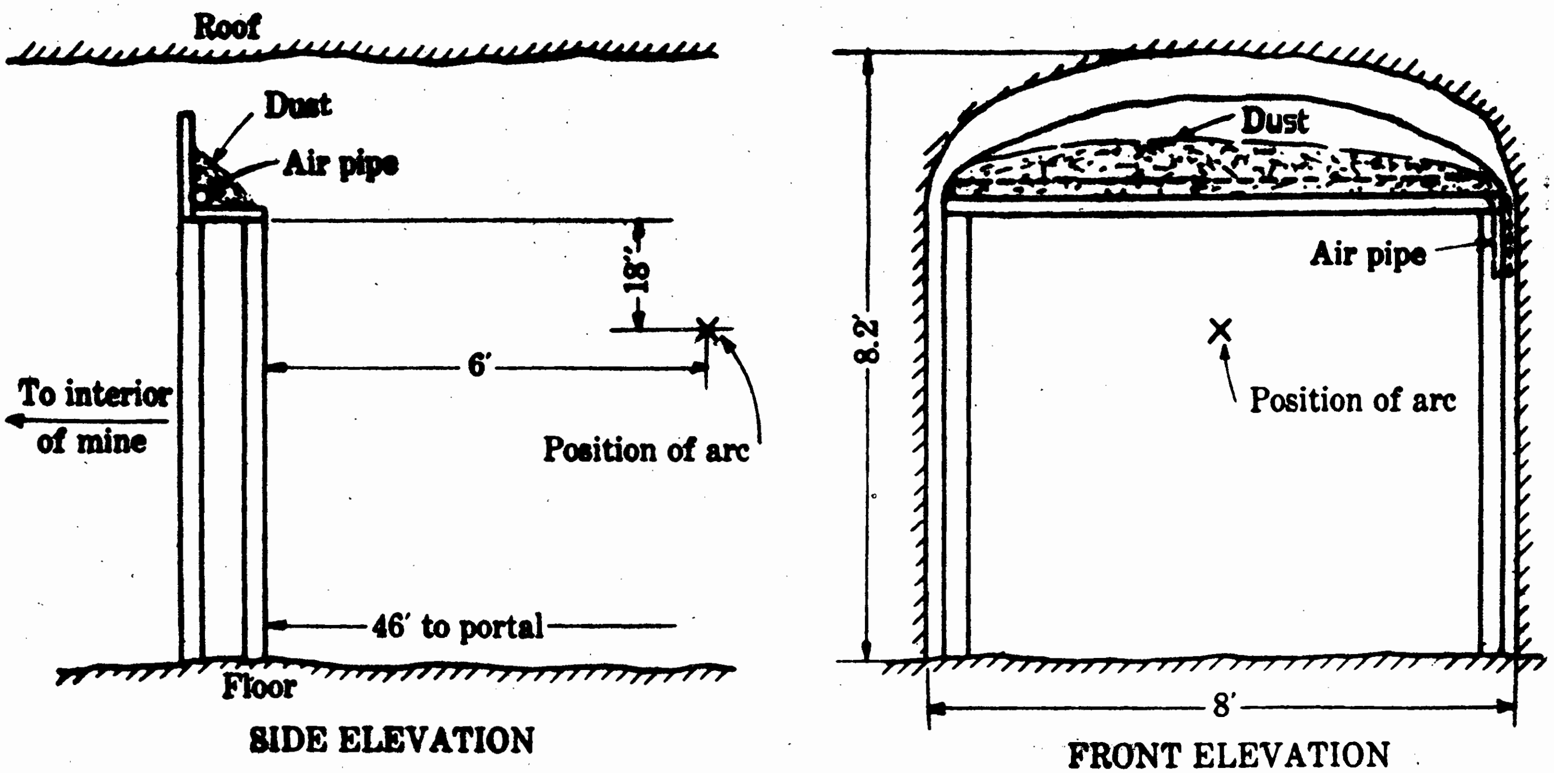


FIGURE 13

From United States Bureau of Mines R.I. 3044, 1930.

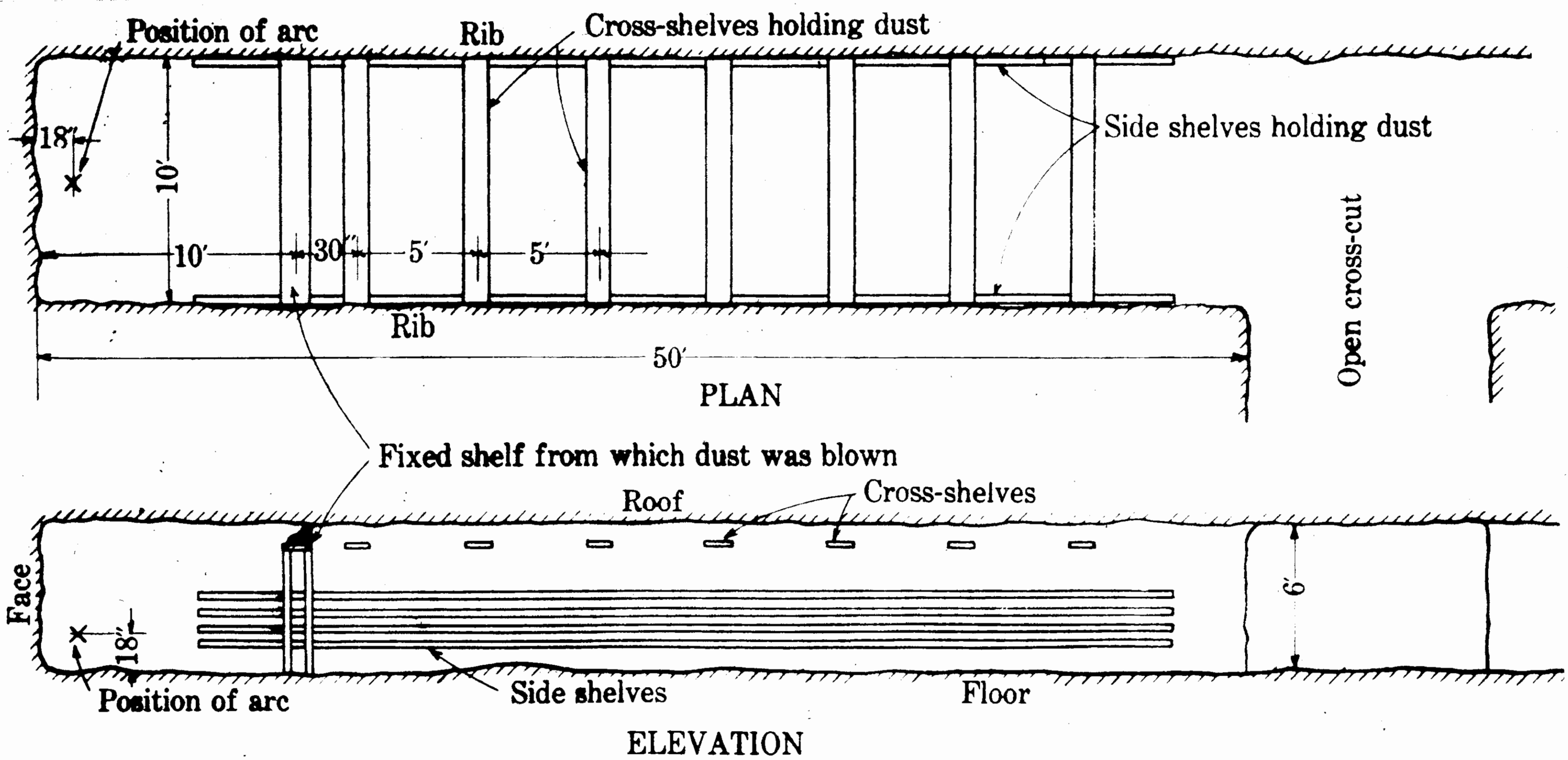


FIGURE 14

From United States Bureau of Mines R.I. 3044, 1930.

dead end with the shelves upon which the dust is placed; the shelf and arc being the same as in Figure 13 but with the arc nearer to the floor. The valve on the air line is opened by energizing a solenoid circuit in proper synchronism with the operation of the pressure and flame velocity recorders which are used in the tests. The Pittsburgh coal dust used was ground so that 90 percent passed through a 200-mesh sieve.

Two types of tests were made: (1) those in which there was dust only on the shelf carrying the air pipe, and (2) those in which additional dust was placed on the other shelves to determine if the initial inflammation could raise it and develop a self-sustaining explosion.

In tests of type (1), various amount of coal dust were blown from the shelf and the extent of flame and the pressure developed were determined. Table I gives the results of the tests.

TABLE I

Test	Quantity of Dust, pounds	Length of Flame, feet	Pressure, pounds per square inch		Time taken for flame to travel 50 feet, seconds
			50 feet from arc	150 feet from arc	
1	100	75	2.5	6.0	0.42
2	50	75	1.3	5.5	0.22
3	25	50	---	6.3	0.17
4	10	50	---	---	----
5	10	35	0.5	1.5	----
6	10	50	0.8	2.4	1.05
7	5	20	0.1	0.5	----

A reduction of the quantity of dust from 100 to 50 pounds caused no decrease in length of flame; only a slight decrease in pressure and an increase in the velocity of flame travel. This indicated that the larger quantity produced too dense a cloud and that part of the coal dust could not be consumed thus retarding, by heat absorption, the combustion of that which was inflamed.

In tests of type (2), studies were made to determine whether or not a self-sustaining explosion could develop from the inflammation of 10 pounds of coal dust. In these tests, 50 pounds of additional coal dust was placed on the cross shelves from the arc to the cross cut. These shelves were so arranged on a narrow support that a small force would upset them and thus spill the dust. In these cases, the flames extended about 35 feet beyond the last shelf and the pressures were twice as great as in the previous tests.

In another test, 100 pounds of coal dust was used besides 10 pounds blown into the arc. It was distributed on the side shelves and floor to the open cross cut and on the cross and side shelves and on the floor in the cross cut. A well developed explosion resulted. Flame extended at least 100 feet beyond the ends of the dust loading in both entries and pressures of five to nine pounds per square inch were recorded. These tests show that a wide-spread disaster may result when as little as 10 pounds of fine dry bituminous coal dust is ignited by an arc in a mine.

Data and Results from the Study

The dusts which were used in these tests were prepared from coal seams of different parts of the country and ranging from lignite to meta-anthracite. The samples are listed in Table II which gives the source classification, proximate analysis, heat content, volatile ratio, and the volatile content on a dry and mineral-matter-free (DMM-free) basis.

TABLE II

CLASSIFICATION AND ANALYSIS OF COAL SAMPLES (26, p. 3)

Classification of Coal	Proximate analysis percent: Moisture	Volatile matter	Fixed carbon	Ash	Heat value, BTU/lb.	Volatile ratio, percent $\frac{V}{V + F.C.}$	Volatile matter (DMM-free), percent
Meta-anthracite	0.2	3.0	64.1	32.7	8,720	4.5	2.0
Anthracite	4.1	4.0	82.5	9.4	12,830	4.6	3.6
Semi-anthracite	2.1	8.2	80.4	9.3	13,600	9.3	8.4
Low-volatile	3.6	16.5	72.8	7.1	13,960	18.5	17.8
Medium-volatile	2.7	24.7	65.9	6.7	14,050	27.2	26.7
High-volatile	1.2	34.8	56.0	8.0	13,770	38.3	37.8
do.	3.0	40.5	53.7	2.8	14,060	43.0	42.8
Sub-bituminous	10.7	34.6	46.3	8.4	11,020	42.8	42.2
Lignite	39.9	27.0	30.0	3.1	6,800	47.3	47.0

The analyses indicate that the moisture contents of the samples ranged from 0.2 to 39.9 percent. However, except in special experiments on the effect of moisture, no dust had more than three percent moisture when the explosibility tests were performed. Several samples were dried at 75 degrees Centigrade to reduce their moisture content.

Most studies were made on coal dusts of through-200-mesh (minus 74-micron-square)fineness. Some experiments were conducted with so called mine-size dusts in which all particles were finer than 20-mesh and 20 percent by weight passed through a 200-mesh sieve. In tests on the effect of particle size, samples composed of definite sieve fractions were used. That is, mesh sizes ranging from 48- to 65-, 65- to 100-, 100- to 150-, 150- to 200-, 200- to 270-, 270- to 325-, and through 325- were used. In another set of experiments, the effect of varying proportions of through 200-mesh particles in the samples was studied. In this case, fine dust was added to coarse aggregates consisting of particles between 20- and 200-mesh.

The amount of dust used in most tests was one gram or less. In the explosion-quenching and pressure-release experiments conducted in the galleries, samples of 10 grams to a few hundred grams are dispersed and exploded. Most experiments were performed in air but in some tests the dust was dispersed and ignited in oxygen. To determine the limiting percentage of oxygen that will support combustion, mixtures of air and CO_2 were used in a few tests.

The inert dust used to determine the relative inflammability of coal dust in the laboratory is Fuller's Earth in which all particles were finer than 20-mesh, 44 percent finer than 100-mesh, and 30 percent by weight finer than 200-mesh. The relative inflammability of a dust is defined as the percentage of inert dust in a mixture with the combustible dust that is required to prevent ignition of the mixture by a given igniting source. The results obtained from the above tests are given in the form of graphs.

Figure 15 shows the incombustible or rock dust requirements for preventing flame propagation in dust clouds of various ranks of coals. It may be seen that the incombustible requirement is less than 50 percent for coal dusts of volatile ratios below 0.20 and the value becomes 65 percent or more only at volatile ratios of about 0.30 to 0.35. Regulations in the United States require a minimum of 65 percent incombustible in all mines where the volatile ratio of coal is above 0.12.

Figure 16 shows the effect of volatile content on the lower explosive limit, minimum igniting energy, and ignition temperatures of coal dusts. The curves show that the ignition sensitivity increases and the lower explosive limit decreases with increased volatile content. In oxygen this effect becomes less important above a volatile content of about 20 percent.

Figure 17 shows the effect of volatile content on the maximum pressures and rates of pressure rise developed by explosions of dust clouds in a closed bomb. Here the rates above 15,000-20,000 lb. per square inch per second are approximations. The curves show that in air the pressure and speed of the explosions increase with increase in volatile matter whereas in oxygen there is little change in intensity

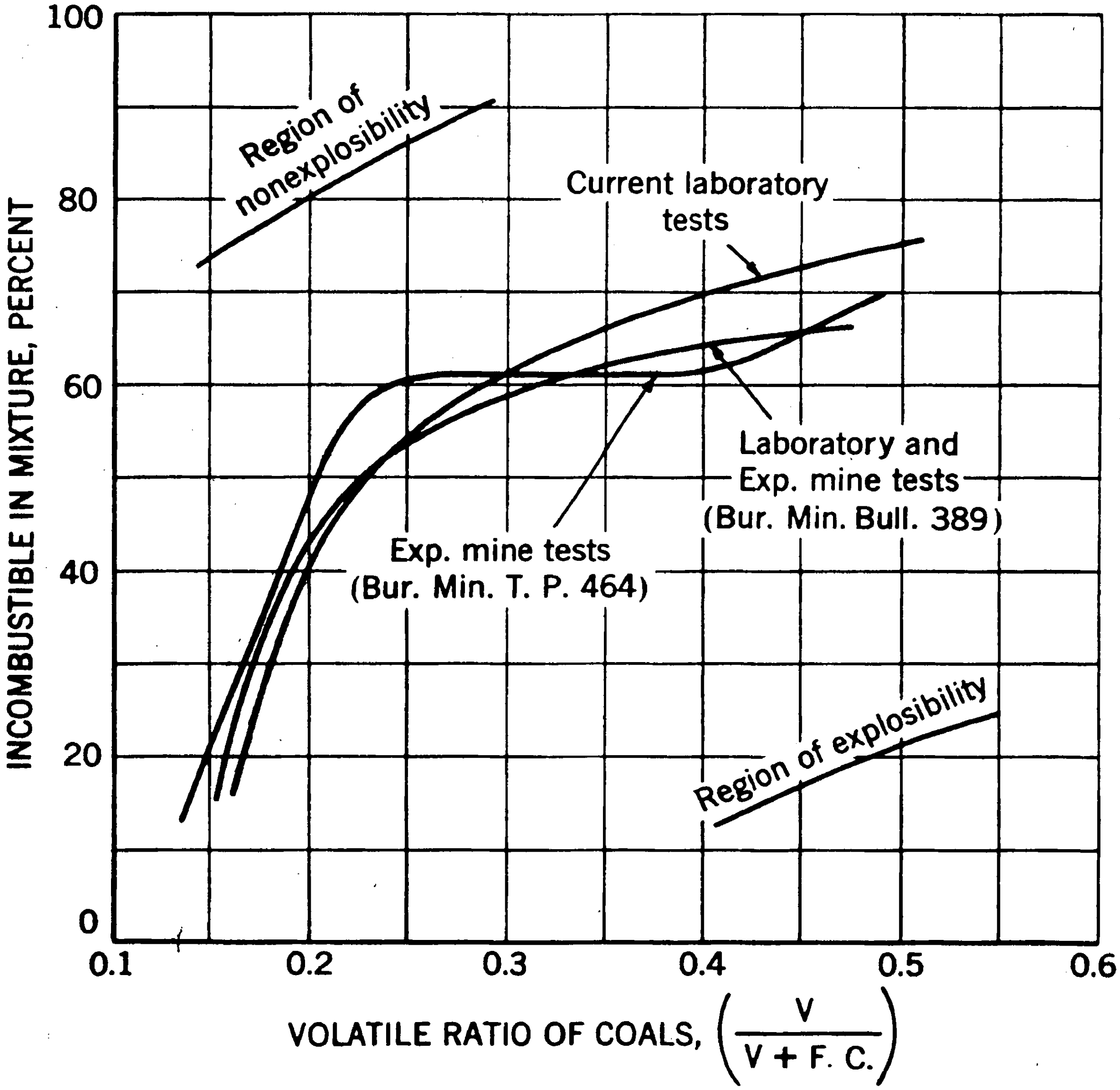


FIGURE 15

From United States Bureau of Mines R.I. 5052, 1954.

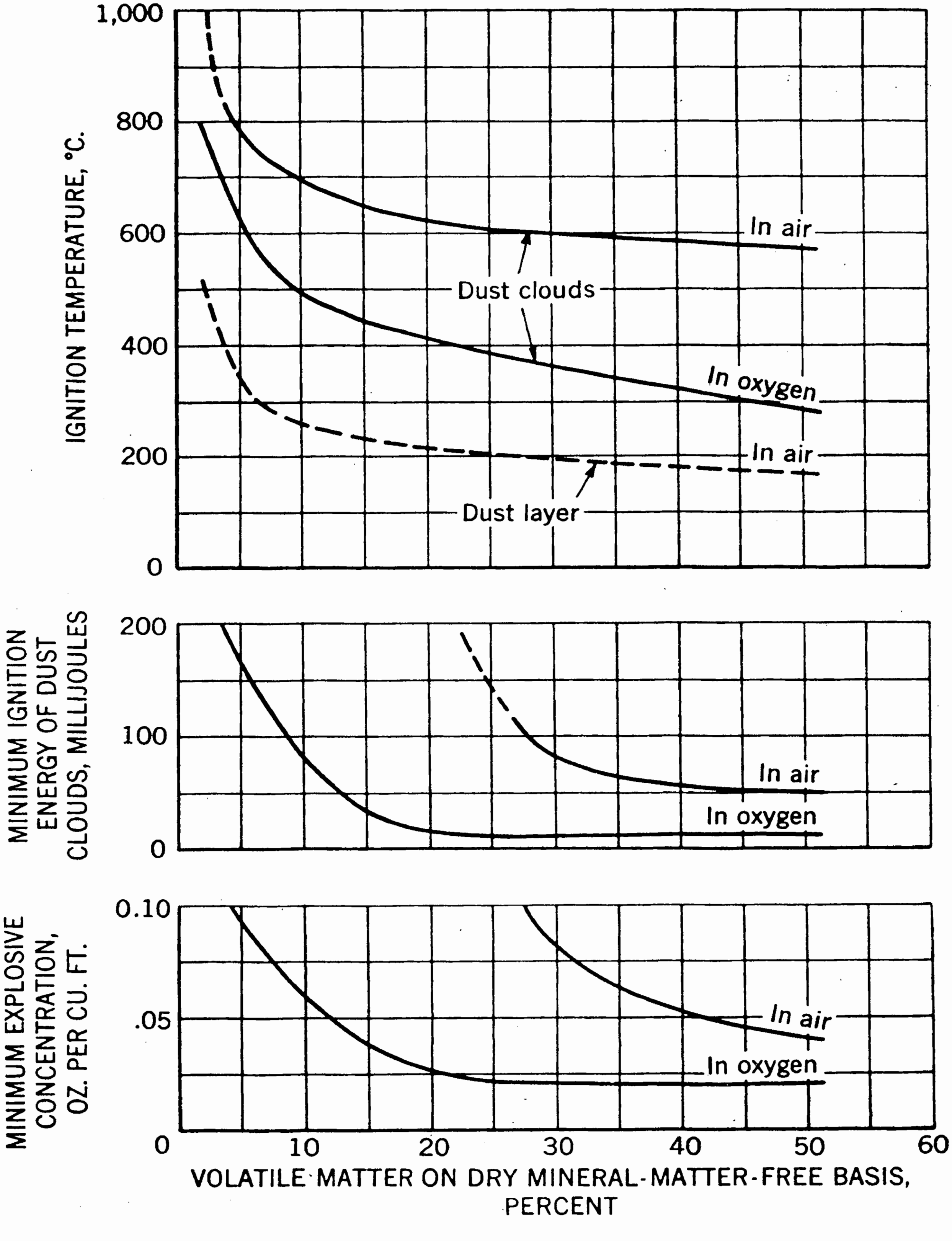


FIGURE 16

From United States Bureau of Mines R.I. 5052, 1954.

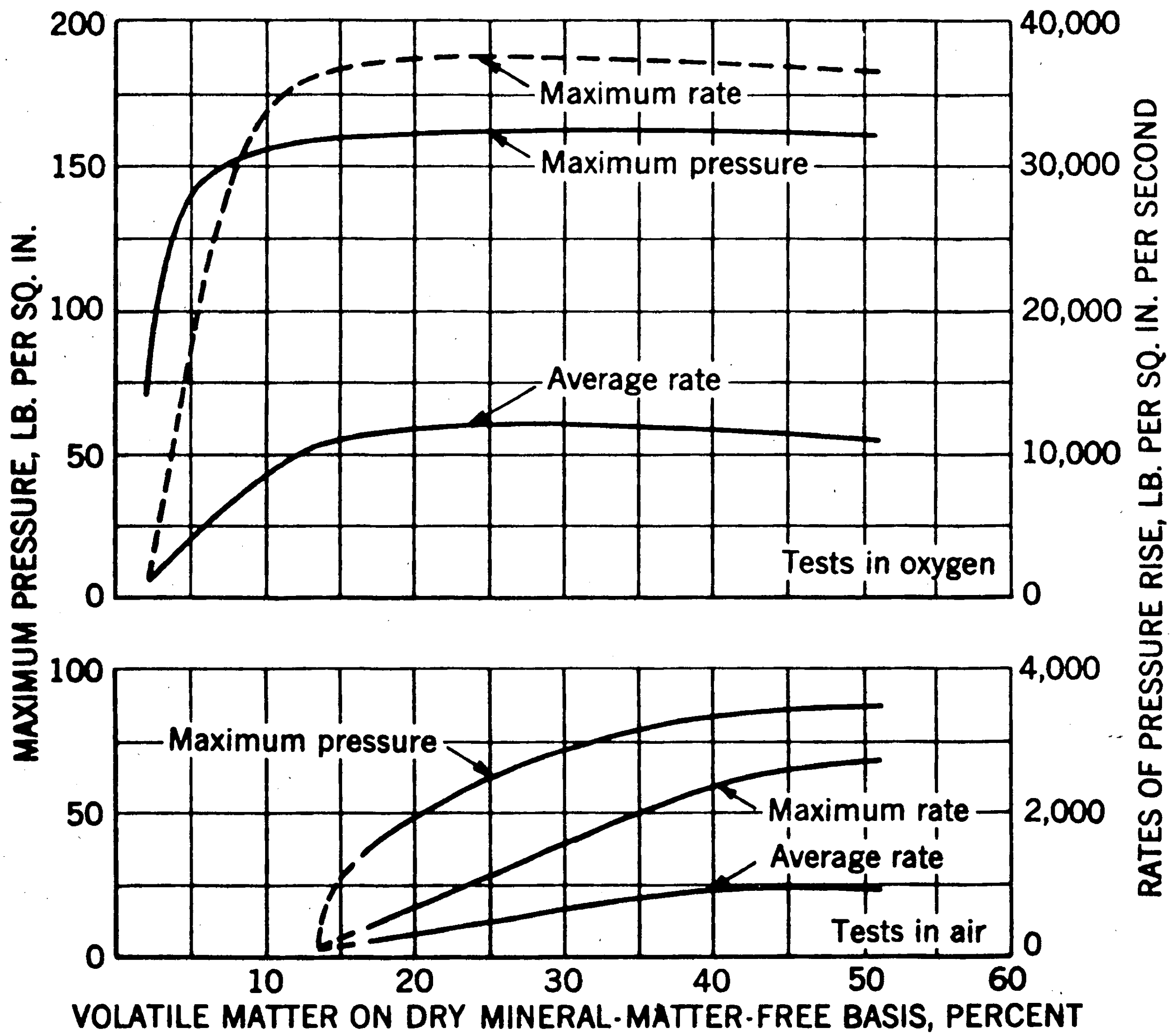


FIGURE 17

From United States Bureau of Mines R.I. 5052, 1954.

above 20 percent volatile matter.

Figure 18 shows the effect of dust concentration on the pressures and rates of pressure rise which were developed by explosions of fine bituminous coal dusts. Data for the other dusts gave curves of similar shape. The maximum pressures and rise rates in the air tests attained maximum values at concentrations of 0.05 to 1.00 ounces per cubic foot. Explosion tests in oxygen could not be performed at concentrations above 1.00 ounce per cubic foot; the limit being set by the strength of the test equipment.

The effect of initial pressure on explosions was studied with one bituminous coal dust. Pressure tests were made at concentrations ranging from 0.10 to 2.00 ounces per cubic foot. In all cases, the maximum explosion pressures and the rate of pressure rise increased linearly with the initial pressure. In one case, at initial pressures of 0, 10, and 20 p.s.i., the explosion pressures were determined to be 73, 99, and 125 p.s.i.; the average rates were 2,100, 2,700, and 3,350 p.s.i. per second; and the maximum rates were 4,300, 5,700, and 7,100 p.s.i. per second, respectively.

Figures 19 and 20 show the effects of particle size on various explosive characteristics. Increase in particle size reduces ignition sensitivity and explosion pressure. Dust clouds of fractions coarser than 150- to 200-mesh could not be ignited in air by electric sparks as shown in curves in Figures 6 and 7.

Figure 20 shows that the ignition temperature of dust clouds is fairly constant as the particle size increases to about 180 microns

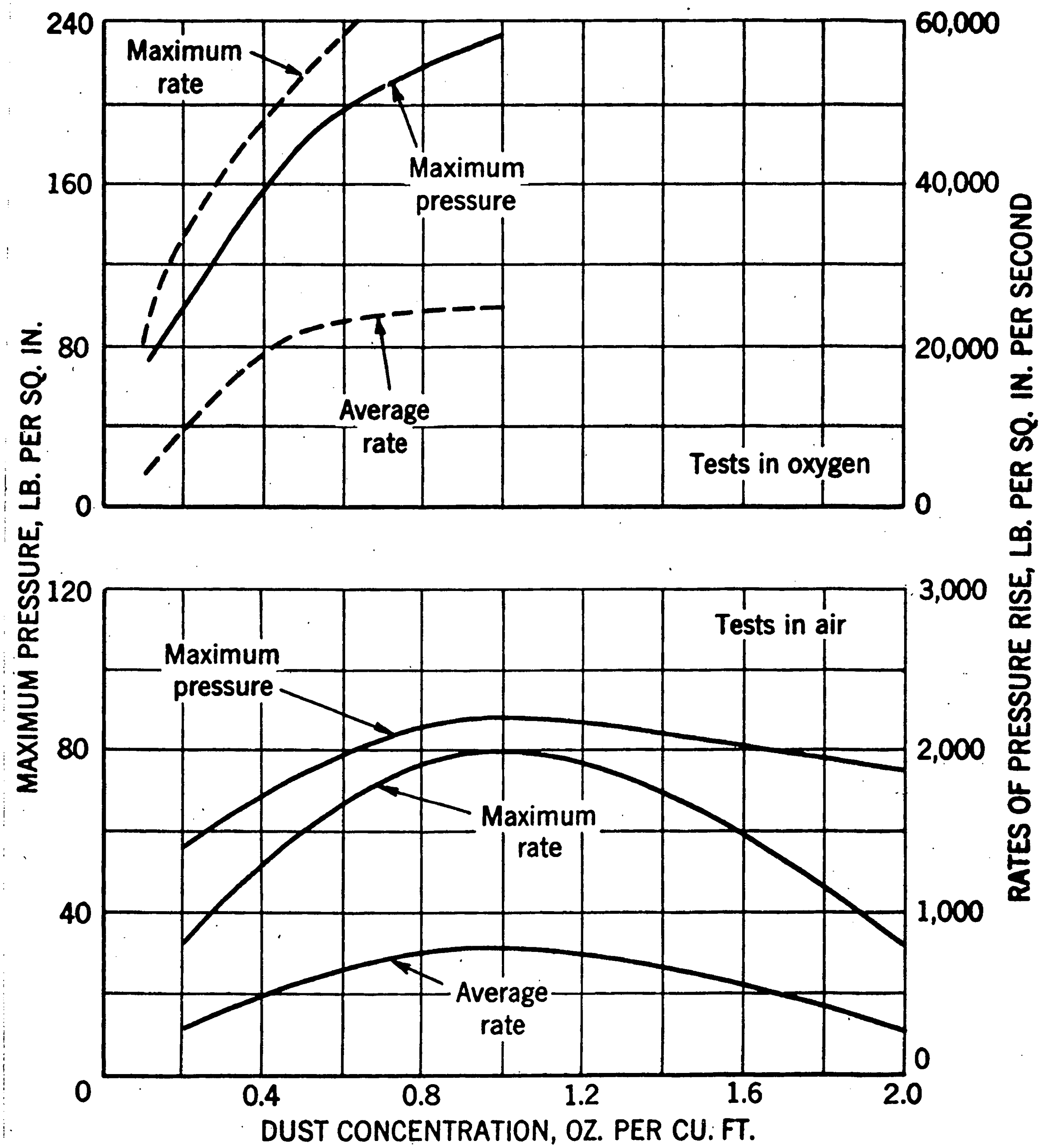


FIGURE 18

From United States Bureau of Mines R.I. 4195, 1948.

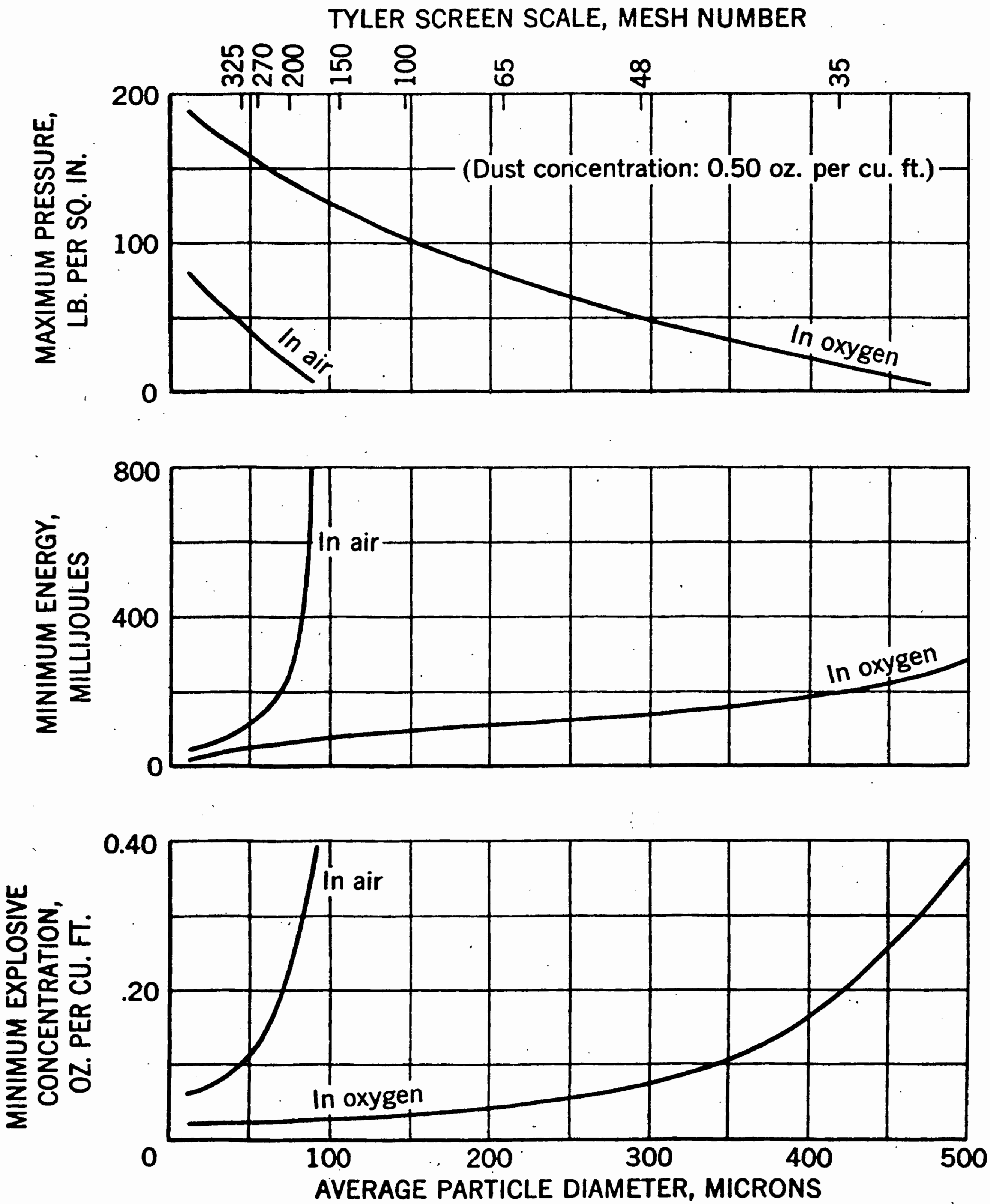


FIGURE 19

From United States Bureau of Mines R.I. 4195, 1948.

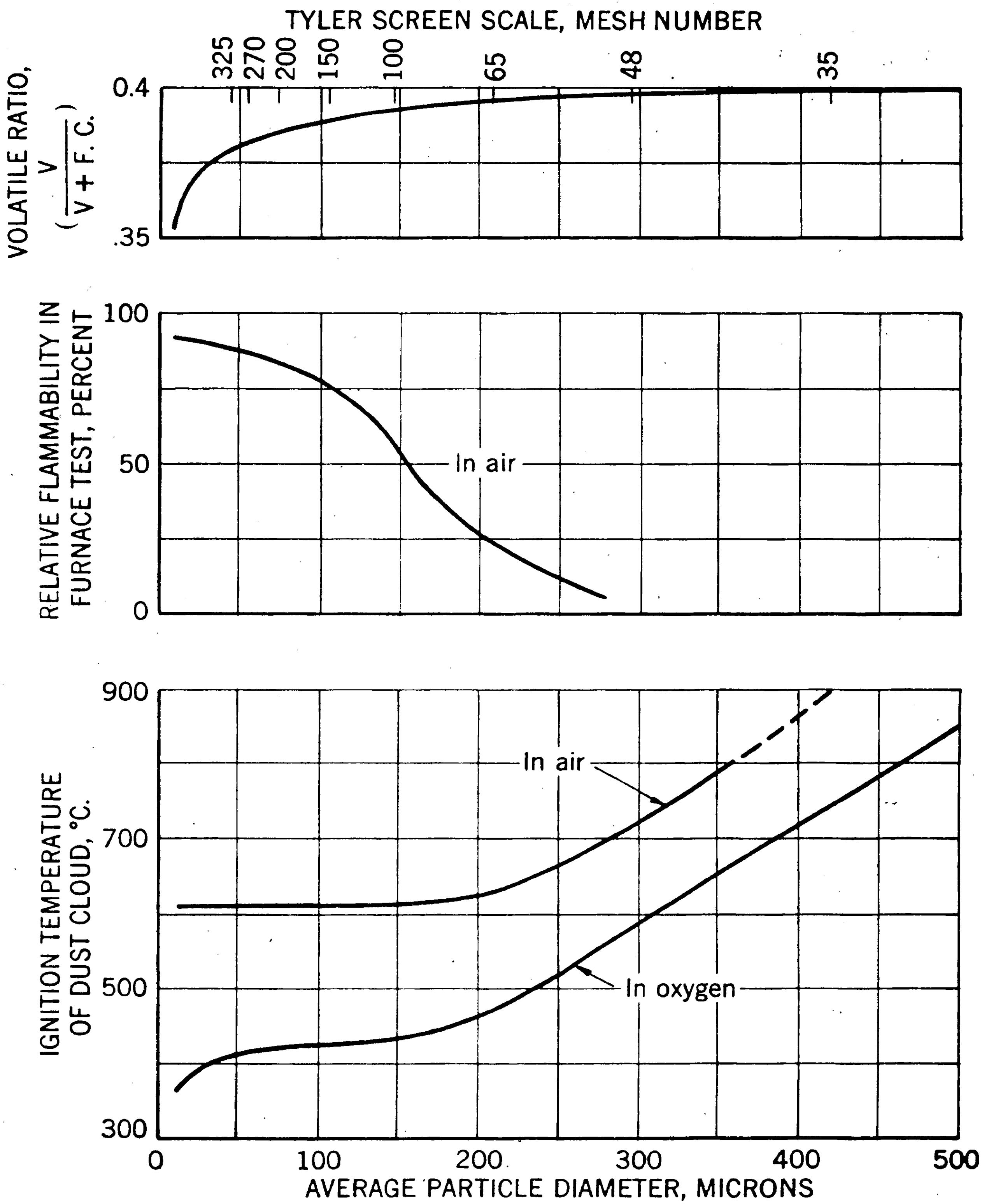


FIGURE 20

From United States Bureau of Mines R.I. 4195, 1948.

(65- to 100-mesh); this being about 200 degrees Centigrade lower in oxygen than in air. Any further increase in size results in rapid rise in the ignition temperature.

Figure 21 shows the effect of various proportions of fine dust particles on the explosibility of Pittsburgh coal in air. The two lower curves indicate that the addition of fines, up to 30 to 40 percent of the total by weight, results in a very significant reduction in the minimum explosive concentration and the minimum igniting energy. Any further increase has a much lesser effect.

Figure 22 shows the effect of moisture on explosibility of coal dust. For fine Pittsburgh coal dust there is little change in the lower explosive limit up to about eight percent moisture, but beyond that the required dust concentration increases sharply.

Figure 23 shows the relationship of the temperature in the laboratory furnace to the relative flammability of mine-size Pittsburgh coal dust and the permissible oxygen content in the atmosphere for through 200-mesh dust. The results obtained indicate the following limiting oxygen contents respectively for spark tests and furnace tests; (a) semi-anthracite 1 and 11.5, (b) low-volatile 1 and 10.5, (c) medium-volatile 18.5 and 10.5, (d) high-volatile 17.0 and 10.5, and (e) lignite 15.0 and 7.5.

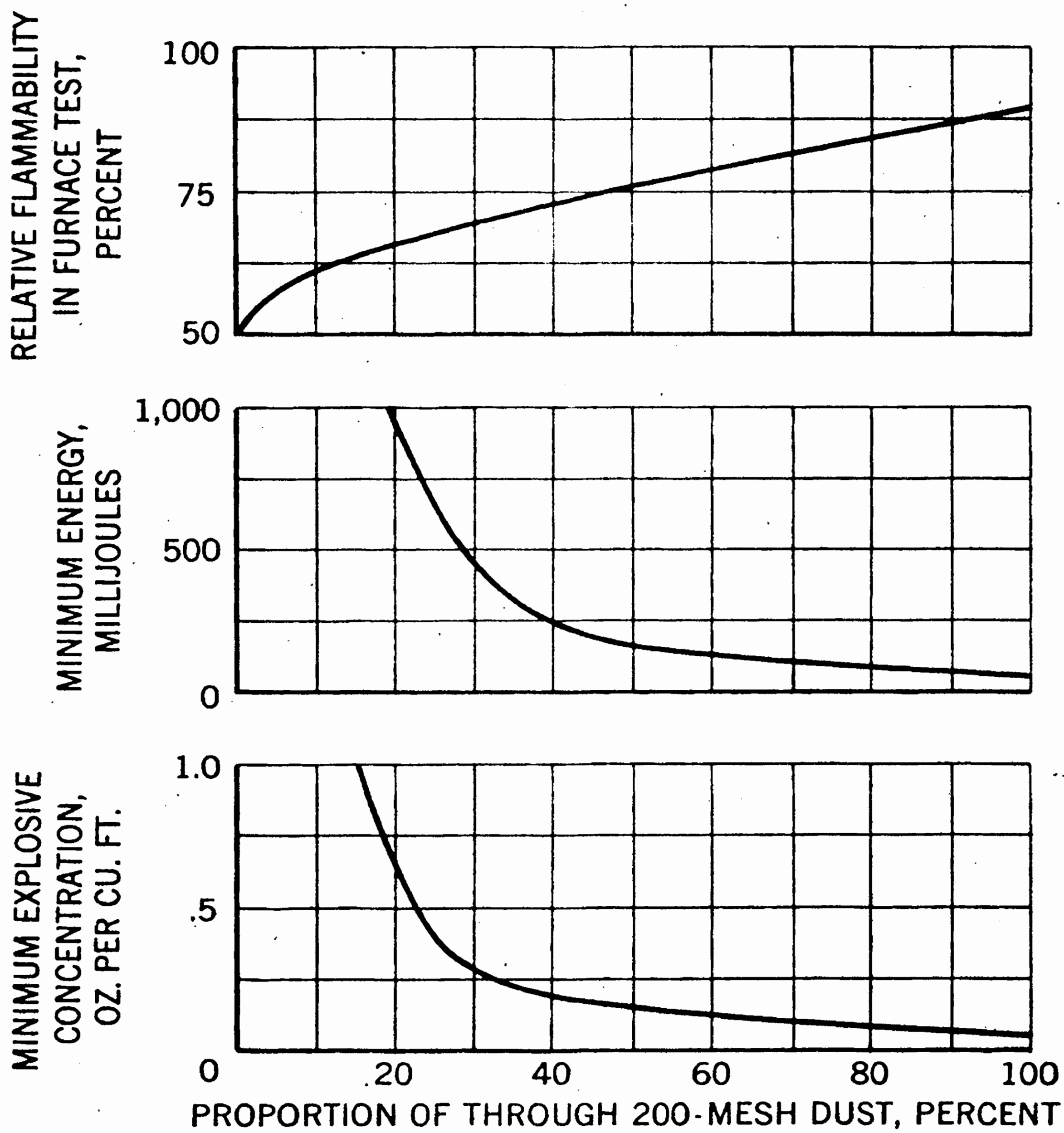


FIGURE 21

From United States Bureau of Mines R.I. 5052, 1954.

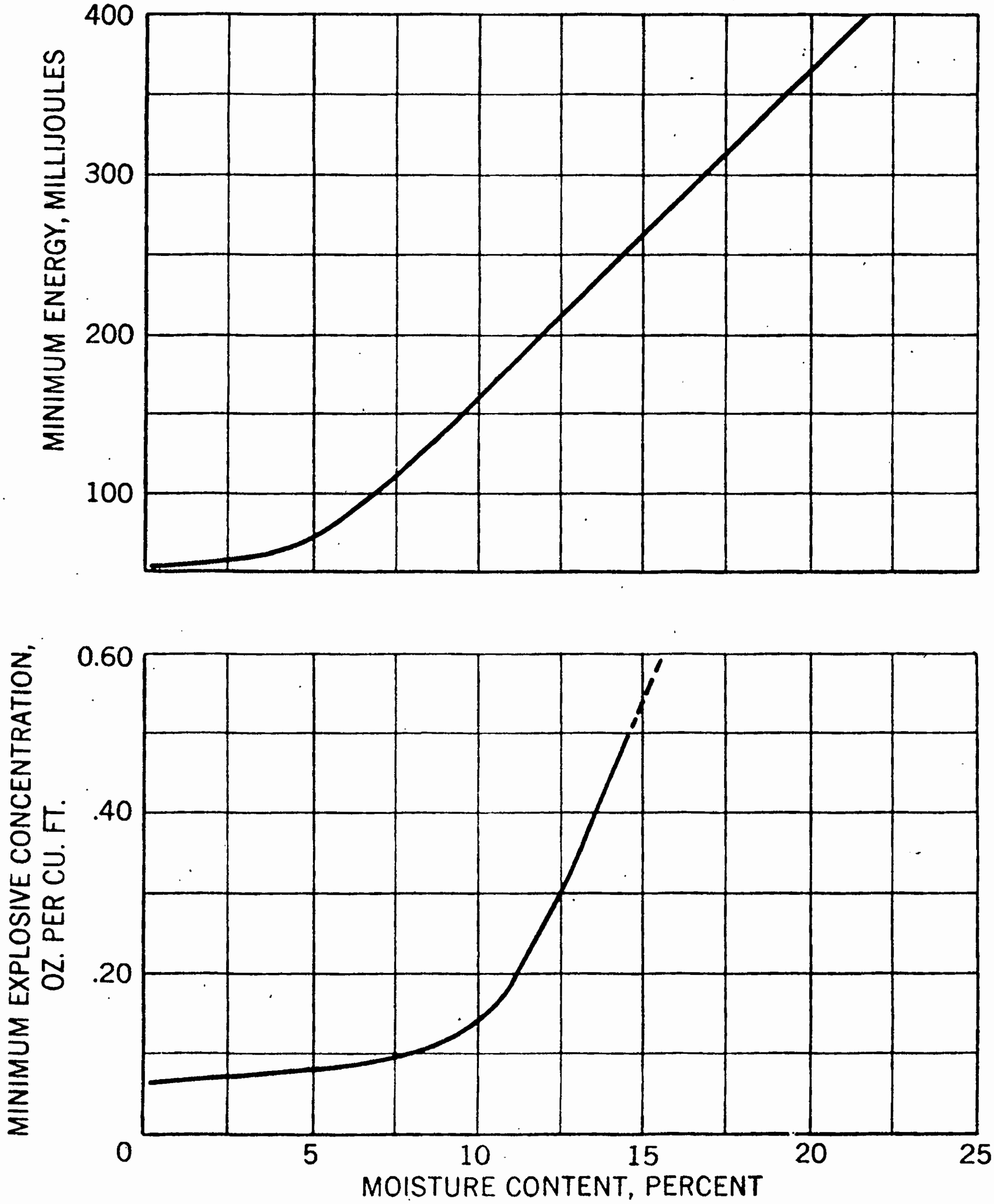


FIGURE 22

From United States Bureau of Mines R.I. 5052, 1954.

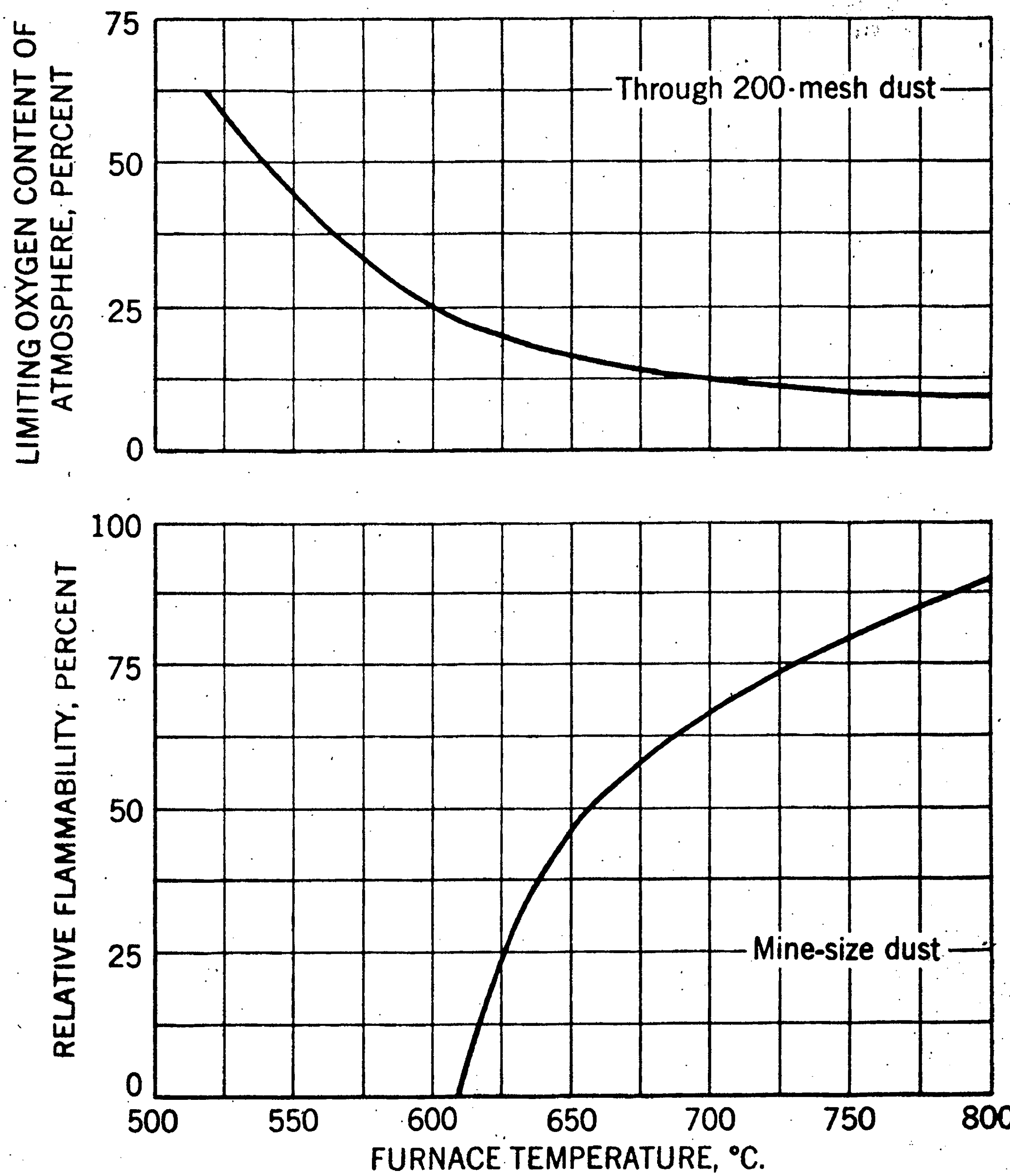


FIGURE 23

From United States Bureau of Mines R.I. 5052, 1954.

Some effort was made to develop an experimental method for evaluating the upper explosive limits of dust clouds. Although an entirely satisfactory method has not yet been found, approximate data indicate that fine bituminous Pittsburgh coal dust has an upper limit of about 12 ounces per cubic foot.

CHAPTER IV

PREVENTING COAL DUST EXPLOSIONS

General

In days past, to a considerable extent, there had been a belief that explosions were a mining hazard which could not be prevented. Such belief is based on the human failure to observe necessary safety precautions. There is no logical reason to think that mine explosions can not be prevented when proper preventive measures are taken. In general, this will involve the prevention of coal dust formation, prevention of dust dispersion to form a cloud, prevention of ignition of the dust, and as a final step, prevention of explosive propagation.

<u>Prevention of Formation of Coal Dust</u>.

It is a recognized fact that, in the coal mining industry, coal dust must be allayed in order to promote the safety and health of the workers. The use of water provides the principle means of dealing with the dust-abatement problems. In the earliest attempts, it was common to sprinkle road beds from water cars. Water was manually thrown from the tank car with buckets or allowed to run by gravity into the center of the road bed. Later, it was gravity sprinkled through perforated pipes and, finally, the tank car method was improved by pumps which were used to develop enough pressure to sprinkle the roof and ribs as well as the floor. The tank cars were then replaced by pipe lines with connections for fixed sprays or hoses to thoroughly wet all surfaces of the roadways.

The quantity of dust that is raised into the air during coal under-cutting operations ranges from 65.5 to 163.9 million particles per cubic foot of air and averages 120.9 particles for the dry method. (23, p. 5). When water is used on the mining machines in the same working places, the dust ranges from 3.6 to 42.9 and averages 21.8 million particles per cubic foot of air. If the dust which is raised during wet under-cutting is taken as 100 percent, the average without water is 554.6 percent. That is, the operation becomes 5½ times as dusty.

Water at the face. Water can be used at the face to wet coal before and after blasting and during loading. Arrangements can also be made for cutting and drilling machines to wet the coal cuttings produced. Water reaches the working faces at pressures ranging from 20 to 150 pounds per square inch. A half-inch pipe with a valve and a 50-foot length of half-inch hose are connected to machines when necessary for wetting before and after blasting as well as during loading. The arrangement of the water pipes and the method of discharging the water vary.

One method of equipping the machines with water pipes is to bend the half-inch pipe to clamp across the front of the motor casing. A water hose is connected to the machine and water discharges through 1/8-inch holes on the chain and bar at the front end of the machine. Arrangements are shown in Figures 24, 25, and 26. (23, p. 10).

Various other methods are possible, but the general plan is the same in all cases; only the details and points of discharge of the water

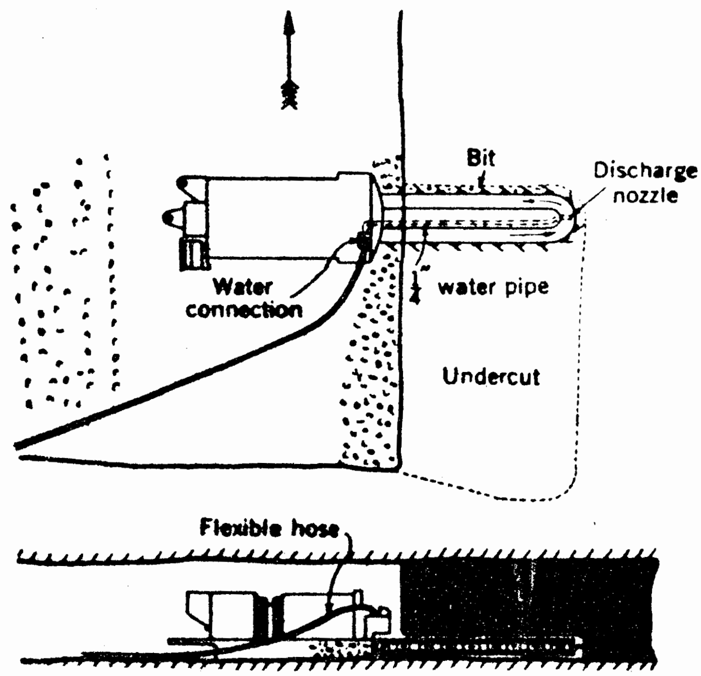


FIGURE 24

From United States Bureau of Mines Technical paper 593, 1939.

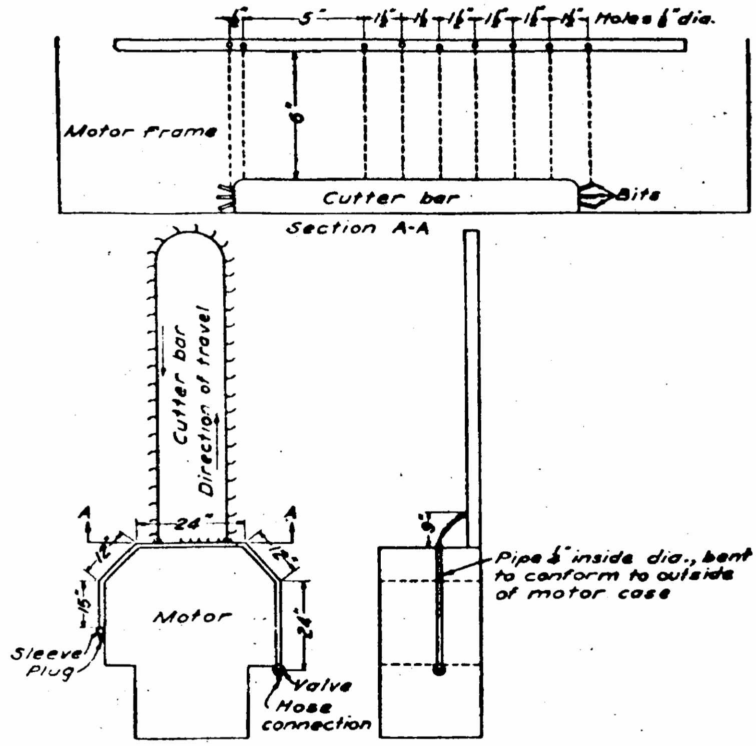


FIGURE 3.—Water line on short-wall machine.

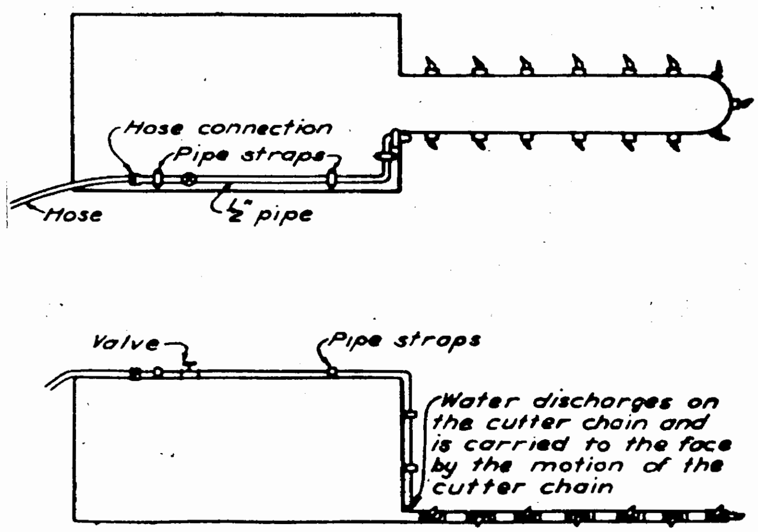


FIGURE 25

From United States Bureau of Mines
Technical paper 593, 1939.

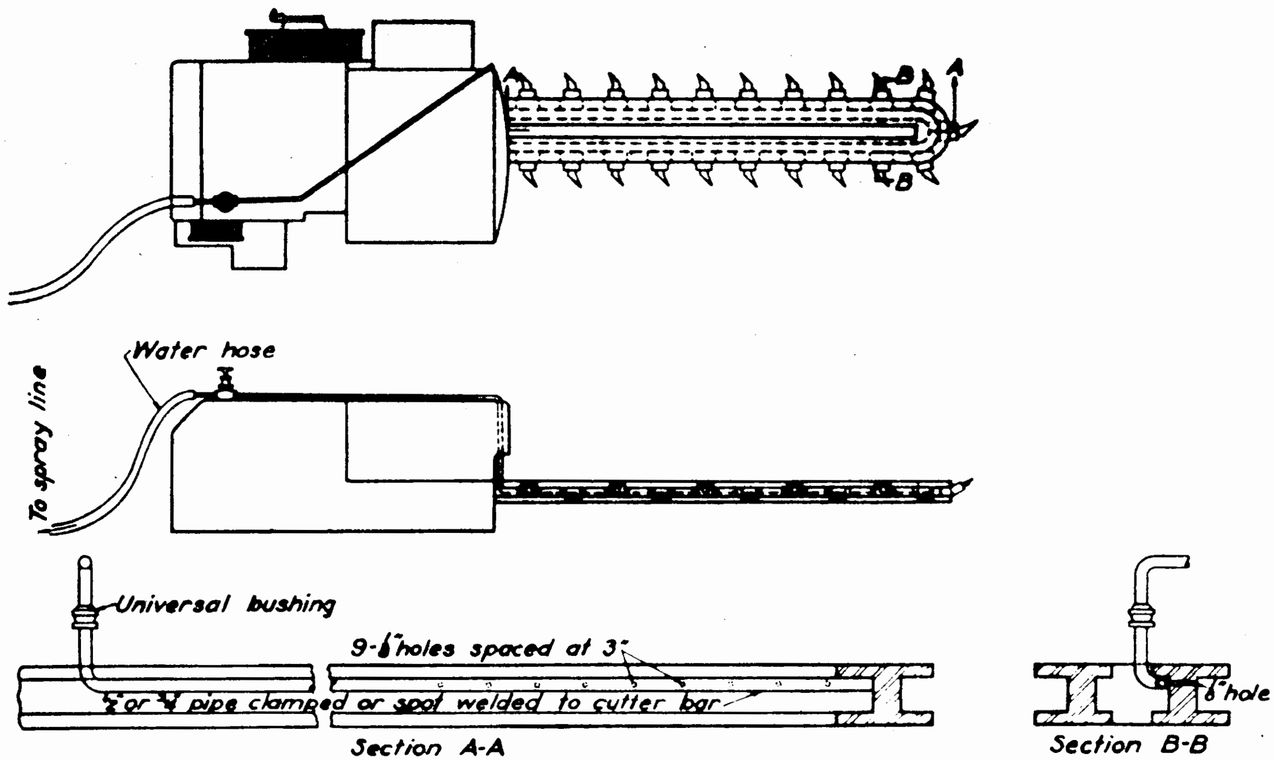


FIGURE 26

From United States Bureau of Mines Technical paper 593, 1939.

differ in each case. In all the methods shown, the cutting machine has a pipe over the top of the motor housing from where it extends inside of the cutter-bar to the cutter-bar head.

In all the above cases, varying numbers and sizes of jets and nozzles have been tried to spray water.

Case 1. Three jets were used, each having a 1/8-inch diameter bore. Two were set near the chain where it entered the cut and one where the chain emerged from the cut. Trouble arose from choking of the jets and, therefore, this size was not very popular.

Case 2. One jet of 1/4-inch diameter bore was used on the chain where it entered the cut. No troubles arose from choking the jet, but dust allayment was not fully attained. The loss of efficiency in dust reduction was accentuated in places where the speed of cutting was greatest. Even where the cutting speed was 2½-feet per minute, the pile of cuttings had large patches which were quite dry.

Case 3. Two jets each of 1/4-inch diameter bore were used. Jets were set one over the chain where it entered the cut and one over the chain where it emerged from the cut. No trouble arose from choked jets; and for all speeds of cutting, the whole of the gummings, both in the cut and in the cutter track, was uniformly damped throughout.

In the above tests the pressure of water was maintained constant. Tests show that the number, size, and arrangement of the jets in the manner described in case 3 is the arrangement to be recommended for under cutting.

The quantity of water required depends upon operating conditions and tests were made to determine the quantity of water necessary to give the desired results under various conditions. The data obtained are given in Table III.

TABLE III

TESTS FOR QUANTITIES OF WATER ON CUTTER-BARS, CALCULATED WEIGHTS IN TONS (23, p. 23)

Mine	Coal Thickness		Width of Under-cut		Depth of Under-cut		Height of Under-cut	Moisture	Coal Entirely Cut	Coal Under-cut	Wetted Cuttings	Water in Cuttings	Water/Ft. of Face
	Ft	In	Ft	In	Ft	In	In	%	Tons	Tons	Tons	Gal.	Gal.
A	4	11	41	7	5	7	4	12.93	47.09	3.19	3.53	81.39	1.95
A	4	11½	39	0	5	8	5	17.45	45.20	3.80	4.43	88.56	2.38
B	5	6	35	0	7	6	4	15.70	59.55	3.61	4.13	124.48	3.56
B	4	9½	35	0	7	3½	5	13.17	50.43	4.38	4.86	114.91	3.29
C	7	5	22	0	7	0	5	4.14	47.49	2.69	2.72	7.18	0.33
C	7	9	24	0	7	6	5	5.88	58.59	3.15	3.24	21.54	0.89
D	3	10	30	0	5	0	5	15.54	23.72	2.58	2.95	88.56	2.95
D	3	1½	32	0	5	2	5¼	13.57	21.31	2.98	3.33	83.66	2.61

Water consumption tests with wet undercutting using various numbers and sizes of jets were made and the results are given in Table IV. (8, p. 758).

TABLE IV

WATER CONSUMPTION TESTS WITH WET UNDERCUTTING

Yards Cut	Cutting Speed ft/min	Water Used in gallons	Water Used in gal/yd	Jets Used No.	Jets Used Size Inch	Condition of Gummings
80	4.4	396	4.95	1	7/32	Mainly damp with dry patches
96	4.1	511	5.35	1	7/32	Mainly damp with dry patches
100	4.1	548	5.48	1	7/32	Mainly damp with dry patches
120	5.7	176	1.47	2	3/16	Much dust in gummings
30	6.4	129	4.30	2	3/16	Very little dust
70	3.8	322	4.60	2	3/16	Very little dust

<u>Allaying dust during operation of continuous-mining machines</u>. A high pressure triplex plunger pump that created a static pressure of 800 pounds per square inch at no delivery was used to develop high hydraulic pressures to force water or solutions of water mixed with wetting compound to the continuous-mining machine at desired pressures. The pump was driven by a 5-H.P., 500-volt, direct-current motor; and the pump and motor were assembled into one working unit.

The pump was placed along the entry at a water sump about 300 feet from the machine. Water was delivered from the pump to within 50 feet of the machine through a 3/4-inch pipeline, and then to the machine through a 3/4-inch rubber hose. Water was dispensed at the machine through seven spray nozzles, which were placed strategically and effectively on and near the cutting and conveyor head, as shown in Figure 27. Only five of the seven spray nozzles were placed on the right side of the cutting and conveyor head of the machine in positions corresponding with C and D nozzles on the left side of the machine. Table V shows the positions, manufacturer's numbers, and orifice diameters of the spray nozzles used.

TABLE V (45, p. 5)

Position of Spray Nozzles	Manufacturer's Designating Numbers for the Spray Nozzles	Diameter of the Orifices of the Spray Nozzles 64ths of an inch
A	No. 5	5/64
B	No. 5	5/64
C	No. 5	5/64
D	No. 4	4/64
E	No. 3	3/64
F	No. 6	6/64
G	No. 4	4/64

When the seven spray nozzles were operated at full capacity, the pressure created by the pump decreased from 800 to 300 pounds per square inch without any apparent decrease in the effectiveness of the spray nozzles. The effectiveness of water used alone during operation of the

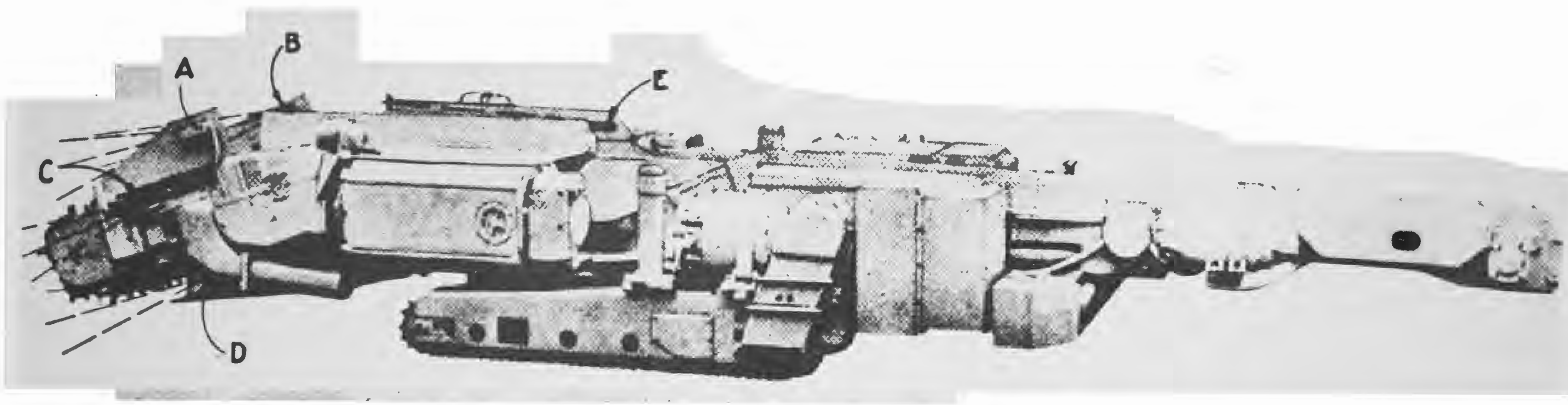


FIGURE 27

From United States Bureau of Mines I.C. 7608, 1951.

machine was satisfactory with the indicated arrangement of the spray nozzles, because the mist from the nozzles covered all outlets of coal dust from the machine. When a wetting compound was used with water, results improved very much and virtually all particles of coal dust were agglomerated or flocculated and settled to the floor at the cutting and conveyor head.

The wetting compound. Wetting compound, which is a balanced blend of wetting and dispersion chemicals, when added in the concentrated form in minute quantities to water, will cause the mixture to penetrate and allay coal dust by producing almost instant dispersion of the liquid over the dust particles. The wetting power of the compound-water solution is based not only on the reduction of surface tension of the solution but also upon the reduction of the interfacial tension between the solid and the liquid.

The compound is available in the form of a solidified capsule or cartridge. It is a multiphase compound and has various trade names, one popular name is "PERMINAL W". One to three pounds of the compound are used with 1000 gallons of water. A solid cartridge of the material is placed upright in a steel receptacle, which is connected to the pipeline at the discharge end of the pump. Water from the discharge end of the pump is admitted to the bottom of the unit and around the sides of the cartridge, dissolving the wetting agent. Figure 28 shows a schematic sketch of the steel receptacle.

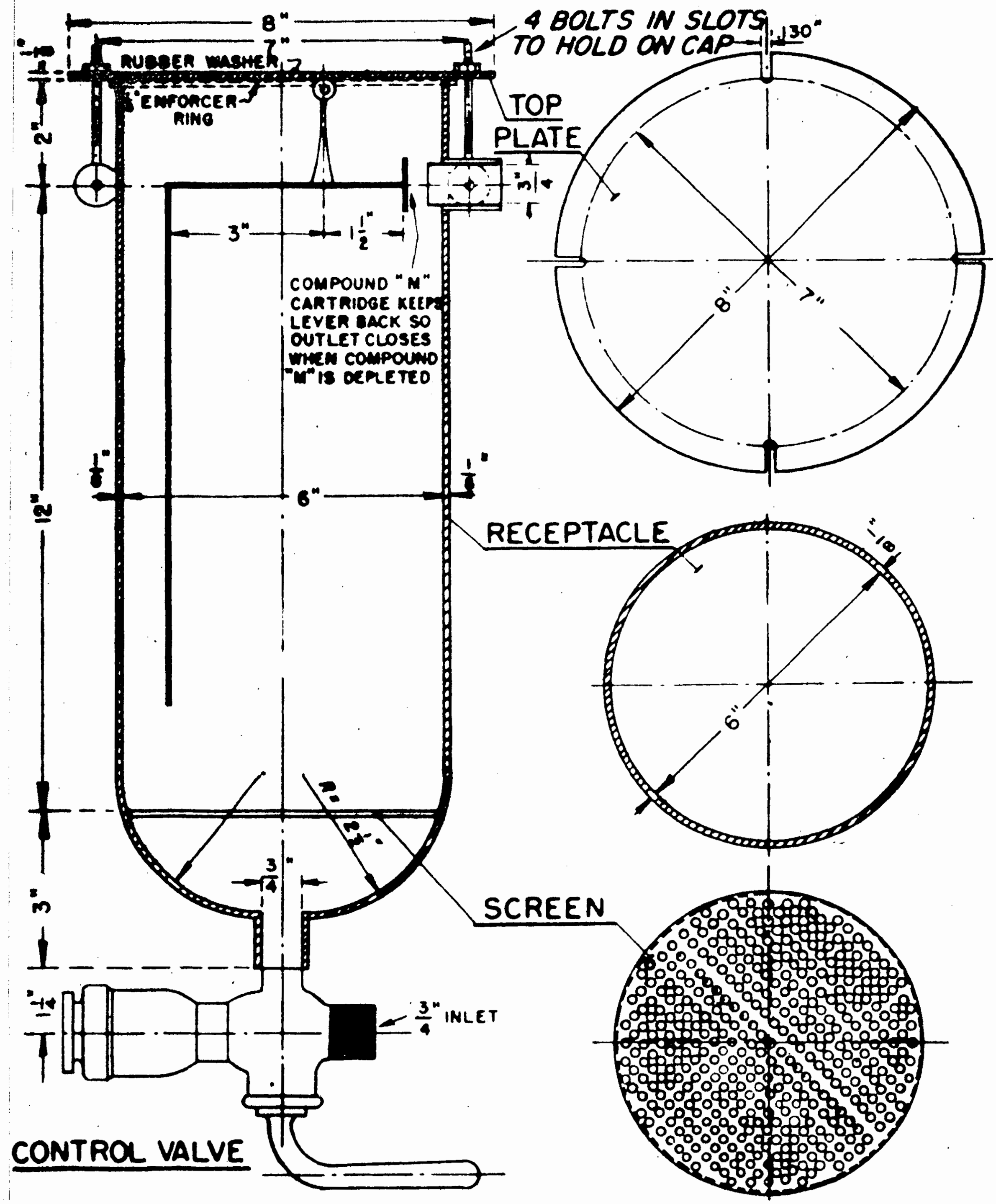


FIGURE 28

From United States Bureau of Mines I.C. 7608, 1951.

The pump mentioned discharges about 12.8 gallongs of water per minute at a pressure of 600 pounds per square inch. It is a positive displacement type and operates at a fairly constant speed.

The spray nozzles affixed to the continuous-mining machine contain swirl plates with seven holes. The apex spray angles formed by the spray nozzles containing seven plates are as follows:

No. 1 spray nozzles formed an apex spray angle of 0.0 degree.

No. 2 spray nozzles formed an apex spray angle of 12.5 degrees.

No. 3 spray nozzles formed an apex spray angle of 15.0 degrees.

No. 4 spray nozzles formed an apex spray angle of 20.0 degrees.

No. 5 spray nozzles formed an apex spray angle of 26.0 degrees.

No. 6 spray nozzles formed an apex spray angle of 31.0 degrees.

The design of the high pressure spray nozzle is shown in Figure 29.

Dust conditions during a test. At the beginning of a 24-hour test during three consecutive working shifts, a sheet of white paper 24 by 24 inches, which had absorbed moisture and was damp, was placed in the return-air course 30 feet outby the working face, in an upright position 10 degrees from horizontal and facing the direction of the return-air current from the working face where the machine was operating. A negligible amount of coal dust was deposited on the paper when water mixed with compound was used to allay the dust.

Air-borne dust samples were collected at places "A", "B", and "C", indicated in Figure 30. Three samples of air-borne dust were obtained while the machine was operated during each of the following arrangements:

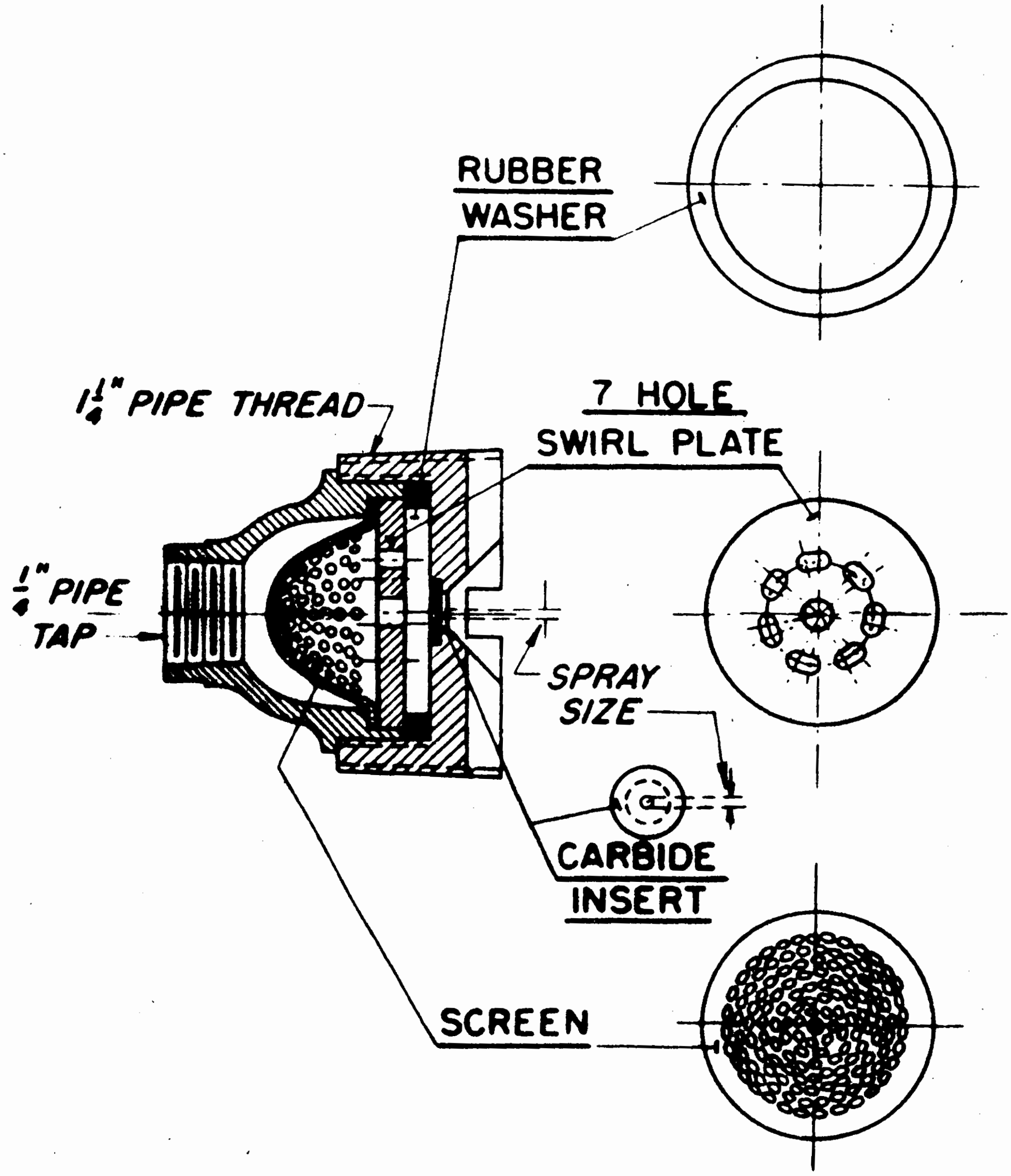


FIGURE 29

From United States Bureau of Mines I.C. 7608, 1951.

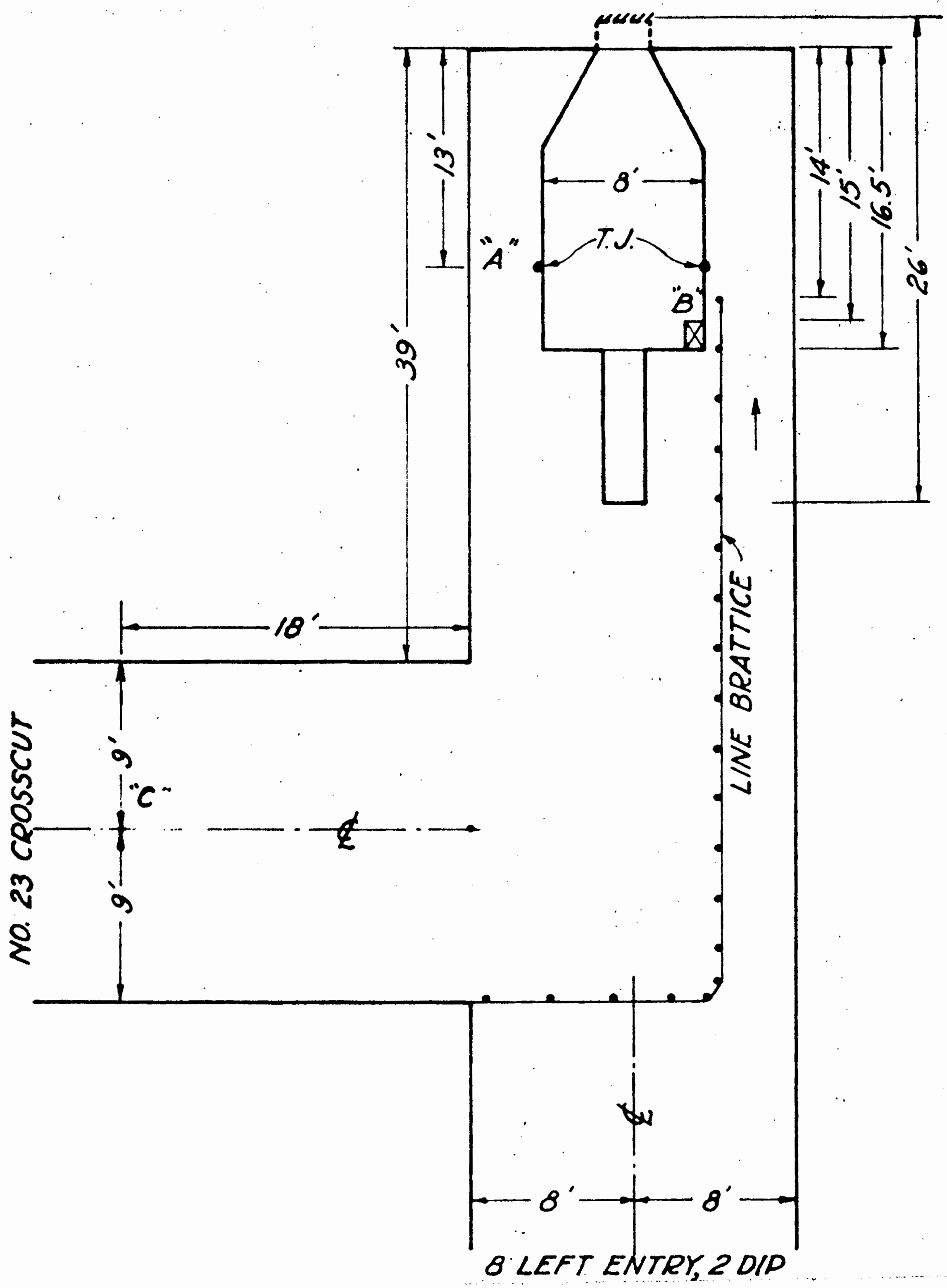


FIGURE 30

From United States Bureau of Mines I.C. 7608, 1951.

(a) without dust-allaying facilities, (b) with water alone, and (c) with a solution of water and wetting compound.

A volume of 11,485 cubic feet of air per minute was circulated at the end of the line brattice, 14 feet from the face of 8 left entry. The velocity of the air current was 296 feet per minute, and the cross-sectional area behind the line brattice was 38 square feet. Table VI gives the density of the air-borne dust samples mentioned above.

TABLE VI (45, p. 10)

DENSITY OF AIR-BORNE DUST SAMPLES COLLECTED DURING OPERATION OF CONTINUOUS-MINING MACHINE

Sample Number	Place Where Sample was Collected	Collection Time, Minutes	Without Dust-Allaying Facilities	Dust Allayed with Water	Dust Allayed with Wetting Compound
1	A	5	1,128,00	- -	- -
2	B	5	152,00	- -	- -
3	C	5	736,00	- -	- -
4	A	15	- -	18.98	- -
5	B	15	- -	5.88	- -
6	C	15	- -	7.40	- -
7	A	15	- -	- -	1.20
8	B	15	- -	- -	.30
9	C	15	- -	- -	1.62

The U. S. Bureau of Mines midget impinger dust sampling apparatus was used to collect air-borne coal dust samples during operation of the machine. Counting of dust was made by a midget microprojector and dust sizes of 10 microns or less in diameter were considered when making dust counts.

Tests and observations have indicated that in bituminous coal mines 40-million particles per cubic foot constitutes a dusty atmosphere; this figure is recommended as the maximum allowable concentration. (45, p. 8).

Water on haulageways. Standpipes for sprays for loaded cars are connected to the main-line water pipe, and a pipe across the track is drilled with 1/8 to 3/4-inch holes, two to four inches apart, and installed in room entries, partings, or side tracks where trips are gathered and from which they are pulled. These standpipes are controlled by valves.

Water Infusion System

Water infusion as a means of dust suppression is a technique by which dust can be allayed at the source, that is, on the face, for it is there that occurs the heaviest concentrations of men and the primary causes of an explosion. As with all processes, there must be some standard by which the efficiency of water infusion may be judged. This is provided by the arbitrary standards laid down by the National Coal Board, England. These standards, which should not be exceeded, are 850 particles per cubic centimeter (650 p.p.c.c. for anthracite) in the size range of 1 to 5 microns (one micron is 1/25,000th part of an inch) for coal and 450 p.p.c.c. in the size range of 0.5 to 5 microns for rock dust. (66, p. 755).

In introducing an infusion system, the factors to be considered are: (a) the porosity of the coal and the surrounding strata; (b) the presence of impervious beds; (c) the probable effect on roof and floor conditions; (d) the method of working, and (e) the rate of gas emission, if any.

The above factors influence the location, spacing and depth of the holes, the pressures and the quantities of water to be infused.

The conception of water flowing along a coal bed can be better understood by considering the pressur required to force water into the seam. When attempts are made to infuse a seam, three types of pressures are involved: (a) the static pressure of water available; (b) the open flow pressure, that is, the water pressure with the infusion tube open to the atmosphere; and (c) the infusion pressure, or the pressure needed to force water into the seam. Although it is true that the infusion pressure varies according to the position of the seal and also on the presence of breaks, both natural and induced, there is, in the absence of breaks, a figure which can be looked upon as typical of the seam.

Experiments were made to find out the most suitable time during the cycle of operations when infusion should take place. Twelve holes were drilled in a four foot thick seam to a depth of 4½ feet; each hole was infused for five minutes at a maintained pressure of 140 p.s.i. to 200 p.s.i. before the face was cut, and 13 to 16 gallons of water were allowed to enter each hole. Twelve similar holes were drilled in the same seam and the face was cut; when infusion was tried, not only was difficulty experienced in inserting the infusion tubes, but in no case did the infusion pressure exceed 10 p.s.i.; and water ran through the seam and appeared near the floor almost at once.

Influence of cracks and breaks. This property of resistance is the paramount characteristic around which a successful technique of deep infusion is developed. A rough experiment was made to ascertain the size of breaks which would reduce resistance, by welding a four inch diameter plate at right angles at the end of an infusion gun and securing a similar solid plate by three adjustable set screws to the first plate in such a manner that predetermined gaps could be made between the plates. A static water pressure of 260 pounds per square inch was available, but with a gap of 0.005 inch, water began to flow and the pressure was reduced by 100 p.s.i.

Variations in resistance. Experiments have shown that not only the resistance of one seam is different from that of another but that resistances are not constant in all horizons of the same seam. In one case, in a four foot thick seam, it was found that a hole drilled nine inches from the roof had a pressure 100 p.s.i. more than a similar hole drilled in the center of the seam.

Tests in the Experimental Mine

Figure 31 shows a typical face for infusion test. The coal face

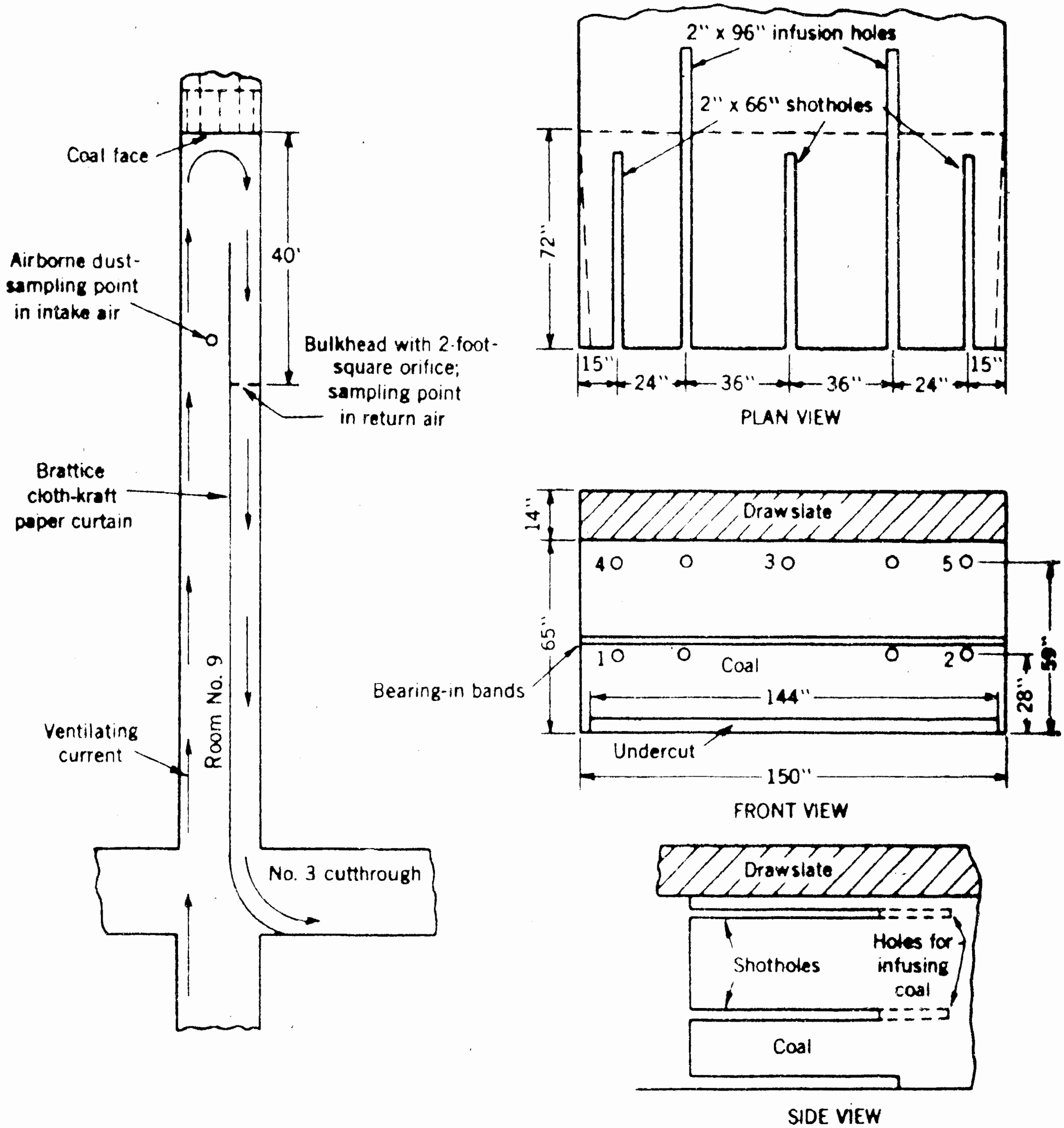


FIGURE 31

From United States Bureau of Mines R.I. 5353, 1957.

was undercut to a depth of six feet; in tests without infusion, coal faces were normally nine feet wide, and in tests with infusion, 12 feet. Arrangements of shot holes are shown in the plan and front view of the figure. For the infusion tests, four two-inch diameter horizontal infusion holes, generally eight feet long, were drilled in the face before undercutting. Water was injected simultaneously into the two bottom holes, then into the top two holes.

Injection was accomplished by a water-infusion seal shown in Figure 32. The quantity of water injected into the face per test ranged from 116 to 394 gallons. Water supplied from a tank car was maintained under pressures ranging from 70 to 100 p.s.i. The maximum water pressures in the infusion holes ranged from 18 to 98 p.s.i. The rate of water flow was 1.0 to 6.6 gallons per minute. The average volume of water injected into the coal face was 10 gallons per ton of coal mined. In some tests, a wetting agent was added to water in the proportion of one part per thousand, as recommended by the manufacturer.

The coal face was undercut 20 to 120 minutes after water was injected. Drilling the shot holes generally was accomplished the same day as undercutting; blasting and coal loading were performed on succeeding days. Permissible explosive was used to blast the shot holes. Approximately 11 tons of coal were mined from a 9 foot face and 15 tons from a 12 foot face.

Float dust samples were collected during each mining operation. Dust samples of 1 to 10 micron sizes were collected by midget impingers in the intake and return air. Sampling generally was begun one minute

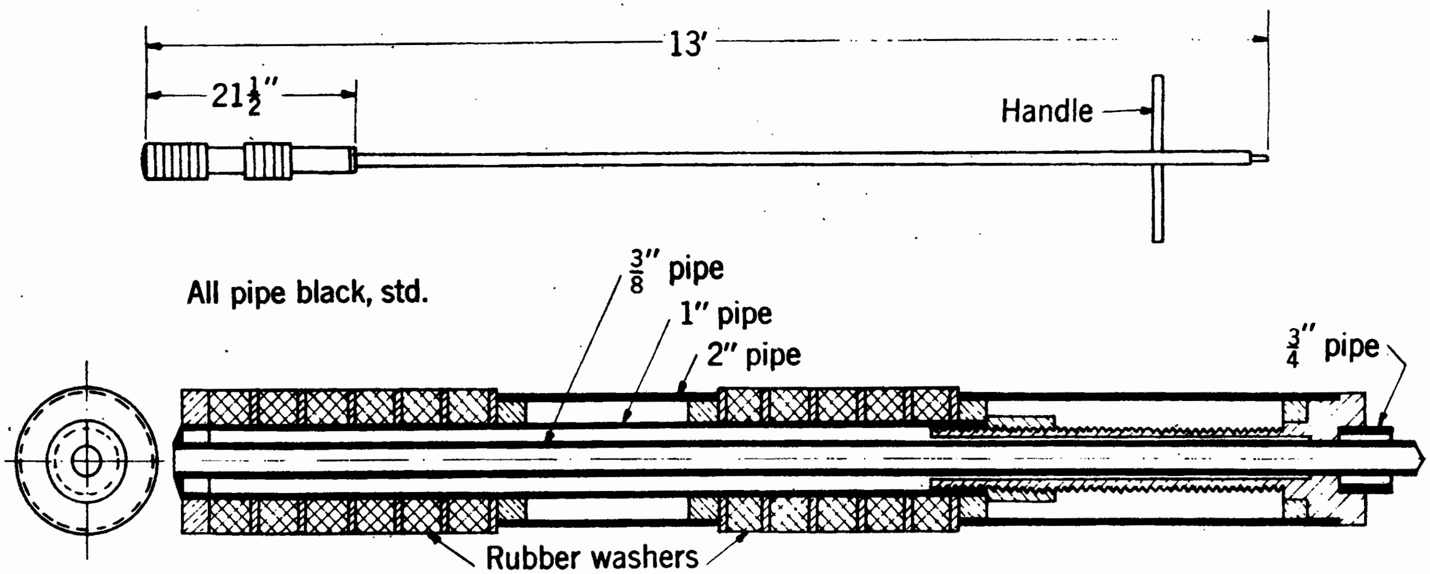


FIGURE 32

From United States Bureau of Mines R.I. 5353, 1957.

before an operation and was continued for about five minutes after the operation. Figure 33 and Table VII show the air-borne dust encountered during mining operations. Results show that injection of water into the coal face did not appreciably reduce the amount of dust liberated by undercutting, shot firing, or drilling the top holes. There appeared to be a slight reduction in float-dust resulting from infusion when the bottom holes were being drilled.

TABLE VII

FLOAT DUST IN RETURN AIR DURING MINING OPERATIONS (46, p. 8)

Test No.	Under Cutting	Dust in Return Air, billion particles/minute: Drilling, Top Holes	Drilling, Bottom Holes	Shot Firing	Loading Cars	Cars
1	70	80	5	15	10	120
2	20	30	0	45	5	55
3	50	50	10	290	5	15
4	30	45	10	220	10	25
5	60	50	5	180	5	5
6	45	40	5	120	5	35

Moisture retained in coal. The loss of weight was measured for drill cuttings and under cuttings passing through 20-mesh after drying at 105 degrees Centigrade for two hours. Table VIII gives the moisture content of coal cuttings.

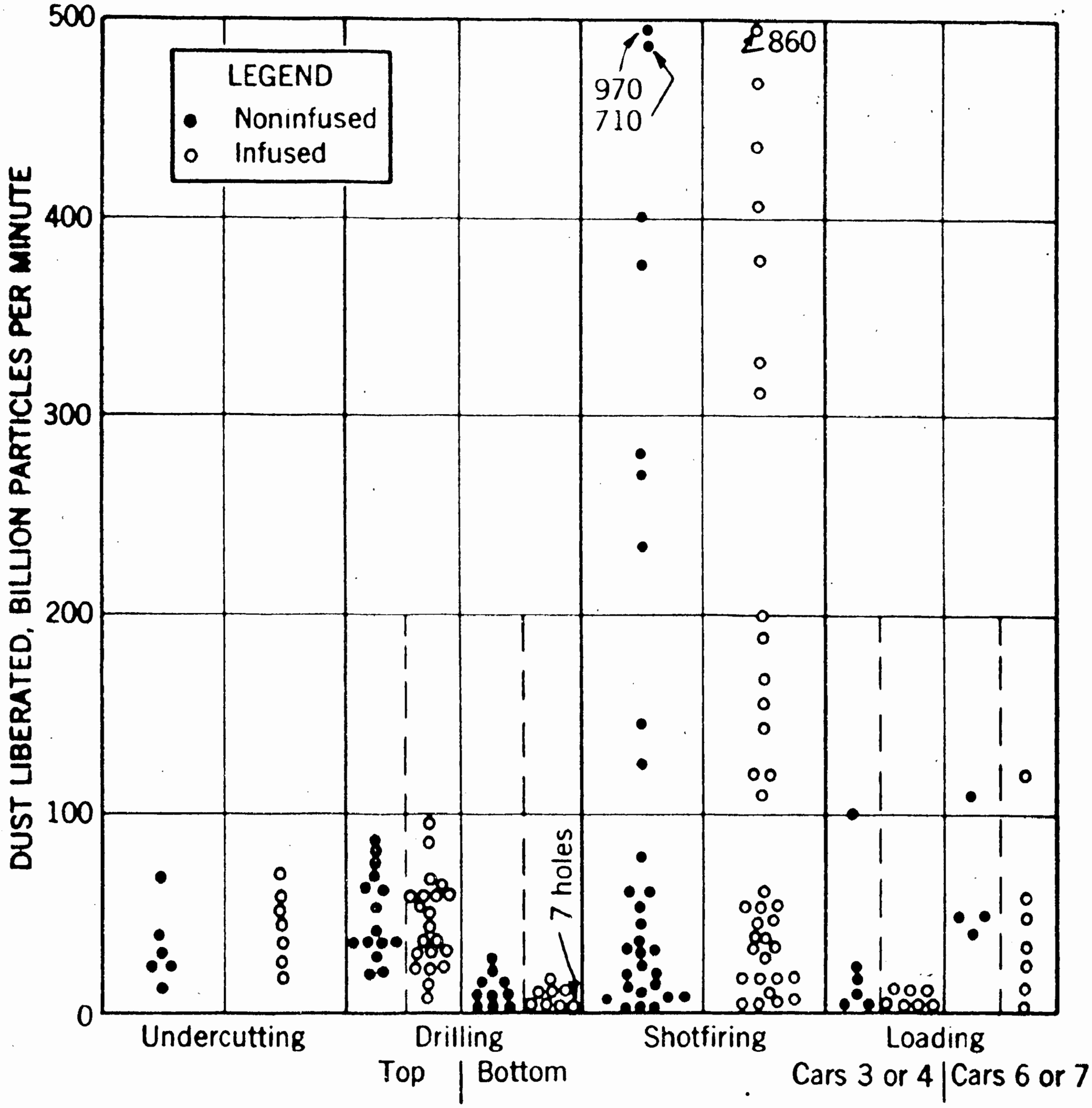


FIGURE 33

From United States Bureau of Mines R.I. 5353, 1957.

TABLE VIII

MOISTURE CONTENT OF COAL CUTTINGS (46, p. 6)

	Noninfused		Infused	
Operations	Number of Samples	Average Moisture Content, percent	Number of Samples	Average Moisture Content, percent
Under-cutting	6	3.8	7	6.7
Drilling top shot holes	15	1.8	21	2.0
Drilling bottom shot holes	10	2.3	14	4.4

Water Infusion of Coal Pillars.

Results of infusing coal pillars with water and water mixed with wetting agents have indicated conclusively that much air-borne dust created and released normally during mining, especially during drilling and blasting, can be "killed" or abated at its source by this method. (41, p. 2).

Water forced into holes drilled into coal pillars travels through the coal along the cleat planes naturally occurring within the coal. The coal that is strongly cleated at close intervals may be expected to offer less resistance to water flow from the drill hole than will coal which is more massive in structure. It is apparent that within the one area of coal, or even within the one pillar, there are zones which permit greater water flows than others. This is believed to be due to the effects of roof pressures and roof-floor convergence.

Visual observation of a pillar indicates that the periphery of a pillar consists of coal broken away from the solid coal by parallel pressure cracks caused by mining. If infusion holes drilled into such a pillar are not sealed beyond this broken periphery, water escapes along these cracks to the floor.

Holes may be bored at about midway in the coal bed. Where the bed shows the tendency to crush, which would result in the distortion and loss of holes so bored, it is advantageous to bore in any part of the bed which may be harder. Very long holes are not desirable in pillar infusion as length decreases the control over where the water goes. Figure 34 shows two methods of infusing a pillar. In general, the worse the pillar conditions are with regard to roof conditions, floor heave, and crushing of the coal; the shorter the holes and the closer their spacing should be.

A drawing of a water infusion seal suitable for pressures up to 350 pounds per square inch is shown in Figure 35. It is important that all flows in the pillars be measured and controlled. Uncontrolled pillar infusion leads to accumulation of water on the dip side and may have adverse effects on the stability of the floor and roof.

In pillar infusion it is desirable to infuse twice, the initial and major infusion should be done prior to splitting the pillars. That is, before the goaf line has advanced to the pillar; this should be followed by a second infusion during the actual extraction of the pillar to ensure the presence of surface moisture on the surfaces of the cleat planes and other cracks within the coal.

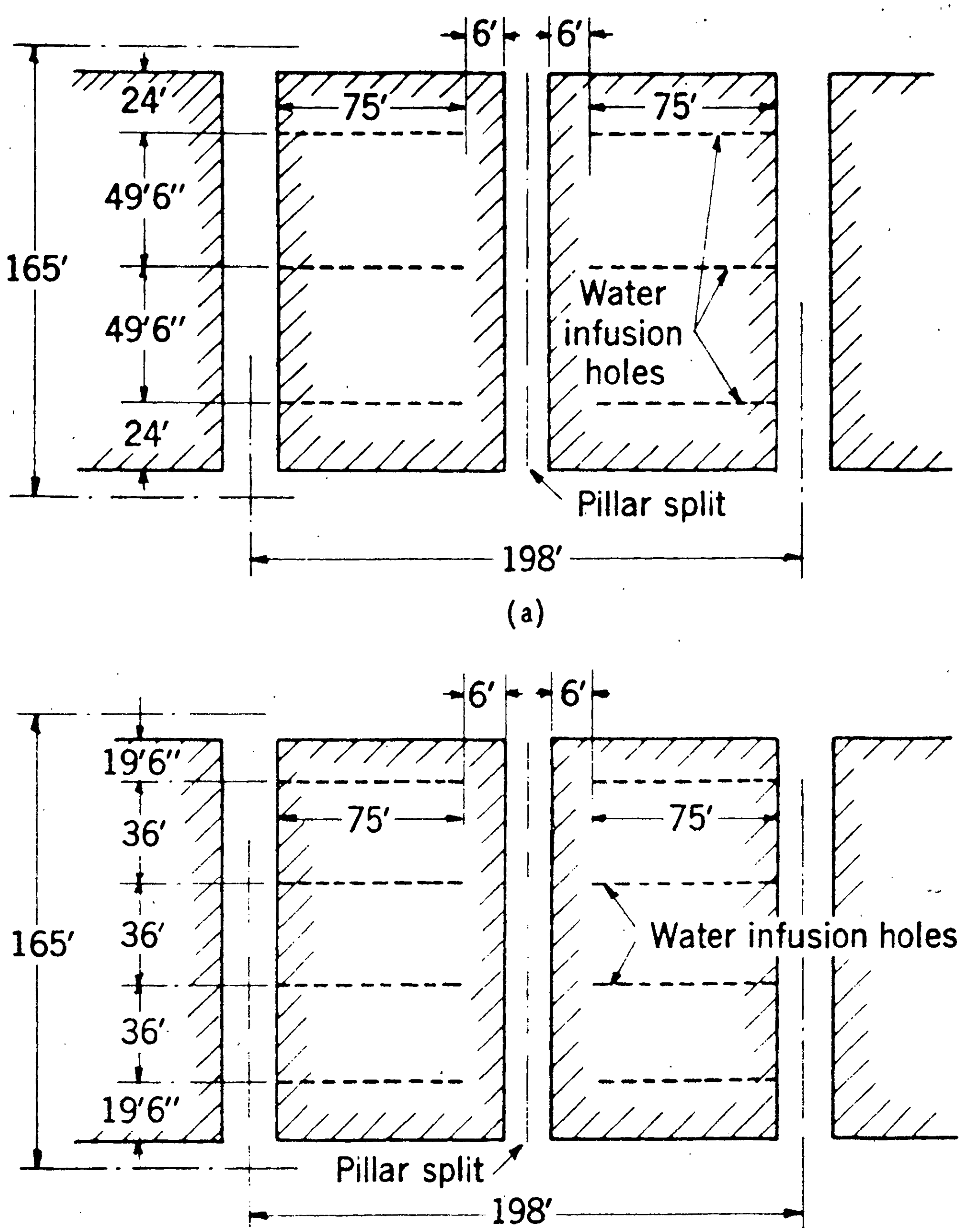


FIGURE 34

From United States Bureau of Mines 4836, 1951.

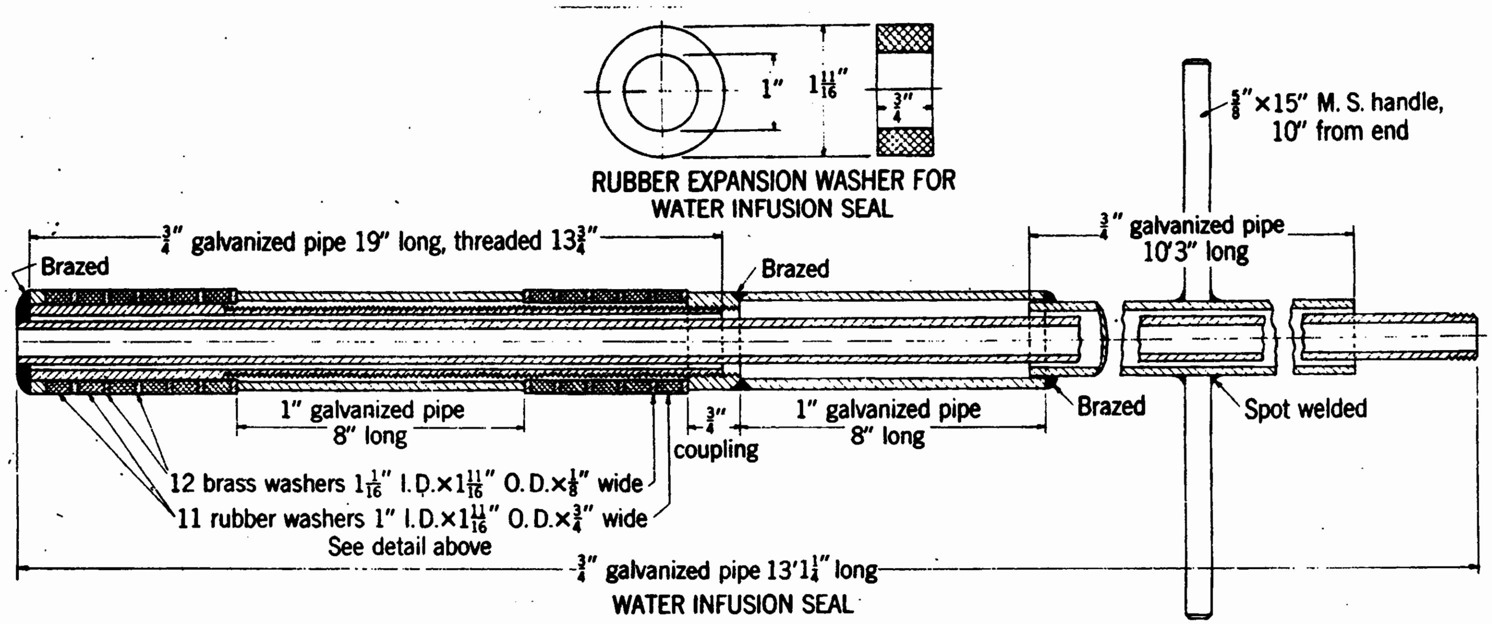


FIGURE 35

From United States Bureau of Mines R.I. 4836, 1951.

The data available indicates that with pressures in the range of 50 to 100 pounds per square inch, flows of as much as 30 gallons an hour are normal, that above 30 gallons an hour they are suspicious; and that when flows reach 90 gallons an hour, infusion should be stopped.

Pulsed Infusion.

This is infusion of water combined with shot-firing. Pulsed infusion results in a better spread of the explosive energy, the impulse on firing being transmitted through the water to the coal.

Infusion is done with a hydraulically operated impulse-firing gun. This gun differs from other types in that there are two water circuits, one for the infusion flow and the other for the water seal. Figure 36 shows the section of the gun. The gun consists of two metal tubes with a rubber sleeve insert. When the sealing pressure is applied, the assembly is fore-shortened, expanding the rubber ring diametrically against the sides of the hole. A 3/8-inch minor valve controls the flow. In one case, the water pressure recorded is 750 p.s.i. and water is supplied by a 5 g.p.m. pump.

Figure 37 shows the pattern of the infusion holes; the diameter of each hole is 1-11/16 inches and 16 holes are placed at approximately mid-position in the seam. The gun is inserted into the first hole and the sealing valve is opened. This expands the rubber sleeve and draws the inner tube back in relation to the outer case. When the gun is securely sealed in the hole the sealing valve is closed and infusion proper is commenced. The infusion pressure is about 250 p.s.i. and the

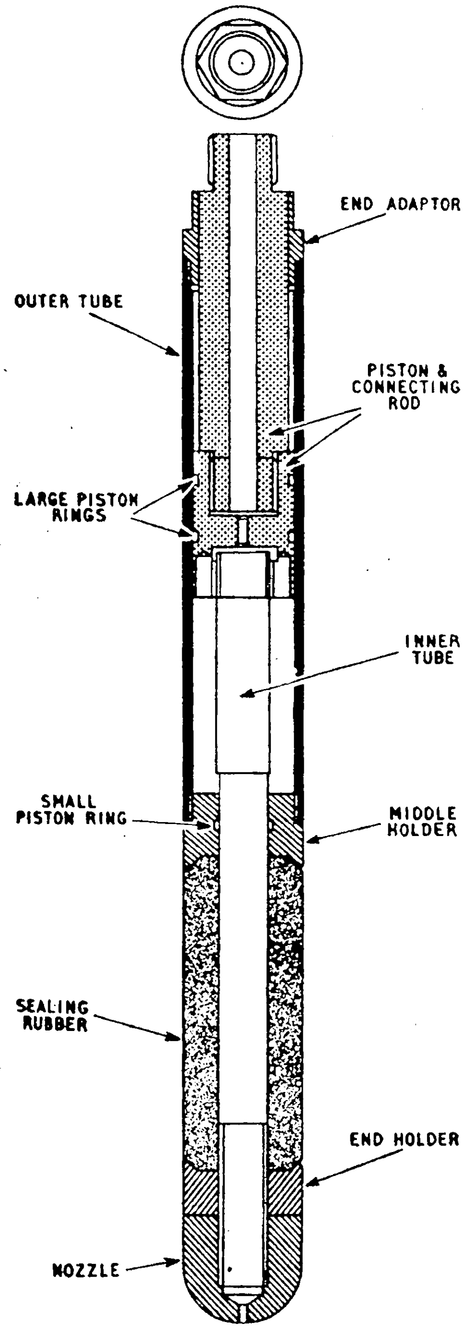


FIGURE 36

From Colliery Guardian, Vol. 196, 1958.

rate of flow is about 2½ gallons per minute. Infusion is continued until the water is seen to be oozing out of fissures and breaks. After the hole has been infused, it is charged with a special submarine explosive. The wires from the detonator, which is placed in the cartridge, are brought to the outside of the hole and the infusion gun is again inserted, care being taken to avoid any barring of the detonator wires. No stemming is required. Figure 38 shows the section of the pulsed infusion hole. During firing, the pressure developed is about 50 p.s.i. At the time of firing, the shot-firer keeps the combined water meter and guage unit under observation to ensure that the seal is being maintained. Before the shot is fired, to prevent the infusion gun being ejected from the hole, it is attached to an adjacent prop by means of a safety prop.

Due to water in the hole, the explosive force spreads more uniformly and, consequently, higher yield of large coal with considerable reduction in the amount of dust is achieved. The absence of objectionable smoke is also a notable feature. By drilling six-foot flanking hole at 45 degrees, pulls obtained were five feet and the coal very easily parted from the roof.

Without the use of water infusion technique, the same face required 40 holes with an average of 16 ounces of explosive in each hole. (66, p. 759).

High Pressure Water Infusion.

The principles of high pressure water infusion tube is shown in Figure 39. (65, p. 306). It mainly consists of two separate tubes, one fitting closely into the other. Water enters into the tube through the

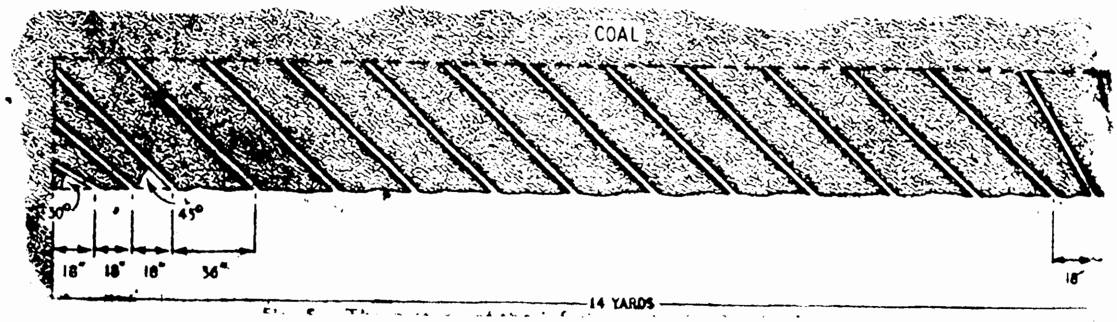


FIGURE 37

From Colliery Guardian, Vol. 196, 1958.

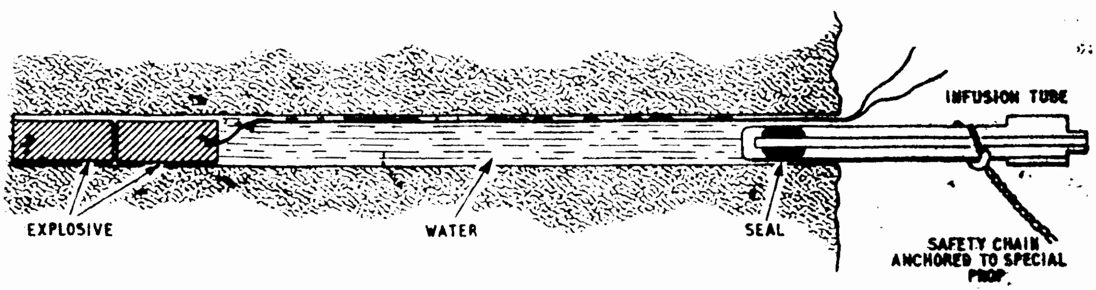


FIGURE 38

From Colliery Guardian, Vol. 196, 1958.

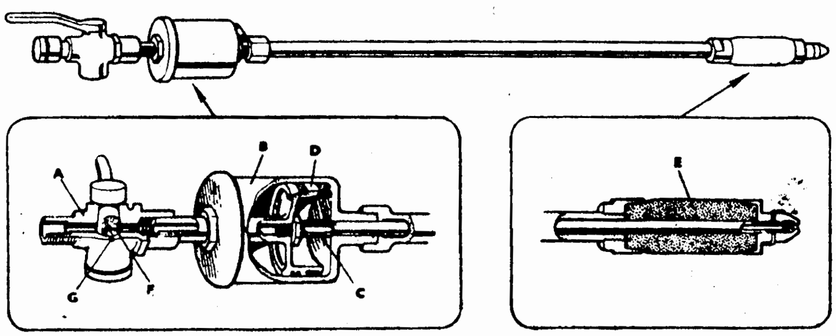


FIGURE 39

From Colliery Guardian, Vol. 194, 1957.

control cock A and then into the cylinder B via the ports C. Due to the pressure of water, the piston D is forced back, and as a result, the inner tube is forced out into the outer tube. This makes the rubber seal E to expand and grip the hole. After the hole has been infused, the cock A is operated and the water in the cylinder and inner tube comes out through the vents F and G, thus releasing the pressure and allowing the tube to be withdrawn.

In high pressure infusion, much depends upon the pump employed. The pump used is a five ram constant delivery type which delivers 11 gallons per minute at 950 r.p.m. and at pressures ranging up to 1,000 pounds per square inch, requiring an input of 10 H.P. The pump is fitted with an adjustable relief valve by means of which the pressure can be varied from 0 pounds to 1,000 pounds per square inch.

A new type of high pressure infusion gun (65, p. 405) can treat holes as long as 30 to 40 feet and develop an infusion pressure of 3,600 p.s.i. when the static pressure is 4,000 p.s.i. The amount of water required in high pressure infusion depends on mine conditions. In one case, where the mine was very dry and dusty and the seam was four feet thick, 1,000 gallons of water were required for every 40 yards of face. High pressure infusion can be applied periodically as opposed to shift-by-shift infusion in case of ordinary water infusion.

The following general conclusions can be drawn on the basis of the tests carried out under varied physical conditions: (a) water infusion system can be adopted for dust suppression in any seam in which the cleavages or slips are pronounced, (b) in the dustiest seams, from 80 to

90 percent of the air-borne dust comes from the face of the cleavage planes or slips, and also the crushed coal bordering these planes. Following saturation of the seam by water infusion methods the dust can be limited under extreme conditions to 1,000 particles per c.c., (c) in the dust spot samples taken, it is estimated that 80 percent of the particles are below five microns, (d) with medium and bad conditions of roof and floor, boreholes up to eight feet in length and spaced from eight yards to 12 yards apart have been found to give the best saturation results, (e) trials in any new seam should include preliminary tests to decide the horizon of the boreholes so as to avoid seepage of water into roof or floor, (f) if the holes are fed simultaneously, it is necessary to ensure a plentiful supply of water, with a water column along the face at least two inches in diameter, (g) the rate of travel of water infusion or moisture in a seam along any particular path appears to be largely independent upon the pressure of water. The pressure seems to be necessary only to maintain the flow to advance along the numerous planes where the capacity of the seam does not impose a limit upon the general rate of saturation, (h) although the period taken for saturation is usually longer with low water pressure, saturation seems to be more effective, (i) in certain seams, the system of water infusion has a decidedly beneficial effect on the working of the coal, as it falls much easier. In seams where shot-firing was carried on it has been considerably reduced, (j) dust concentration is so reduced that the full benefit of better illumination is obtained, (k) the method of water infusion does not seem to have any detrimental effect on the screening operations

or on the marketing properties of the coal. Evaporation takes place and the moisture is greatly reduced during the transit of the coal from the face to screens.

Mine Roadway Dust Consolidation.

Use of water for allaying coal dust has been dealt with in the earlier chapter. It has been found that, in order to lay thick and irregular deposits of dust, it is necessary to obtain as rapid a surface wetting as possible and to avoid the spraying of too large a volume of water in one operation. The various wetting agents available, "Perminal W" and "D.S. 103" are the most suitable for the purpose.

The procedure to be adopted to get satisfactory wetting is as follows: (a) spraying with a two to three percent solution of wetting agent, at a rate of from two to four gallons per 100 square feet of floor area, according to the ability of the surface to carry it without any loss by drainage, (b) repetition of this treatment, after one hour, if the average thickness of the dust-deposit approached one inch, (c) completion of the treatment, after a further hour, by spraying with water alone in sufficient quantity to moisten the depost throughout.

Tests were made by treating roadways according to the procedure outlined and it was found that: (a) periodic watering was adequate to cope with much more than the moderate deposition of fresh dust on the treated roadway, (b) for a thinner dust deposit, second spraying with wetting agent might be omitted without disadvantage and with economy, (c) a fine spray was not essential, though preferable, for the initial application of wetting agent, while for subsequent spraying and for periodic treatment with water a coarse spray was adequate.

Treatment of a roadway. (35, p. 214). A full length of 600 yards of road was treated with wetting agent, Perminal W. The double treatment with Perminal solution was given, except over a short portion of the level roadway where least dust was present. The solution was applied from two 10-gallon limewash sprayers, and the final spraying with water was through a coarse hose nozzle or from ordinary garden watering cans with hoses. Spraying was carried out with some care to direct the spray towards the thicker deposits of dust and to avoid pools of excess liquid. 1,400 gallons of Perminal solution were required for the treatment and no appreciable change occured in the humidity of the air.

Dust on the roadway was laid satisfactorily and it dried within a day to give a firm floor. The floor has since been treated at intervals of two or three weeks with water, roughly applied from a hose, in sufficient quantity to bind any loose surface dust. The treatment had rendered this section of the road pleasant to travel and had entirely eliminated the dust.

The use of calcium chloride. A solution of calcium chloride is as readily absorbed as water by the road dust once the deposit has been wetted by Perminal solution. The effect of wetting agent upon $CaCl_2$ is to bind coal dust and maintain it in a moist condition, thus eliminating the need for periodic re-watering.

For the purpose of treatment, an ordinary oil-barrel can be used.

The barrel is mounted on a tub frame, as shown in Figure 40, and a guage-glass is fixed at one end of the barrel, its connection to the inside of the barrel being at the bottom. The guage-glass is protected at the sides by a wooden frame, and at the front by a sliding steel strip. Behind the guage-glass is fixed a paper scale marked off in intervals of two gallons. The top of the barrel has a rectangular section, 8 by 6 inches which is cut out and a fitting was made to which a double acting hand-pump could be attached. Two sets of flexible rubber tubes could be coupled to the delivery end of the pump.

One hose set had two deliveries of very fine sprays for use with the Perminal solution, and the other hose had only one delivery through a 1/8-inch diameter nipple and onto a dispersing plate for use with the calcium-chloride solution. In order to ensure equal treatment along the full length of a roadway, it is divided into short lengths of about 20 feet. The amount of calcium chloride required depends upon the humidity of the roadway.

To find the minimum proportion of $CaCl_2$ necessary to bind the dust on the floor after a preliminary treatment with Perminal solution without making it uncomfortably damp, three test lengths, each 100 feet, were laid out at the outby end of the previously treated roadway. The first length, for comparison, received no further treatment; the second was coarsely sprayed with 100 pounds of calcium chloride dissolved in 40 gallons of water; the third was similarly sprayed with a solution containing 50 pounds of calcium chloride in 40 gallons of water and, some time later, as the moisture content and general conditions of the

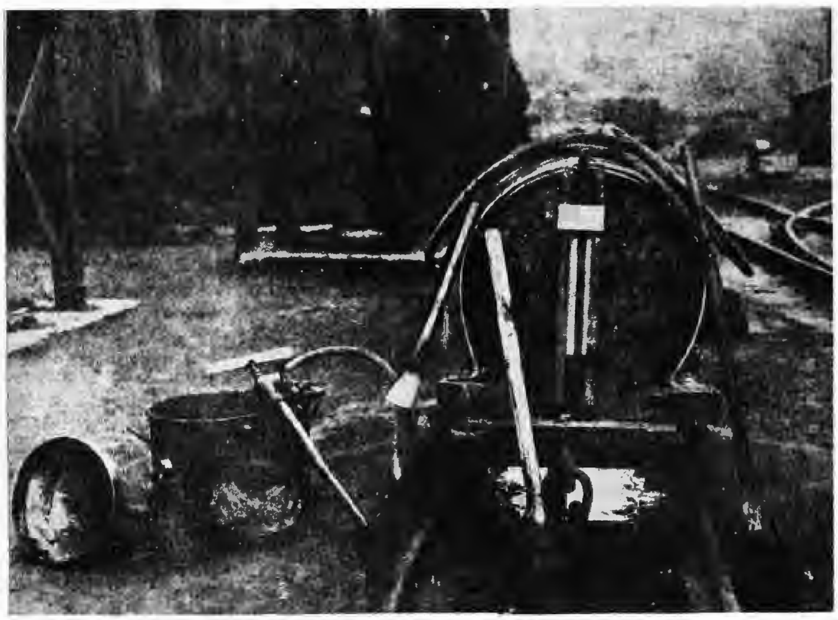


FIGURE 40

From Trans. Inst. of Min. Eng., Vol. 90, 1936.

dust on the floor indicated, it was further treated with a solution containing 100 pounds of $CaCl_2$. Calcium chloride solution soaked into the dust deposit thoroughly and there was no excess solution. The surfaces treated with $CaCl_2$ dried slowly. After several weeks, the untreated test length had a 1/4-inch layer of eroded material covering the consolidated dust. On the second length, there was a little eroded surface material but, although it was not damp, it could not be crumbled easily; the well-walked parts of this length of roadway were consolidated firmly. The third length also consolidated well.

The results are summarized below in Table IX.

TABLE IX (35, p. 256)

	Test Length Number		
	No. 1	No. 2	No. 3
Calcium chloride used, pounds	Nil	100	150
$CaCl_2$ in road dust, percent	Nil	0.9	1.5
Moisture in road dust, percent	5	7	10
Condition of road surface, after several weeks standing	Dry, considerably eroded	Dry, little erosion	Slightly damp no erosion

Incombustible Dust for the Prevention of Explosions

The application of a sufficient quantity of fine, incombustible dust has been considered as the most efficient and practical means of preventing the initiation and propagation of dust explosions in coal mines. The effectiveness of this method has been proved by experiments and experience in this and other countries.

The Federal Mine Safety Code for Bituminous Coal and Lignite Mines of the United States, July 24, 1946, contains the following provisions with regard to rock dusting:

Article VI - Section 2 - Rock Dusting

(a) All mines, except those mines or those locations in a mine which are too wet or too high in incombustible content to propagate an explosion, shall be rock-dusted to within 40 feet of the faces of rooms and entries; however, if the mine or any part of it is wet but becomes dry, the mine or portion of the mine so affected shall be rock-dusted as soon as it becomes dry.

(b) In mines partially rock-dusted or in mines that are required to start rock-dusting, haulageways and parallel entries connected thereto by open crosscuts shall be rock-dusted. Back entries shall be rock-dusted for at least 1,000 feet outby the junction with the first active entry. Inby this junction, the rooms, entries, and crosscuts shall be rock-dusted by generalized rock dusting as provided for in section 2(a).

(c) Where rock dust is applied, it shall be distributed upon the top, floor, and sides of all open places and maintained in such quantity that the incombustible content of the mine dust will not be less than 65 percent. In trackless entries and air courses, protection in lieu of generalized rock dusting may be had by using bags filled with rock dust, if they are placed at regular intervals and staggered as described in Bureau of Mines Report of Investigation 3411.

(d) When methane is present in any ventilating current, the 65 percent of incombustible matter shall be increased one percent for each 0.1 percent methane.

The above code recommends a 65 percent content of inert or non-combustible material in the coal dust. This is an average figure based upon numerous tests by the U. S. Bureau of Mines.

In other countries the use of rock dust is also governed by regulations. The British Regulations prescribe that (60, p. 171):

> (1) The resultant dust shall always consist throughout of a mixture containing not less than 50 percent of incombustible matter.
>
> (2) The incombustible dust used shall (a) be noninjurious to health and (b) contain not less than 50 percent by weight of fine material capable when dry of passing a sieve with 200 mesh to the linear inch, provided that if more incombustible dust is used than is necessary to comply with (1), the percentage of fine material may be reduced proportionately but shall not fall below 25.

To determine the amount of incombustible dust required to render coal-dust harmless, consideration must be given to a large number of factors. These factors may be grouped under three main heads, namely, those related to: (a) the coal dust, (b) the incombustible dust, and (c) the mining conditions.

The main factors as regards (a) are: (i) the relative inflammability of the coal-dust; (ii) the size range of its particles; and (iii) the amount present.

As regards (b), the important factors are the physical and chemical properties of the dust, such as: (i) specific gravity, specific heat, size range and shape of the particles, and the CO_2 and hydrated water-content; (ii) the hygroscopic and the balling and caking properties of the dust; (iii) the dispersability of the dust; and (iv) the amount present.

As regards (c), they are: (i) the condition in respect of roughness of surface and presence of moisture on the rock or other surfaces on which the dusts are deposited; (ii) the temperature and humidity of the atmosphere; (iii) the mode and rate of deposition of the coal and incombustible dusts; (iv) whether the dusts are in intimate mixture or in layers; (v) the thickness of the deposit; and (vi) the composition and resistance of the exposed surface layer to movement by an air-blast.

Large scale experiments have shown that for all practical purposes, the volatile content of the dust can be taken as a measure of its inflammability. Different percentages of incombustible dust are thus required for different coal dusts.

An incombustible dust is believed to be effective in preventing the propagation of coal-dust explosion mainly by reason of its capacity for absorbing heat and/or its screening effect when suspended in a cloud with coal-dust. The physical and chemical properties on which the effectiveness of the dusts depend, have been the subject of many investigations, but it is still doubtful which property is most important.

The important physical and chemical properties of the more commonly used materials are given in Table X.

Specific gravity suggests itself at once as a factor having a bearing on the action of a dust, yet definite conclusions on this point are difficult to obtain. The incombustible dusts in general have a specific gravity of from 2.2 to 2.7, approximately twice that of the

TABLE X

SOME IMPORTANT PHYSICAL AND CHEMICAL PROPERTIES OF INCOMBUSTIBLE DUSTS (60, p. 174)

Dust	Specific Gravity	Specific Heat (20^{o}-$100^{o}C$)	Fineness: % Thru 200 Mesh	Fineness: Surface Factor Sq. Cm/Gm	Dispersion: Equal Wts. Shale= 100	Dispersion: Equal Vol. Shale= 100	Moisture at $103^{o}C$%	CO_2 %	Percentage wt.-loss on ignition at $700^{o}C$	Percentage wt.-loss on ignition at $940^{o}C$
Shale	2.732	0.2148	84.62	3,049	100	100	1.56	-	-	26.81
Fuller's Earth	2.569	0.289	88.48	1,810	-	-	4.96	11.45	-	18.98
Gypsum	2.204	0.4244	91.02	2,220	85.5	57	13.98	-	22.1	64.42
Anhydrite	2.717	0.1961	68.00	2,880	117	86.6	0.82	-	2.91	57.35
Lime pulp	2.31	0.2457	45.65	2,225	120	80.4	2.18	39.56	-	45.03
Limestone	2.68	0.2094	81.76	2,690	98	80	0.618	30.50	-	30.76

average coal-dust. Extra fineness of particles can alone make up for this difference in specific gravity when the particles are in suspension.

Dispersability of dust. An incombustible dust can be an effective flame suppressor when it is raised into a cloud. Its dispersability and buoyancy, that is, its ease of cloud formation and its capacity to remain suspended in a cloud over an appreciable interval of time, are very important factors of its efficiency.

Figure 41 shows the apparatus for measurement of dispersability of dusts. (60, p. 179). It consists essentially of a cubical box containing three recessed windows, one of which is illuminated by an electric bulb. At the other windows, photronic cells are fitted and connected to galvanometers. An equal weight of dust was used in each case and dispersed centrally between the windows by a puff of compressed air from a jet in the bottom of the box. The shadow of the dispersed dust or the amount of light cut off from the cell opposite the lamp was measured by the deflection of the galvanometer from its normal position. In the case of dust particles reflecting light, it was assumed that some light would be reflected into the cell opposite the lamp and thus reduce the shadow and hence the dispersion value of the dust. The second cell was incorporated to measure the light, on the assumption that the amount of light reflected to both cells would be equal. By plotting galvanometer deflections to a time basis, the dispersion curves of the various dusts, varied in respect of fineness and moisture content, were obtained. A typical curve is shown in Figure 42.

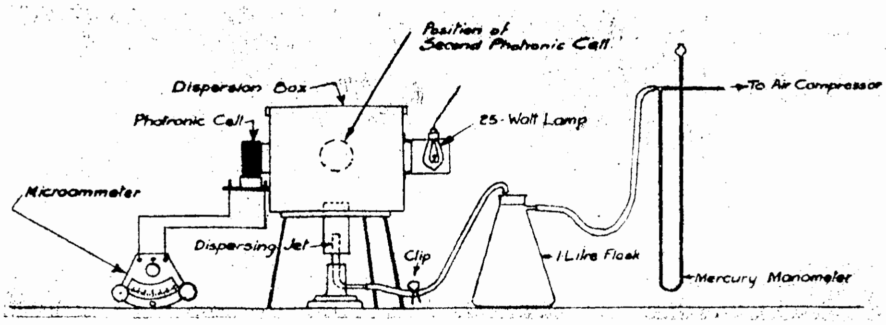


FIGURE 41

From Trans. Inst. of Min. Eng., Vol. 94, 1938.

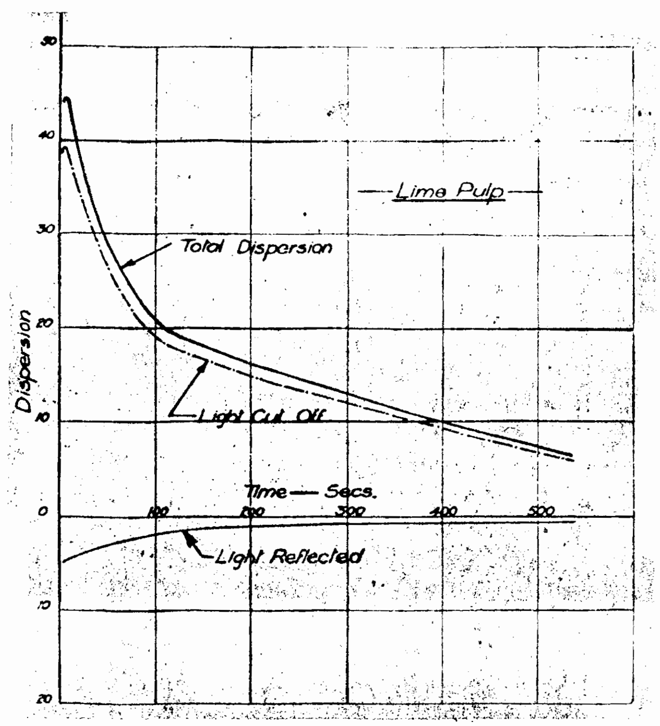


FIGURE 42

From Trans. Inst. of Min. Eng.,
Vol. 94, 1938.

Relative to shale as 100, the values obtained are given in Table XI and Table XII.

TABLE XI

RELATIVE DISPERSION VALUES OF INCOMBUSTIBLE DUSTS (EQUAL WEIGHTS) (60, p. 177)

Dust	Percent Light Reflected	Total Dispersion
Shale	8.7	100
-100 mesh coal dust	- -	226
Softener product	7.6	170
Limestone	8.85	98
Pixie powder	9.45	77.5
Gypsum	8.50	85.5
Anhydrite	10.0	117
Lime pulp	12.5	120

The effect of fineness in all cases is to increase the dispersion value of the dust. The surface of the dust is very important in suppressing flame; taking into account the specific gravities of the dusts, in a 50/50 mixture by weight of coal and shale dust of equal fineness, there will be more than twice as many coal as shale particles, and hence, more than twice the surface. It is therefore desirable that values should be based on a bulk basis.

TABLE XII

DISPERSION VALUES FOR VARIOUS MIXTURES OF SIZED FRACTIONS OF DUSTS (60, p. 178)

Dust	All Above 200 Mesh	Normal Dust	All Thru 200 Mesh	50% Thru 200 50% Above 200	35% Thru 200 65% Above 200	25% Thru 200 75% Above 200
Shale	9	55	56	42	35	31
Anhydrite	7½	38	39	27½	20½	19
Gypsum	6	29	35	22	17½	15
Pixie powder	7½	31	31½	22	17½	15
Softener product	12½	56	56½	44	37	28½
Limestone	9½	31	32½	30	22½	18

Testing the dusts on an equal volume basis, the dispersion values are found to be more nearly alike. The values are given in Table XIII.

Effect of moisture on the efficiency of dusts. As moisture is added to a dust, its dispersability is rapidly reduced. The finer the dust, the more severely is it affected. Figure 43 shows the effect of moisture on fractions of shale dust. The effect of moisture is more marked at the beginning and has less effect at higher percentages when the dust has reached a mealy condition.

The effect of moisture on the incombustible dust is to require an increased initial quantity of that dust to ensure suppression of inflammation. Increased quantity is required to equalize the reduction in

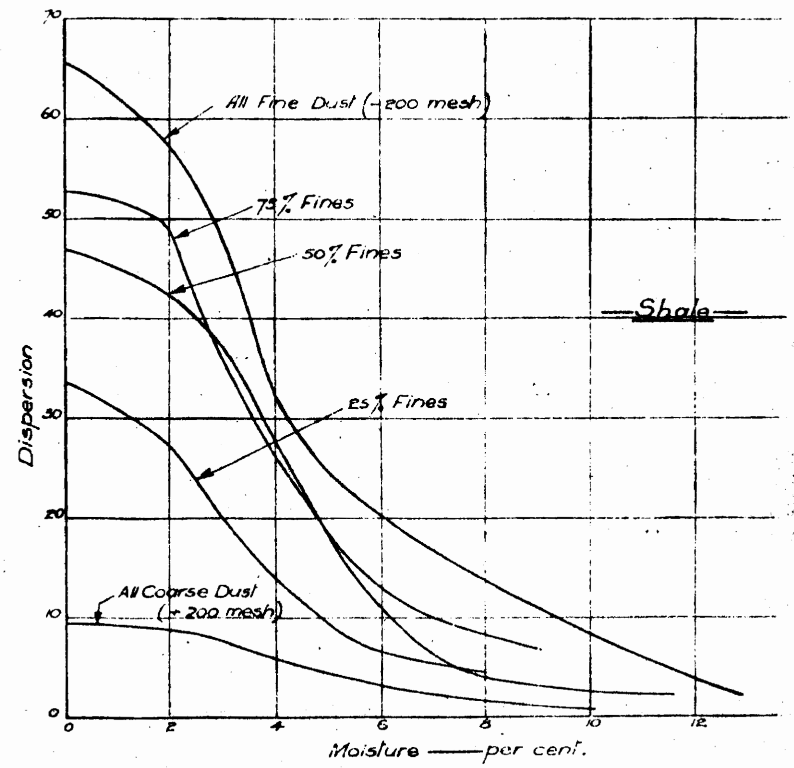


FIGURE 43

From Trans. Inst. of Min. Eng., Vol. 94, 1938.

dispersion consequent upon the addition of moisture.

TABLE XIII

RELATIVE DISPERSION VALUES OF INCOMBUSTIBLE DUSTS
(EQUAL VOLUMES) (60, 180)

Dust	Dispersion Values (equal volumes)
Shale	100
-100 mesh coal dust	98.2
Softener product	80.4
Anhydrite	86.6
Gypsum	57
Pixie powder	57
Lime pulp	80.4
Limestone	80

<u>The movement of dust</u>. The question of the ease of moving a dust is altogether apart from that of its power of dispersing. The ease with which a layer of dust can be moved is more or less a measure of the force of adhesion between the dust and the surface on which it is placed. This force is dependent on the nature of the surface, on the composition and fineness of the dust, and on the method by which the dust was applied. Once the dust is moved, its dispersability properties play a part in the formation of dust clouds. The aim is to prevent the initiation of an explosion by the provision of incombustible dust where ever there is coal dust. This generally necessitates a thin layer of

incombustible dust over all surfaces.

Experiments on the movement of dust from layers under various conditions were conducted and they indicated generally that: (60, p. 189) (a) incombustible dusts vary appreciably in their ability to be moved from a layer. Gypsum was the most difficult to remove. The heavier shales and limestones were more easily removed; but as gypsum has a comparatively lower specific gravity it has been extensively used; (b) coarse dusts offer most resistance with least surface adhesion, and hence, are most easily removed. Fine dusts tend to flatten out their irregularities and form a smooth surface, offering very little resistance; (c) as a consequence of the flattening effect, increased thickness of layers does not assist in a greater removal of dust at velocities up to 100 feet per second. A definite amount of dust is removed at each velocity, irrespective of the increase in thickness of the layer over a given minimum; (d) initial roughness of surface increases the amount removed, a smooth layer gives least removal; (e) shocks and eddies cause greater removal of dust in all cases; (f) for maximum removal of incombustible dust, it must form the top layer; (g) moist and balled up dusts are more easily removed from a dry surface than dry dusts, because balling is equivalent to an increase in particle size. Dispersion, however, is reduced; and (h) a minute film of moisture, such as caused by breathing on the surface of adhesion, greatly increases the adhesion of all incombustible dusts. Coal dust, however, is not affected.

Balling and caking of dusts. Balling is the tendency of dusts to take up the form of spherical agglomerates. Many dusts exhibit a marked tendency to ball in their normal condition; the presence of a large percentage of moisture is not necessary. With the increase in moisture content, balls are more pronounced; larger as a rule and show more cohesion in general. It is apparent that the cloud producing properties of these dusts must be very much reduced. In case of balling in the normal condition, the inclusion of over 30 percent of material below 15 microns causes balling in the average stone dust.

To eliminate balling tendencies, the reduction or elimination of the fines fraction is desirable. The elimination of the fraction below five microns is required for purely physiological reasons. But if this is done, the ability of the dust to suppress inflammation and also its property of adhering to surfaces are decreased.

Incombustible Required to Prevent Propagation of Explosions

After rock dust is applied in mine workings, the dust on the floor frequently becomes rapidly contaminated by coal spilled during haulage, by abrasion of coal ribs, and by coal dust settling from the air current. As a result, the incombustible content of the floor dust often falls below 65 percent, whereas that of the less contaminated overhead dust remains above the required limit. To find out to what extent a deficiency of rock dust on the floor might be tolerated with safety and whether it can be compensated for by excess rock dust on the rib and roof surfaces, investigations were made by the U.S. Bureau of Mines at their Experimental Mine.

About 70 single-entry explosion tests were made in the 1,308 feet long main entry, nine feet wide by seven feet high, of the Experimental mine (32, p. 2). Bituminous Pittsburgh coal dust was used in all tests. Coal dust was prepared so that all particles passed through a 20-mesh sieve and 20 percent by weight passed through a 200-mesh sieve; in a few tests, 28 to 30 percent of the dust was finer than 200-mesh. The rock dust used in the tests was limestone containing 0.1 percent moisture and 99.9 percent total incombustible; 75 to 80 percent of the dust passed through a 200-mesh sieve.

Coal and limestone dusts were mixed intimately on the surface. Separate mixtures were prepared for loading on the floor and on the side and overhead shelves in the test zone. The dust was distributed manually with hand scoops. To study the effect on the rock dust requirements of different dust conditions in mines, the rate of dust loading was varied in different groups of experiments as follows; the values indicate the weight of coal dust in the dust mixtures per linear foot of entry, uniformly applied in the ignition zone: (a) 2/3 pound on rib-roof (one-half is always placed on the longitudinal side shelves and one-half on cross shelves), 1/3 pound on floor; (b) 2/3 pound on rib-roof, 2/3 pound on floor; (c) 2/3 pound on rib-roof, 1-1/3 pound on floor; (d) 1/3 pound on rib-roof, 1-1/3 pound on floor; (e) 1/6 pound on rib-roof, 1-½ pound on floor. The incombustible content of the dust mixtures on the floor ranged from 36 to 68 percent, and in the rib-roof dust from 36 to 86 percent.

To study the effect on the compensation of rock dust deficiency in floor dust, four different igniting sources were used in the investigation. The weakest source is a 5-ampere electric arc, over which 25 pounds of coal dust, 75 to 80 percent through 200-mesh, is projected from a trough by compressed air; 100 pounds of pure pulverized coal dust is distributed in a 50-foot zone near the face. The second source consists of 1,300 cubic feet of 9 to 9.5 percent gas-air mixture in a 25-foot zone near the face of the entry; the mixture is ignited by an electric spark. The third source is a blown-out shot of four pounds FFF black powder, fired from a steel cannon in the coal face into a 50 foot zone of 100 pounds of pulverized coal. The arrangement of the fourth igniting source is similar to the third, but the pulverized coal zone is extended 33 feet at the rate of one pound per linear foot of entry.

The following expression can be used to find out the dust content at the various places: (32, p. 3)

$$I_t = \frac{I_r W_r + I_f W_f}{W_r + W_f}$$

where,

I_t = Incombustible content of total dust, percent;

I_r = Incombustible content of rib-roof dust, percent;

I_f = Incombustible content of floor dust, percent;

W_r = Weight of rib-roof dust (coal plus limestone), pounds per linear foot.

W_f = Weight of floor dust (coal plus limestone), pounds per linear foot.

The effect of reducing the incombustible content of the floor dust is shown in Figures 44 and 45. The data are plotted in three separate blocks on each figure, to correspond with three sources of ignition, in Figure 44 the incombustible in the rib-roof dust is plotted against the incombustible in the floor dust; Figure 45 shows the incombustible in the total dust, computed according to the expression given above, with relation to incombustible in the floor dust.

Tests indicate that deficiency of rock dust on the floor of mine workings can, within certain limits, be compensated for by excess rock dust on the rib-roof surfaces, so that the weighted incombustible content of the total dust around the perimeter of an entry is maintained at 65 percent. The limits depend on the strength of a potential explosion and on the distribution of dust on the floor and overhead. In most mines the amount of dust on the floor is many times greater than overhead. If protection is desired against explosions initiated by a strong source that might disperse say three times as much dust off the floor as off the rib-roof surfaces, then if the incombustible in the floor dust were reduced to 55 percent, the incombustible in the overhead dust would have to be 95 percent to bring the value in the total dispersed dust to the necessary minimum of 65 percent. Under such conditions therefore, virtually no rock dust deficiency can be tolerated.

<u>Improving the Adherence of Rock Dust</u>

During normal application with machines, it is often difficult to make the rock dust adhere to the roof and ribs, particularly if these

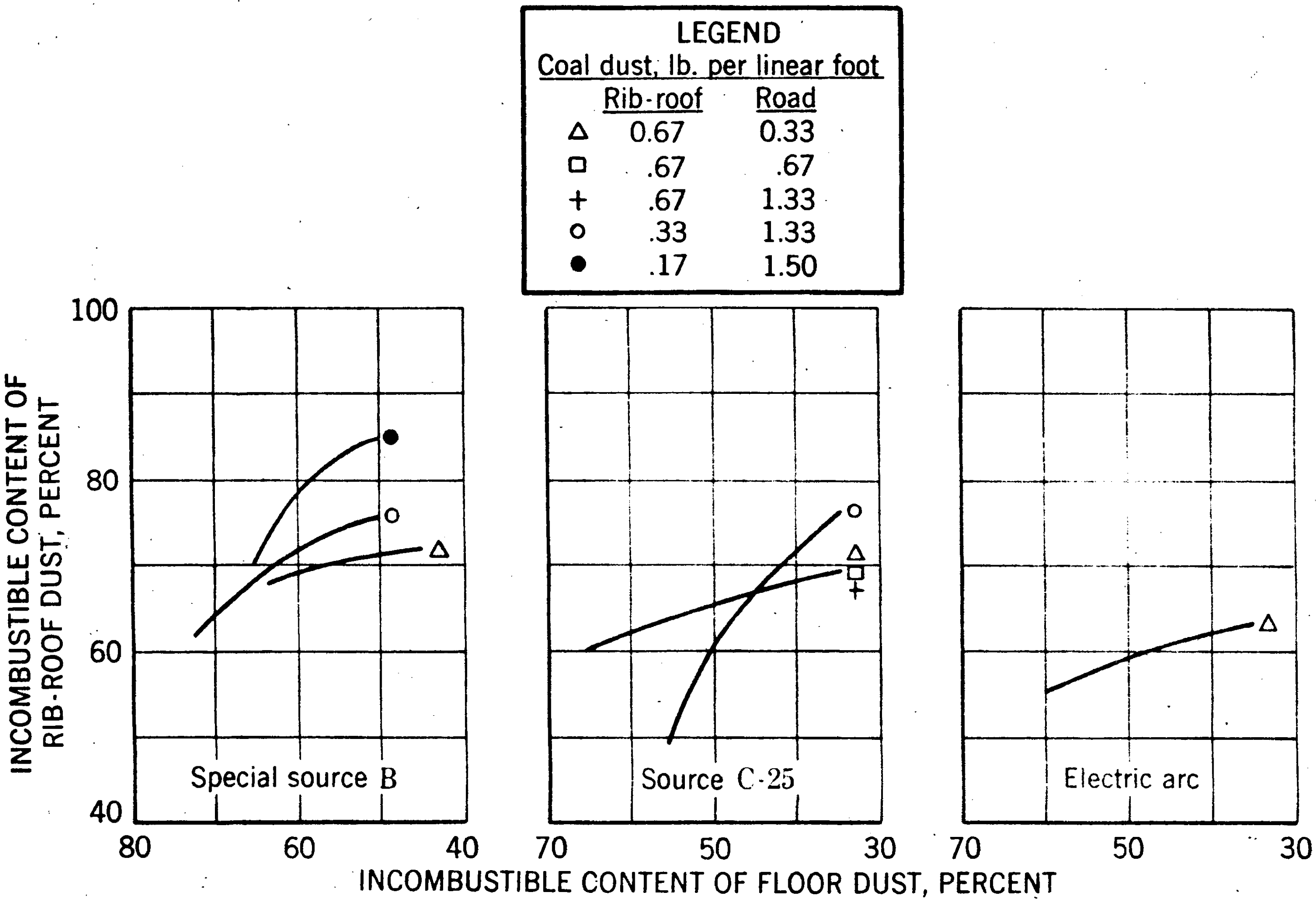


FIGURE 44

From United States Bureau of Mines R.I. 5053, 1954.

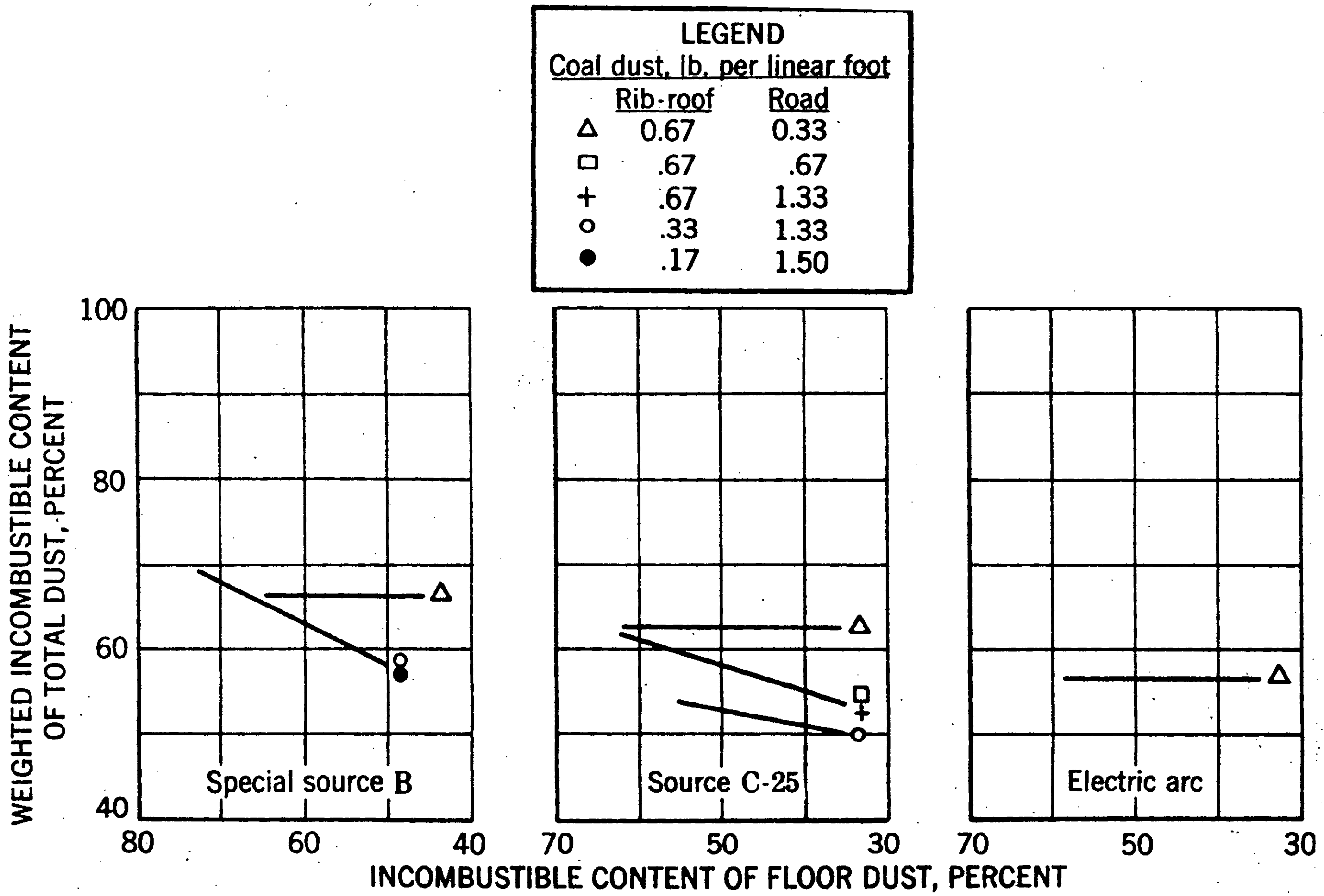


FIGURE 45

From United States Bureau of Mines R.I. 5053, 1954.

surfaces are dry and smooth. Tests have shown that, even with normally good adherence, not more than 30 to 35 percent of limestone dust sticks to the ribs and roof; the balance falls chiefly on the floor, and some is carried away by the air current.

Some tests and experiments were made by the Bureau of Mines and it was concluded that, when it is difficult to make rock dust adhere to the ribs and roof, these surfaces should be moistened before or during rock dusting. In commercial mines it would be advisable to determine by trial which of the available limestone dusts is most suitable and to what degree the ribs and roof should be wetted to achieve the desired results.

The requirement, that rock dusting in mine entries be carried to within 40 feet of active coal faces, may necessitate the application of rock dust during a working shift. This is particularly true for highly mechanized mines, where a face may advance several hundred feet daily. Conventional rock dusting with machines is a dusty operation, and the dense dust clouds may affect the health of the face workers and impair expensive mining machines. For this reason, in some mines long sections of entries are found without rock dust.

To remedy the situation, various substitute methods of rock dusting and other protective measures have been considered. One procedure involves the application of wet rock dust, either in the form of a premixed slurry or by mixing dry rock dust with water at the nozzle of a rock-dusting machine.

Wet rock dusting. The problem was investigated by the U. S. Bureau of Mines in the experimental coal mine and the objectives of the study were: (a) to determine whether wet rock dust could be applied practically in coal mine entries without creating undesirable dust clouds; to find the proper ratio of rock dust to water; to find the least quantity necessary for a good covering of the rib and roof surfaces; to establish the degree of adherence; and to determine the rate of drying after application; and (b) to study the effectiveness of wetted rock dust, after partial or complete drying, in arresting the propagation of explosions.

The results of the preliminary study showed that: (33, p. 9) (a) a pre-mixed slurry of limestone and water fed from a guniting machine through a gunite nozzle can be applied effectively on mine surfaces and will produce very little dust in the air; (b) limestone dust fed from a guniting machine or from a commercial rock dusting machine, mixed with water at the nozzle, can also be applied effectively and produces little dust in the air. Application with the gunite machine gave a more uniform flow than the small rock dusting machine; (c) a minimum of four pounds of rock dust, applied wet, was needed per linear foot of entry (9 to 10 feet wide by 7 feet high) to cover the ribs and roof completely. Six gallons of water mixed with 100 pounds of limestone gave a satisfactory mixture for nozzle application; the slurry required slightly more water. About 85 percent of the wetted rock dust adhered to the rib and roof surfaces; (d) the rate of drying of the rock dust varied with the humidity and with the velocity of the air current. At normal air flow, at relative humidities below 80 percent, the rock dust dried completely in one to three days; at humidities of 80 to 90 percent, it dried in about one week; and at higher humidities, several weeks were needed for drying; and (e) during dry rock dusting, when the air flow in the entry was maintained at 6,000 cubic feet per minute, the dust count, 25 feet from the nozzle, was found to be as high as 5,000 million particles per cubic foot of air, and 100 feet downstream the count was about 2,000 m.p./cubic feet. When the slurry was applied, the dust count was less than 0.5 percent of the foregoing values; when limestone dust and water were mixed by a nozzle, the dust count ranged from 1 to 10 percent of the foregoing values.

After application and drying of the wetted rock dust, a small amount of coal dust was distributed by hand over the rock dusted surfaces. The aim was to simulate operating conditions in a coal mine, where fresh float dust is transported by the air current and deposited in the entries daily. To serve as a basis for judging the effectiveness of wetted rock dust, several tests were made with dry rock dusting. The results of the explosion tests indicated that: (a) dry rock dusting was more effective than wetted rock dusting in arresting the propagation of explosions; (b) wetted rock dust was somewhat more effective after complete drying than after partial drying; and (c) dry rock dust distributed on the floor in entries where wet rock dust had been applied on rib roof surfaces helped greatly in limiting explosion propagation.

Figure 46 shows the application of limestone-water slurry on coal rib and Figure 47 shows the adherence of coal dust on wet rock-dusted roof and rib surfaces of mine entry.

Rock Dusting Machines

Compared to throwing the dust by hand, the use of a machine decreases the cost and greatly increases the convenience and efficiency of dusting. A machine blows a thick cloud of dust into the air current, which carries it for long distances, the dust gradually settling out and forming a mantle over the coal dust deposits.

The machine used at the experimental mine of the U. S. Bureau of Mines, consists of a small positive blower, the air from which passes through a two-inch pipe to the ejector chamber into which the rock dust is fed from a hopper; the air and dust, becoming mixed in this chamber, is blown through a hose into the atmosphere.

Figure 48 shows an outline of the arrangement. The blower used has a volume of about 288 cubic inches and is run at about 1,000 r.p.m. The pressure in the outlet pipe when the machine is operating is about two pounds per square inch. The two-inch air inlet is reduced to a nozzle of one inch opening, and the nozzle should extend far enough into the chamber of the three-inch tee so that the nozzle opening is below the outer edge of the dust-hopper opening. The dust then falls or is drawn forward into the air stream and is blown through the hose. A flexible hose is desirable for an outlet so that it can be pointed to any direction and also this permits its connection to pipes through stoppings to direct the dust stream into other air courses or entries.

FIGURE 46

From United States Bureau of Mines R.I. 4688, 1950.

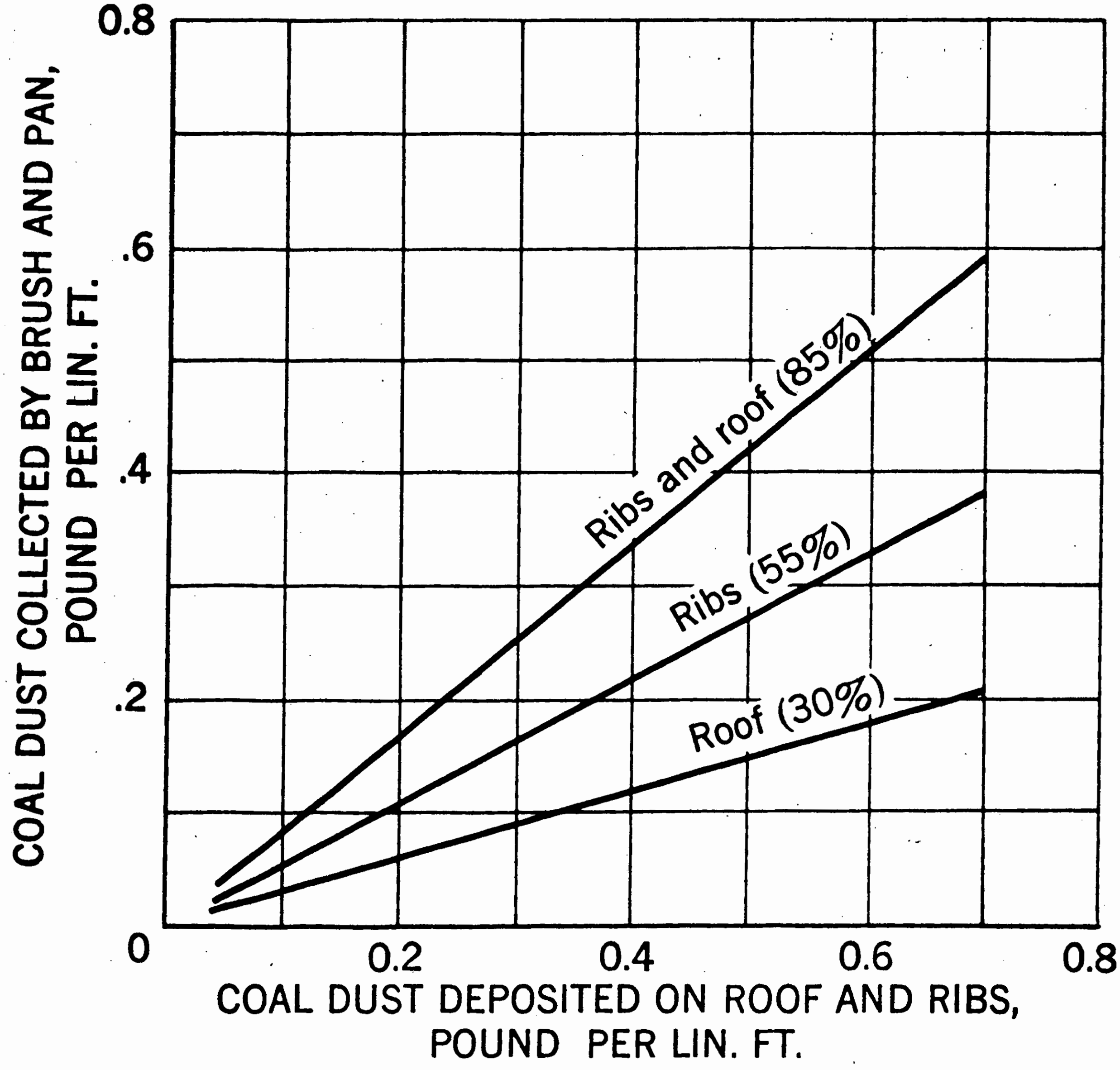


FIGURE 47

From United States Bureau of Mines R.I. 4688, 1950.

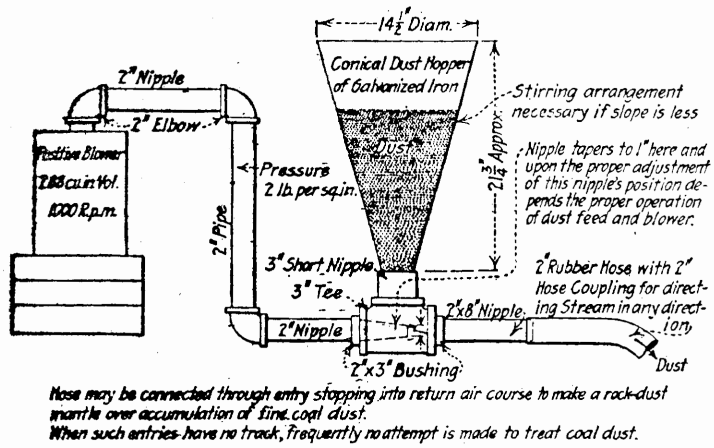


FIGURE 48

From Trans. Inst. of Min. Eng., Vol. 97, 1939.

Figure 49 shows the photograph of a modern rock-dusting machine. This is a high pressure portable type of rock duster for wet or dry dusting. It can deliver from 35 to 60 pounds of dust per minute through 50 to 400 feet of 1-3/4 inch hose. It is equipped with a wet nozzle and water supply at a pressure of 20 to 25 pounds per square inch. Figure 50 shows the wet nozzle. Important features of the machine are: quick-lock cams for attaching the hose, stainless steel hopper and multiple disc clutch for mechanical protections.

Specifications:

Length ----------------------------------	72 inches
Height ----------------------------------	18 inches
Width -----------------------------------	23 inches
Net weight ------------------------------	590 pounds
Motor H.P. ------------------------------	5
Hopper capacity -------------------------	160 pounds

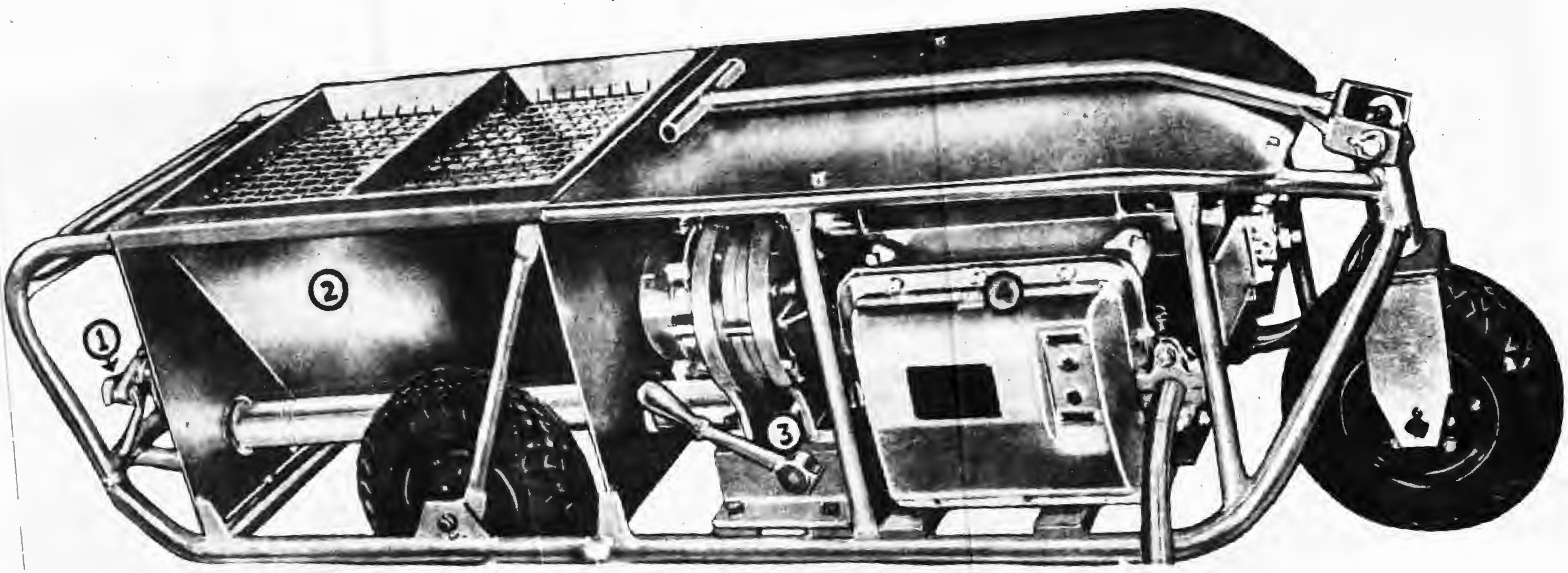


FIGURE 49

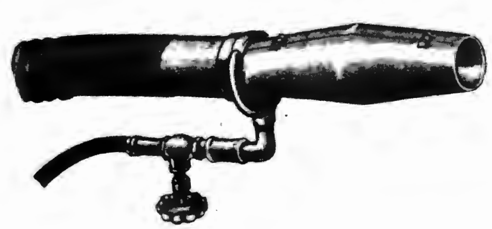


FIGURE 50

CHAPTER V

LIMITATION OF EXPLOSIONS BY BARRIERS

Various types of rock-dust barriers are used to confine coal dust explosions to the limited areas in which they originate. These barriers were first tested in 1910-1911, with favorable results under the conditions of test, at the French official gallery which was then at Lieven, Department of Pas-de-Calais. Shortly afterwards, the U. S. Bureau of Mines, in the course of its investigation of all practical explosion-prevention methods, tried barriers in their experimental mine, using the primary form of simple fixed cross shelves 20 inches wide extending over the passage way and laden with rock dust. The Bureau termed them "Taffanel barriers" in honor of the director of the Lieven testing station.

Development of various types of mechanically operated rock-dust barriers was carried out by the Bureau of Mines. Among these were: (54, p. 5) (a) a box barrier consisting of six to eight individual boxes, (b) a trough rock dust barrier, (c) a rock-dust ventilation stopping, (d) a rock-dust-protected ventilation door, (e) a rock-dust-protected overcast, and (f) a concentrated barrier.

It is beyond the scope of this work to give details of constructioı of all the types of barriers tried, however, some important types are detailed below:

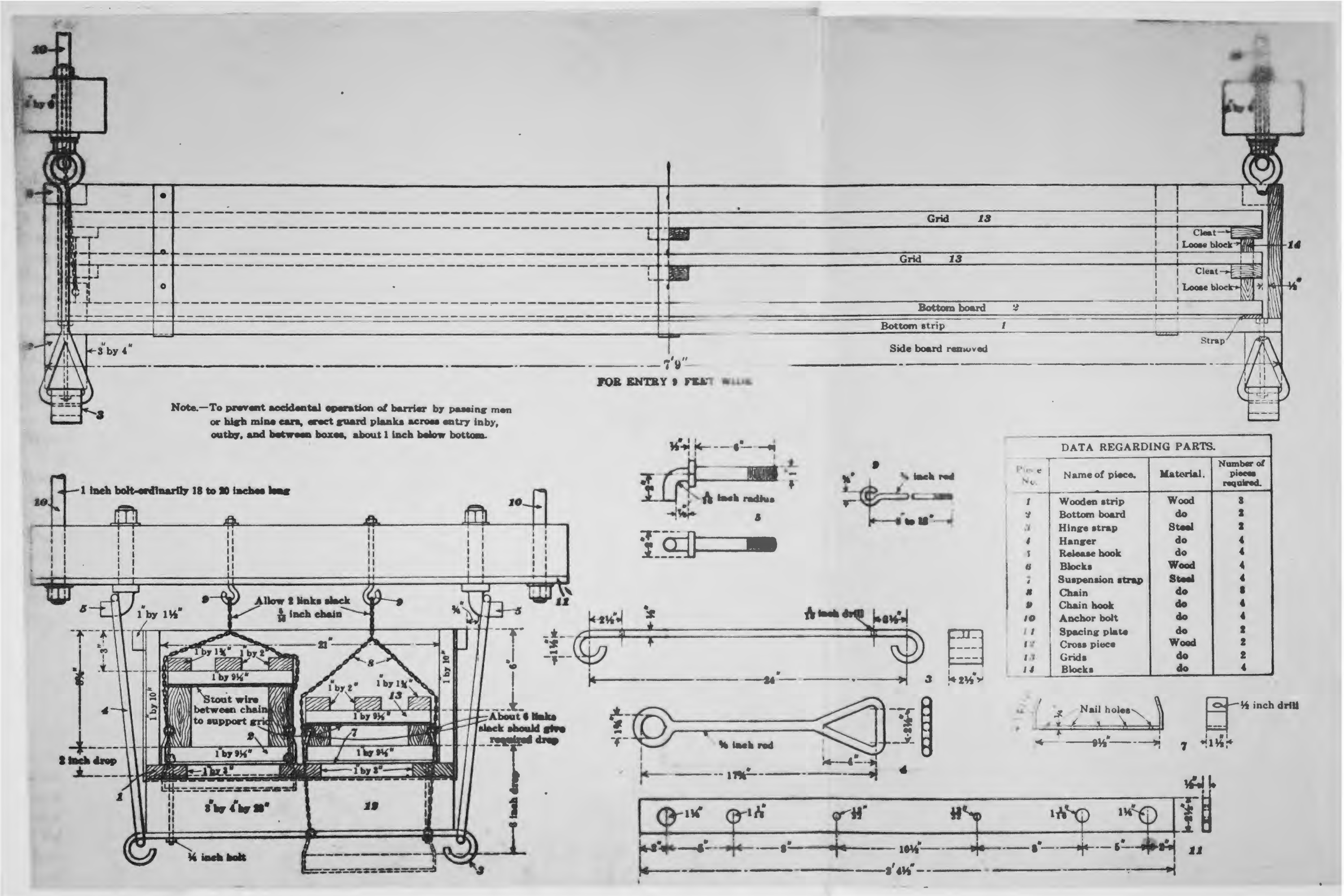
Box Barrier

The box barrier consists of six or more boxes containing incombustible dust, opentopped except for a loose waterproof cover, hung loosely from roof supports two to three yards apart and extending across the entry near the roof in such a manner that the explosion wave will cause them to be upset, thus throwing into the entry a large amount of incombustible dust. The bottom boards of the box are so arranged that after a short fall they are caught by chains, attached to the roof, so that some of the dust is retained on these boards. Two grids within the box, which rest loosely on blocks and are connected to the same chains at distances of three and six inches below the top of the box, also retain some of the dust and allow the balance to fall through the open spaces. The boxes are hung high enough to be clear of traffic.

Figure 51 shows the arrangement and the details of construction of one type. The boxes used at the experimental mine, are about 7½ feet long, 23 inches wide, and 10 inches deep. Each box will hold 700 to 800 pounds of rock dust. The length and number of boxes required will vary with the cross section of the roadway. In the experimental mine the entries average nine feet wide and about 6½ feet high. Six to eight boxes were used.

Trough Rock Dust Barrier

The 12-inch V-trough is constructed of lumber normally 1-inch thick and 12-inches wide. Two boards 12 inches wide, are nailed together at right angles as shown in Figure 52. The ends of the trough



Piece No.	Name of piece.	Material.	Number of pieces required.
1	Wooden strip	Wood	3
2	Bottom board	do	2
3	Hinge strap	Steel	2
4	Hanger	do	4
5	Release hook	do	4
6	Blocks	Wood	4
7	Suspension strap	Steel	4
8	Chain	do	8
9	Chain hook	do	4
10	Anchor bolt	do	4
11	Spacing plate	do	2
12	Cross piece	Wood	2
13	Grids	do	2
14	Blocks	do	4

FIGURE 51

From United States Bureau of Mines R.I. 2977, 1930.

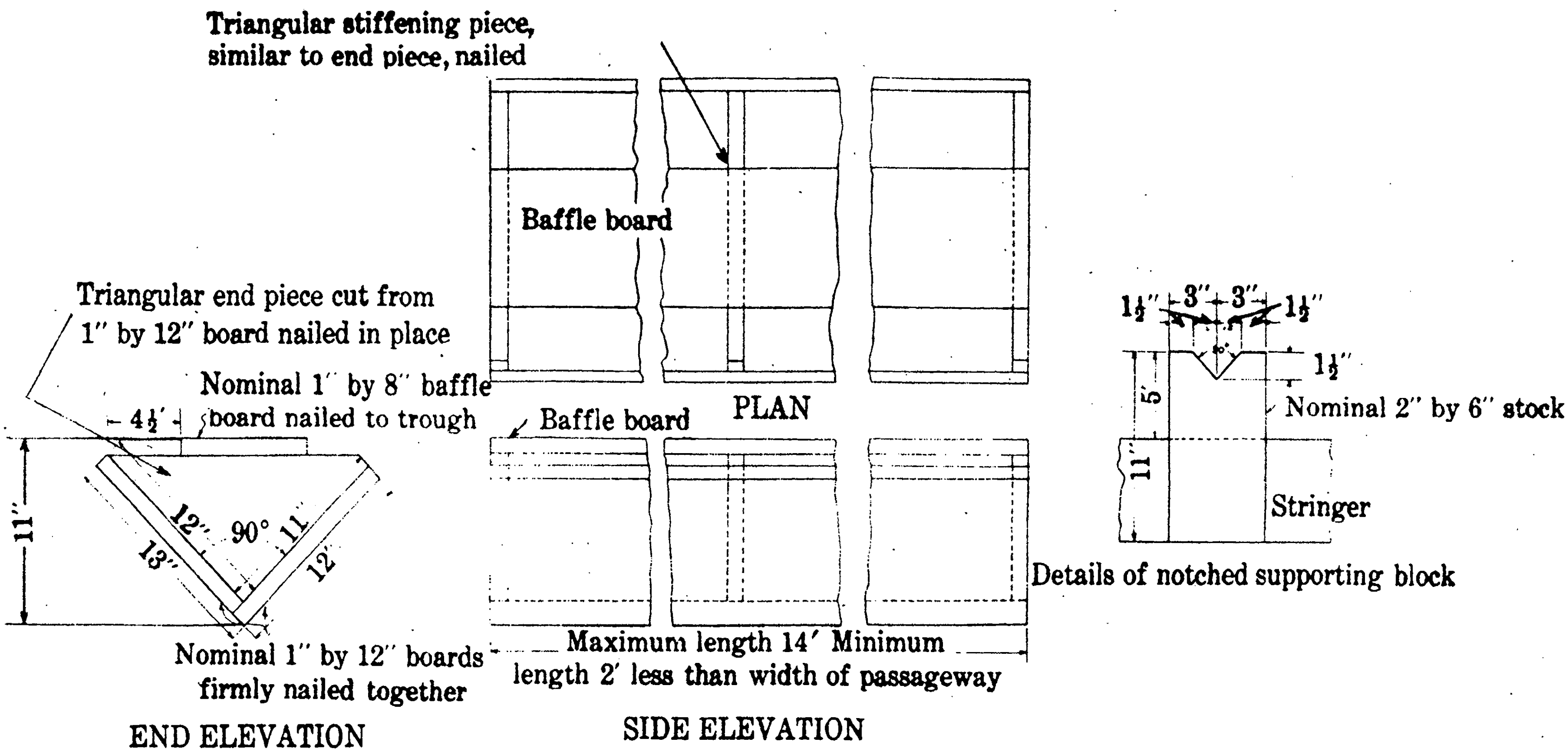


FIGURE 52

From United States Bureau of Mines R.I. 2977, 1930.

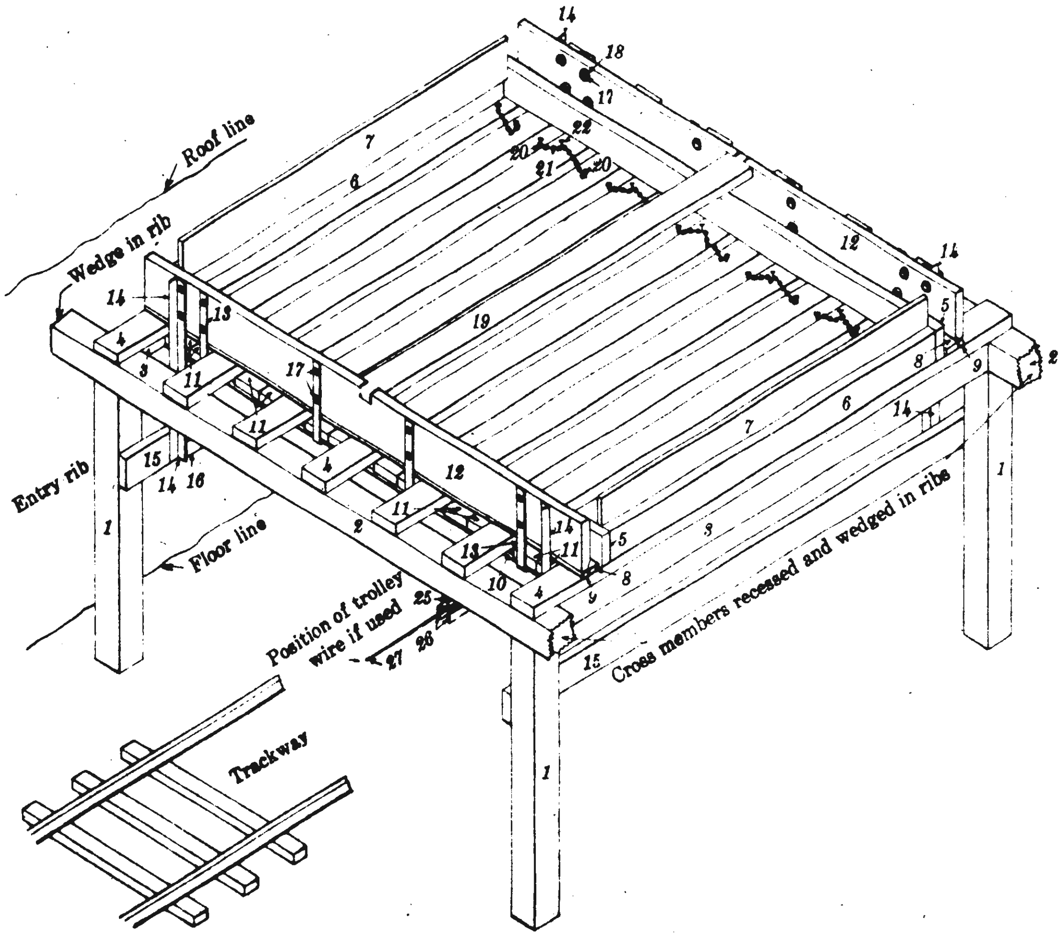
so formed are closed by triangular pieces and a similar piece is nailed in the center of the trough to strengthen it. The trough is covered by a nominal 1 by 8-inch baffle board as shown to prevent mass ejection of the dust. The maximum distance from the end of the trough to the rib is one foot; consequently, the minimum length of a trough is determined by subtracting two feet from the width of the entry.

The troughs are supported near each end by notched blocks, the details of which are shown in Figure 52. These blocks are nailed to stringers parallel to the ribs, which in turn are nailed to 5-inch posts. When round posts are used, a flat place should be cut on them to receive the stringer.

A 12-inch V-trough of this type will hold about 30 pounds of dust per foot of length when level full, that is, a 10-foot trough will hold about 300 pounds of dust.

Concentrated Barrier

Concentrated barriers, as a class, have been the most successful. They are more complicated and costly than the unit class, but they have the distinct advantage that the length of entry covered is only one-third to one-fifth as great. This is important where the roof must be taken down before the barrier is placed. Figure 53 is a general view of the concentrated type barrier for use in a 10-foot entry. It is 10 feet long and is supported by four posts (1) and two crossbars (2); these are all 6 by 6-inches. The cross-bars are hitched and wedged into the ribs to brace the barrier support against the longitudinal strains imposed by



Piece No.	Description
1	8"x6" posts
2	6"x6" cross timbers
3	2"x6"x9'-0" spacers
4	2"x6" fixed boards
5	2"x6" box frame ends
6	2"x6" box frame sides
7	1"x8" side boards
8	1"x3" axle stops
9	1" axle pipes
10	1" roller pipes
11	2"x6"x8'-0" movable boards
12	2"x10" Vane boards
13	3/8"x2" supporting stirrups
14	2"x2"x1/4"x37" reverse movement angles
15	2"x6"x10'-0" reverse stop supports
16	2" thk. reverse movement stop blocks
17	1/2"x3-1/2" sq. hd. machine bolts
18	1/2" round washers
19	2"x6"x8'-0" Vane connector
20	3/8"x2-1/2" Sq. hd. machine bolts
21	24" length 1/4" proof coil chains
22	20 d. nails
25	Guard board suspension
26	Guard boards for trolley
27	Trolley wire

FIGURE 53

From United States Bureau of Mines R.I. 2977, 1930.

an explosion and thus prevent collapse before discharging the rock dust. The end timber sets are also tied together by two 2 by 6-inch longitudinal spacer pieces (3) at the outer edge of the posts. Seven 2 by 6-inch longitudinal fixed boards (4) are nailed on top of the timber sets as shown. A box frame is formed on these fixed boards out of 2 by 6-inch stock consisting of two end pieces (5) and two side pieces (6). The height of the side is increased to one foot by nailing on 1 by 8-inch side boards (7). The spaces between the fixed boards are closed by six pairs of 2 by 6-inch movable or drop boards (11) which are immediately below the fixed boards and lap over them ¼ inch on each side. These movable boards are supported at either end by a one-inch pipe roller (10). Each roller is supported in turn by four stirrups (13) secured by bolts (17) to a 2 by 10 inch vane board (12). The vane board rests, in turn, on a one-inch pipe axle (9) which is prevented from rolling inward by a 1 by 3-inch stop strip (8) nailed entirely across the barrier. There are also fastened near the end of each vane 2 by 2 by ¼-inch angle irons (14) termed "reverse-movement irons." These angles are 37 inches long and their lower ends lie against stop blocks (16) which are nailed to 2 by 6-inch supporting timbers (15) running the entire length of the barrier. The movable boards on either side of a fixed board are connected at each end by a 24-inch length of ¼-inch proof coil chain (21) secured to the movable boards by bolts (20) which pass through the board on its longitudinal center line. The central link of the chain is dropped over a 20d nail (22) driven into the center of the fixed board. The end movable boards are fastened to the end fixed boards by half-length chains. The two vanes which are identical in arrangement with respect to the middle of the barrier are connected by a 2 by 6-inch spacer (19) which rests loosely in notches cut in the center of the vanes. The spacer is to ensure that the vanes act simultaneously to release both ends of the movable boards. The dust is placed in the box frame which will hold 4,000 pounds or more when the depth of dust is one foot, measured from the fixed boards, and the entry width is 10 feet.

Operation of all mechanical rock-dust barriers depends upon the pressure of air forced ahead of an explosion. For this, vanes are essential for the proper functioning of the barrier. The vane facing the pressure is tilted inward and hinges at the axle pipe; the supporting stirrups tilt outward and the roller pipe rolls out from under the movable boards which fall six inches and are then caught and held by the chains. The movement of the front vane facing the explosion is communicated to the rear vane at the other end of the barrier by the spacer. The rear vane, by the effect of the thrust of the spacer, hinges at the lower end of the reverse movement irons and rolls off the axle pipe, at the same time pulling the hooks out from under the supporting roller pipe, thus releaseing the rear end of the moveable or drop boards.

Qualities of a Successful Barrier

To be successful, a barrier must have the following qualities: (a) installation of a barrier and its subsequent operation, accidental or otherwise, must not obstruct the passageway sufficiently to cause an appreciable reduction in the volume of air flowing therein; (b) barriers must not cause injury to men or damage to property if set off accidentally during normal operation of the mine. They should not be located low enough to strike passing cars or to reduce greatly the height of the passage under the barrier; (c) a barrier must be designed to retain one-fourth to one-third of its loading when it is operated by a shock wave before the arrival of an explosion; (d) a barrier must be so designed that it can be placed close to the roof, and it must extend practically across the passageway in which it is placed so that there are no considerable gaps through which flame may dart; (e) a barrier must be of such design that in entries where the air is humid, the rock dust loading can be covered to protect it from moisture. The covering must not interfere with the operation of the barrier.

Kind and quantity of barrier dust. Any rock dust that meets the specifications of the Bureau of Mines for size of the dust and which does not absorb moisture and cake when exposed to a nearly saturated atmosphere will be found satisfactory for barrier purposes. The dust used in the barriers for tests in the experimental mine was limestone, about 70 percent of which would pass a 200-mesh sieve. The specifications set by the U. S. Bureau of Mines require that all particles must be minus 20-mesh and at least 50 percent pass 200-mesh.

The quantity of dust required in a barrier to stop an explosion varies widely with the characteristics of the explosion; however, the quantity recommended by the U. S. Bureau of Mines is based on the amount which is necessary per square foot of cross-sectional area of the entry at the barrier location. As a result of the above testing, and based on the assumption that the barriers will be used in conjunction with general rock dusting, the quantity recommended for the loading of the barriers is 60 pounds per square foot. A higher quantity of about 100 pounds per square foot should be used for places in mines which are not rock-dusted at all.

It is also possible that the quantity of barrier dust required per unit area may vary widely with the size of the entries, but no experimental evidence of this could be obtained.

Protection against moisture. Damp and wet dust cannot be dispersed into a cloud, hence, such dust should not be used. When a barrier is to be installed, moisture conditions at the proposed location should be investigated. If water is dripping from the roof, the position is entirely unsuitable unless a waterproof cover can be placed so as to protect the dust.

Certain passageways of a mine may be damp or wet during some seasons of the year and dry during others, depending upon temperature changes. Return airways from faces at which water is used during mining operations are quite likely to be damp or wet. In such places it is better and possibly cheaper to protect the barrier dust than to keep it in good condition by frequent renewals. A covered concentrated barrier is best for such locations.

<u>Locations of barriers</u>. No hard and fast rule can be laid down for the location of barriers in a mine. They are intended to divide the mine into sections so that flame cannot pass out of one section to another. Any system of mining that provides only a small number of openings for a considerable area of workings, such as panel system, can be readily sectionalized. On the other hand, a system that provides a large number of parallel paths, for example, where rooms are driven through from entry to entry, presents little hope for effective use of barriers. When two or more parallel entries are to be protected, barriers must be installed in all of them directly opposite each other; that is, between the same crosscuts or side entries. When barriers are used to protect main intersections, they must be installed on all sides thereof, as the direction of travel of the flame cannot be foretold and all possibilities must be provided for.

In mines where mechanical loading is used on advancing entries and no water spray is employed, fine coal dust in naturally dry mines is carried into the return air course in such quantities as to make it difficult if not impossible to maintain proper rock dusting. Barriers may be of particular advantage in such places, An excellent arrangement is to place a barrier immediately outby the second crosscut or cut-through from the face and a second barrier two cut-throughs further out. When the entry has advanced two additional cut-throughs, the outer barrier should be taken down and re-erected just outby the then second cut-through from the face. In this way, protection is advanced as the entries advance.

Rock-dust barriers are not effective in stopping fire-damp explosions. The Bureau does not recommend barriers as a substitute for general rock-dusting in preventing coal-dust explosions, but only recommends them as additional defenses in stopping incipient explosions of coal-dust within a section of a mine.

Bag-type Rock-Dust Barrier

To protect trackless entries and other areas where it is difficult to maintain generalized rock dusting, bag-type rock dust devices have been used in a few coal mines. (18, p. 15) Advantages claimed for these devices include the following: (a) they afford good supplementary protection in back entries; (b) they provide added protection against fresh deposits of coal dust; (c) they can be installed easily and quickly on shift, with little discomfort in breathing; (d) they make rock dust readily available for fire fighting; (e) rock dust does not become wet from contact with moisture; and (f) bag type installation can easily follow rapid advance of the coal face.

Tests with various modifications of these devices showed that one of their drawbacks was inadequate dispersion of the rock dust into the air at the proper moment. To overcome this, a burster, consisting of a charge of permissible explosive, was incorporated in the bag-type units. In a few tests the explosive charges were triggered by a manually timed switch, and in others a pressure sensitive or a flame sensitive device was initiated by the explosion itself. The tests indicated that a properly designed system of bag type units, equipped with bursters for dispersing the dust, was capable of arresting coal dust explosions.

Figure 54 gives the dimensions of the supporting board, the trigger wire, and the vane. It also shows the manner in which a rip wire is threaded through the bag of rock dust. The supporting board is of nominal 1 by 6-inch stock with holes bored as indicated. The trigger is made of No. 8 steel wire and is in two parts with connecting loops that act as a hinge. The experimental diameter of the loops must be small enough to permit them to pass freely through the single 3/4-inch hole in the right hand end of the board, as shown in the figure. The bend in the horizontal portion of the trigger wire has a definite purpose. Upon assembly, the weight of the supporting board acting on the bent-up portion forces the entire wire to the right and binds the hook on the left and firmly in the middle hole in the board.

The rip wire should be carefully inserted in the bag. A long stiff wire with an eye in one end serves as a needle for inserting the rip wire, which should be started at A and pulled out at B, started in again at B and pulled out at C, and finally started in at C and pulled out at D. The ends at A and D and the loops at B and C are necessary for operation. The rip wire must not be passed through portions of the bag where the paper has been doubled back and glued or otherwise fastened, as it will not rip through the extra plies of material. It may be noted that 50 pounds of rock dust is loaded in each bag.

Erection of a barrier is begun by placing the posts and cap pieces. Nails are driven part way into the post five inches below the cap and three inches apart and the ends are then bent vertically with a monkey wrench or other convenient tool. Two holes in one end of the

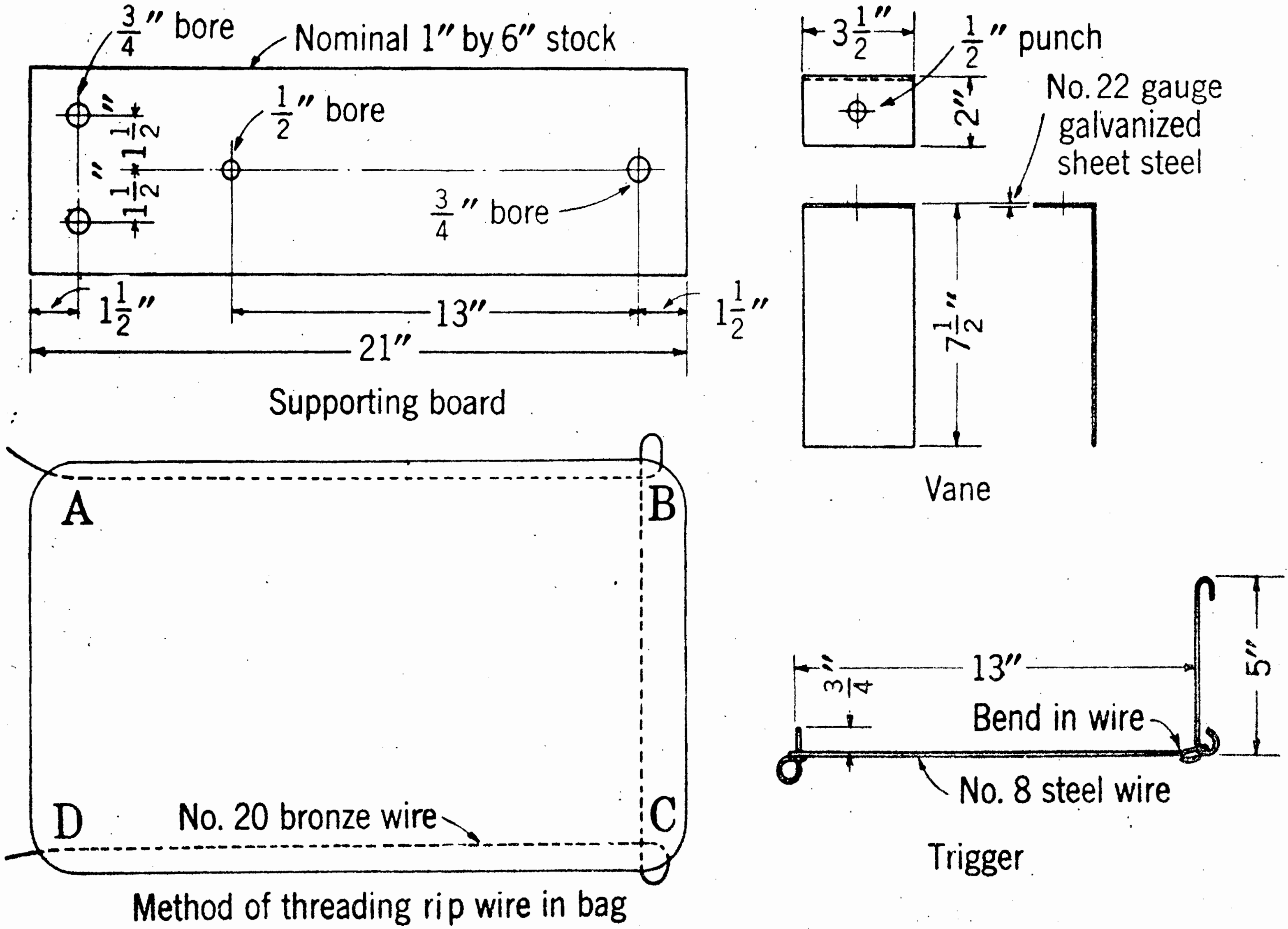


FIGURE 54

From United States Bureau of Mines R.I. 3411, 1938.

supporting board are slipped over the nail heads and the vertical portion of the trigger is passed upward through the single hole in the opposite end of the board and fastened to a staple or bent nail in the cap piece. When released, the board will hang down at an angle, being caught by the projection and the loop at the opposite extreme of the trigger. The bag, with rip wire inserted, is placed on the board with the rip wire side up and the loops in the wire away from the post. The board is raised to a horizontal position and the trigger bent at its hinge so that it lies along the under side of the board. The hooked end passes through the hole in the short leg of the vane and is pushed upwards into the hole near the middle of the board. Finally, the ends and loops of the rip wire are pulled taut and nailed to the cross piece.

A unit operates when enough pressure is placed on the long leg of the vane to rotate it and force the trigger wire out of the middle hole in the board. The board then falls until caught by the end of the trigger. At the same time the rip wire tears the top out of the paper bag, part of the rock dust being spilled immediately, and part being retained on and in the remaining portion of the bag. Figure 55 shows the general view of the barrier.

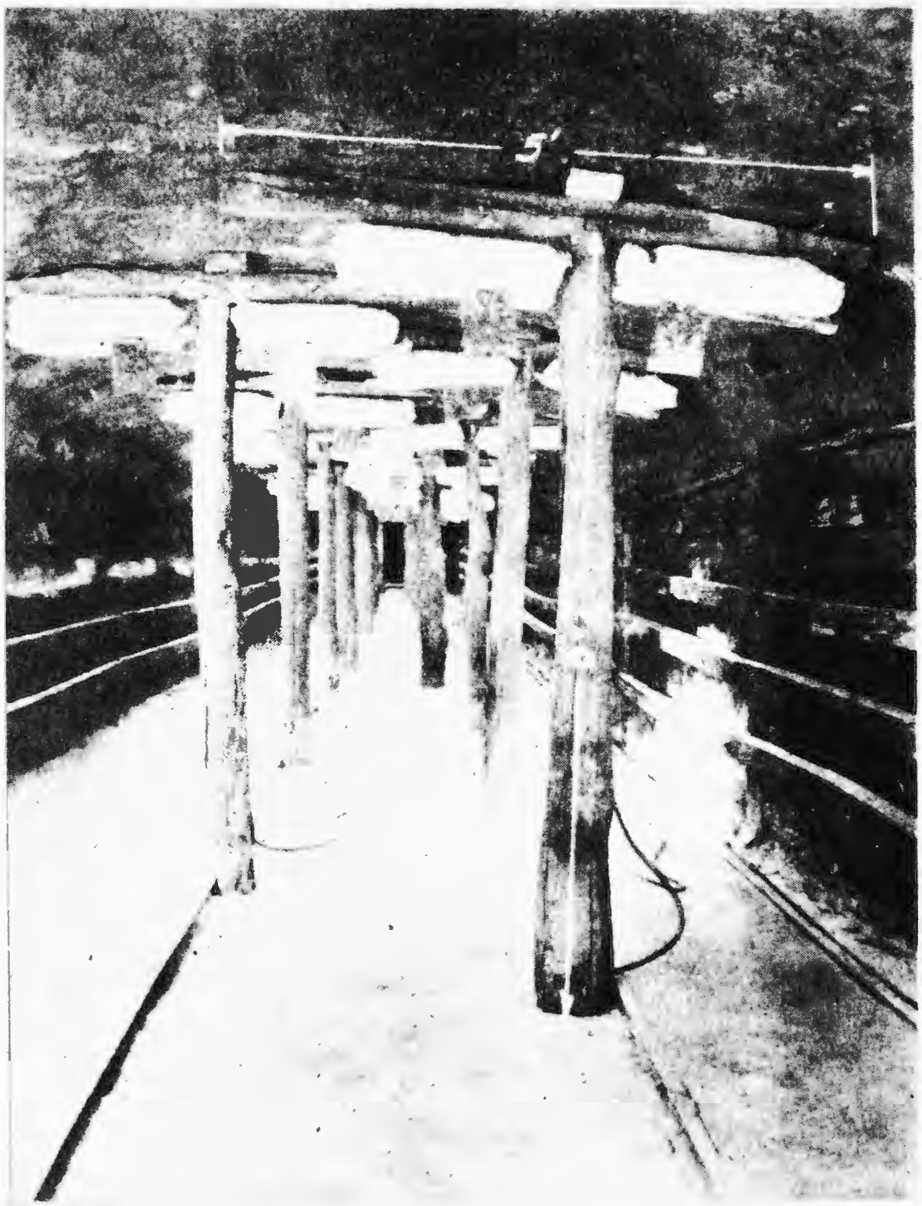


FIGURE 55

From United States Bureau of Mines R.I. 3411, 1938.

Concrete Stoppings to Resist the Pressure of Explosions

General

Stoppings in coal mines serve important and vital functions such as coursing the ventilating current, sealing off abandoned or dangerous workings, sealing off fire areas, and separating mines at boundaries. Stoppings are designed primarily to resist pressures applied laterally through air or gases. Their strength cannot be stated without specifying the direction of applied load. A stopping is principally designed to resist the action of forces at right angles to its face, such as the shocks of blasting, the pressure of gas from instantaneous violent outbursts, sudden compression of the air by falls of roof in closed spaces, and pressures that accompany an explosion of fire-damp or coal dust or both. (56, p. 1).

A concrete stopping, well buttressed into solid coal ribs, has a much greater strength than that indicated by computations made according to formulas for the strength of plain concrete beams. Concrete has little strength in tension. Such a stopping is structurally more nearly a flat arch than a beam as movement of the ends is generally restrained. As a result of this, the central portion of the face to which the pressure is applied is thrown into compression and stoppings cannot fail until the concrete crushes to a depth that may be about one-fifth of the thickness.

Tests in the Experimental Mine

As the strength of stoppings depends largely upon the resistance to compression offered by the coal strata, tests were made to obtain direct information on the compressibility and crushing strength of the Pittsburgh coal bed in the experimental mine. Figure 56 shows the plan of the test chamber. The permanent stopping was pierced by a manhole which could be closed by a steel-plate door. The space between the permanent and test stoppings formed the chamber in which black blasting powder was exploded. The figure also shows the location of stopping No. 2. The succeeding stoppings were placed at adjacent points, and the passageway was widened by excavating the ribs as occasion demanded.

A complete time-pressure record was obtained and the deflection of the stoppings was measured for every test. The deflection measurements included: (a) movement of the outer face parallel to the applied pressure at a number of points; (b) permanent set of the outer face, that is, the difference between the location of the points before and after each test; (c) movement of the ends of the stopping parallel to its length; (d) maximum width of opening of the vertical central tension crack in the outer face which is the first visible sign of stress and appears at low loads; (e) permanent opening of this crack; (f) depth of this crack; and (g) permanent set of the central portion of the inner face.

Construction of Stoppings

The concrete was 1:2:4 machine mixed. Only enough water was used

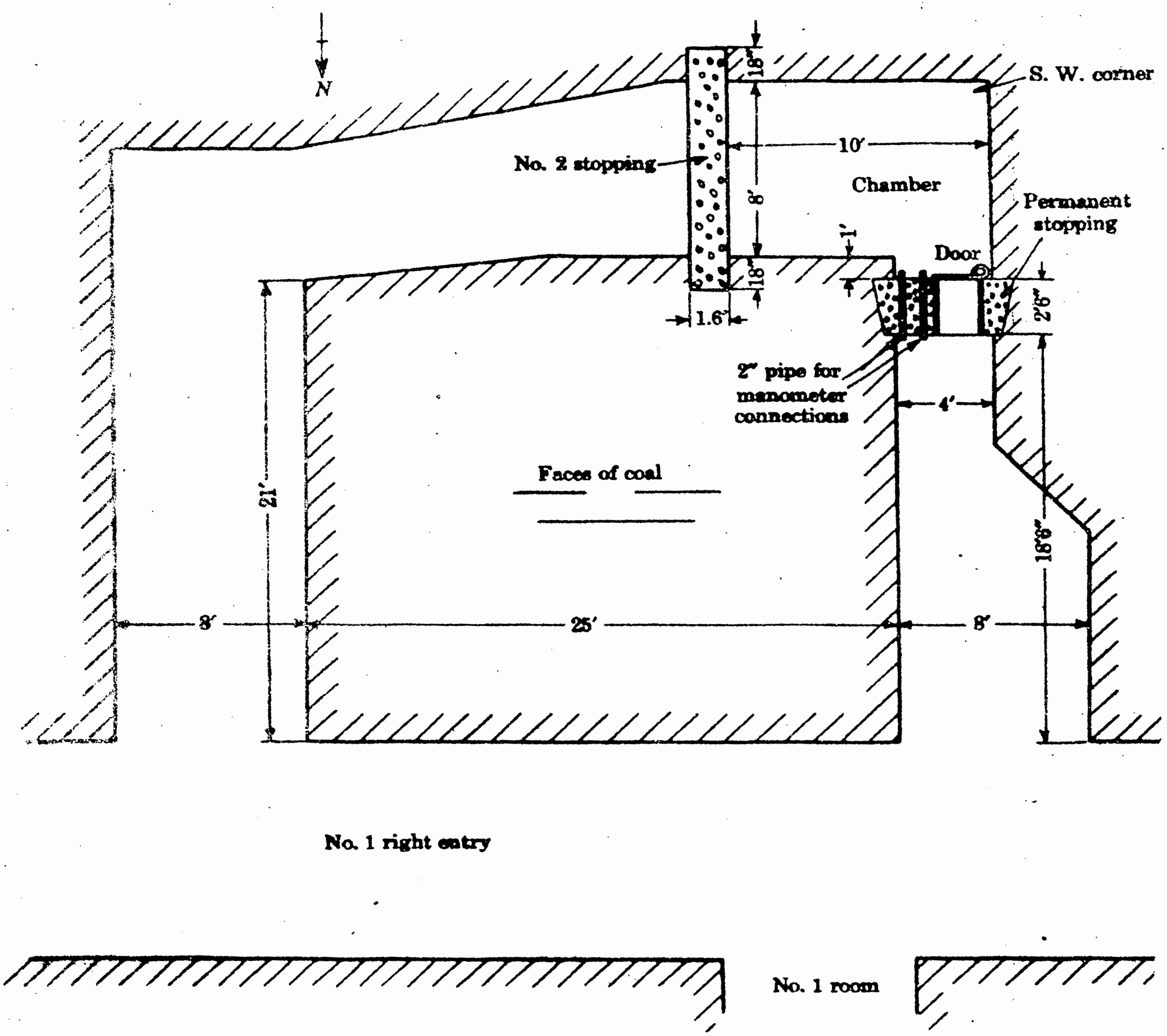


FIGURE 56
From United States Bureau of Mines R.I. 3036, 1930.

to allow the concrete to work well in the mold. It is well known that excess water in concrete weakens it and that the mixture should be as stiff as can be properly worked and tamped in the form.

In actual construction the ribs are first trimmed properly, and the recesses are then dug. The floor is prepared as required in the various tests, and when the concrete used thereon has set, the inner form is built up to the roof flush with the inner edges of the recesses and extending entirely across the passageway. All the concrete for each stopping is placed with no pause in a single day. The concrete is well spaded as it is placed in the form and care is taken to see that the recesses are completely filled. When steel plates are used, the top of the form is kept about six inches from the roof; the concrete is filled into this level and allowed to set. After the form is removed, a layer of mortar is placed on the concrete and the greased plate laid on it. The space over the plate is filled with concrete as far as possible and the job completed with the cement gun. The plate is also fastened to the roof by bolts set in stiff grout in holes bored to recieve them. After the concrete has set, the supports for the deflection indicators are set and the instruments are placed. Figure 57 shows the deflection and permanent set of a stopping under pressure.

<u>Summary of Tests</u>

(a) With an explosion pressure, such as is caused by black blasting powder, the sudden or impact pressure produced is no more severe in disruptive effects on the stopping than static pressure caused by compressed air applied gradually or by hydraulic means; (b) the strength of

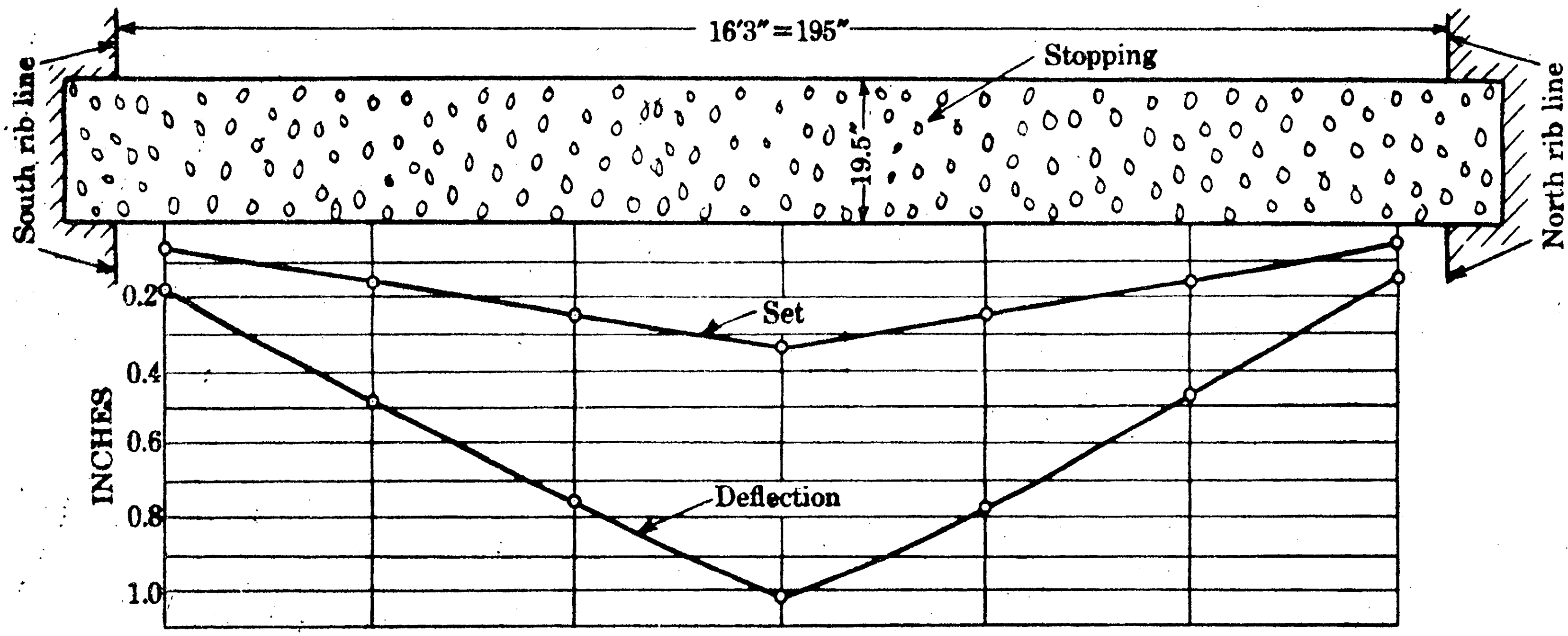


FIGURE 57

From United States Bureau of Mines R.I. 3036, 1930.

a massive concrete stopping is enormously increased by buttressing it tightly against solid walls so that it acts as a flat arch; and (c) a massive concrete stopping buttressed into solid ribs and receiving no support from the roof must have a ratio of thickness to span of 1:10 to just resist an explosion pressure of 50 pounds per square inch; that is, if a stopping is 10 feet wide from rib to rib, it should have a thickness of one foot. (56, p. 11)

Effect of Relief Vents on Reduction of Pressures Developed by Dust Explosions

Description of Gallery-Test Equipment (30, p. 5)

The gallery in which the tests were made is cubical, 4 by 4 by 4 feet in inside dimensions or 64 cubic feet in volume. It is constructed of two-inch tongue-and-groove pine planking, reinforced with steel angles along outer edges and lined on the inside with galvanized-steel sheet.

The gallery has four adjustable opening or vents, one in the top and one each on three sides. The maximum opening of each vent is 18 by 30 inches. When smaller vents are desired, portions of the openings can be closed by ¼-inch steel plates. The fourth side of the gallery has a door set on vertical hinges which, during explosion tests, is held tightly against a sponge-rubber gasket by ½-inch steel bolts. The entire gallery is bolted to steel beams anchored to the floor of a shed in which the gallery is housed.

The dust used for forming the dust cloud in the gallery is placed in four 6-inch diameter hemispherical cups set on the same level above the gallery floor. Dispersion is accomplished by directing a blast of compressed air downward into the cups through gooseneck-shaped pipes. Ignition of the dust clouds was obtained in some tests by a high voltage continuous induction spark across a 3/8-inch gap, and in others by the timed flame of 15 grams of dry guncotton in an open glass vial.

Pressure-time records of all explosions were obtained by means of manometers containing a film on a revolving drum. Later a new type of manometer was used. Figure 58 shows the sketch of the manometer. This instrument can measure pressures of the order of 0.1 to a few pounds per square inch. In this device the diaphragm is replaced by a calibrated spiral spring placed within a thin metal bellows seal. The pressure acting on the closed end of the bellows compresses the spring and actuates a light sliding pin whose motion is magnified optically. This arrangement gives a linear pressure-deflection calibration, and it is not affected by temperature changes. An opposing spring outside the bellows keeps the latter in neutral position when no explosion pressure acts on it. All controls are in a building several hundred feet distant from the gallery.

To ensure safety of the operating personnel while the gallery is being cleaned and other preparations are being made, all electrical circuits leading to the gallery are grounded, except one used for communication via a phone between the control room and the gallery shed. No dust is permitted in or near the gallery at this time, and no smoking or matches are allowed at any time.

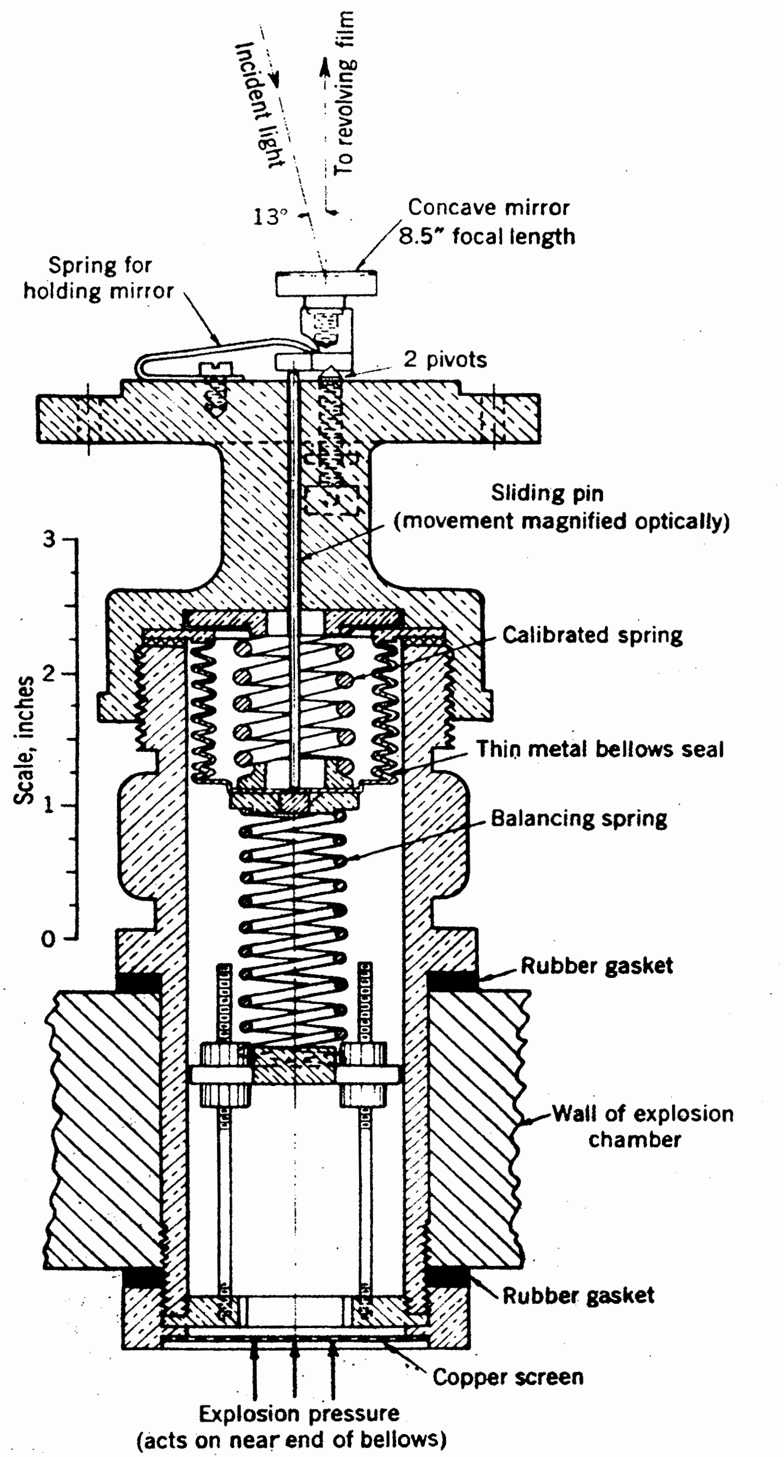


FIGURE 58

From United States Bureau of Mines R.I. 3924, 1946.

When the gallery is ready for the test, the air pressure in the reservoir is adjusted to the desired value by means of a regulator. This is the air used for dispersing the dust in the gallery. Then connections to all the circuits are made in the control room. To close the circuit for the ignition spark, a switch must be closed in the control room and simultaneously a second push button switch must be held closed about 40 feet from the gallery. The latter is an added safety measure to assure that no one is near the gallery during the explosion. After the circuits have been tested, all wires on the control board are grounded, the control box is locked, and the weighed dust samples are brought to the gallery.

The manometers are set, and the dust is placed into the hemispherical cups. When guncotton is used for ignition, this is set in position, then the door is shut and bolted tight. The doors and panels on the gallery shed are opened and everyone goes outside a fenced enclosure surrounding the gallery, about 25 feet distant. When the gates are closed, a warning siren blows for a short time. Then the mercury-switch timing control which automatically makes and breaks all circuits in a prearranged sequence, setting off the explosion and obtaining pressure-time records, is started. Figure 59 shows the effect of unrestricted relief vents on pressure produced.

Various types of vents were tried, and common types being: (a) unrestricted or free openings; (b) hinged or pivoted sashes, which swing outward at a predetermined internal pressure; (c) fixed sashes with light wall anchorages; (d) scored glass panes; (e) light wall panels; (f)

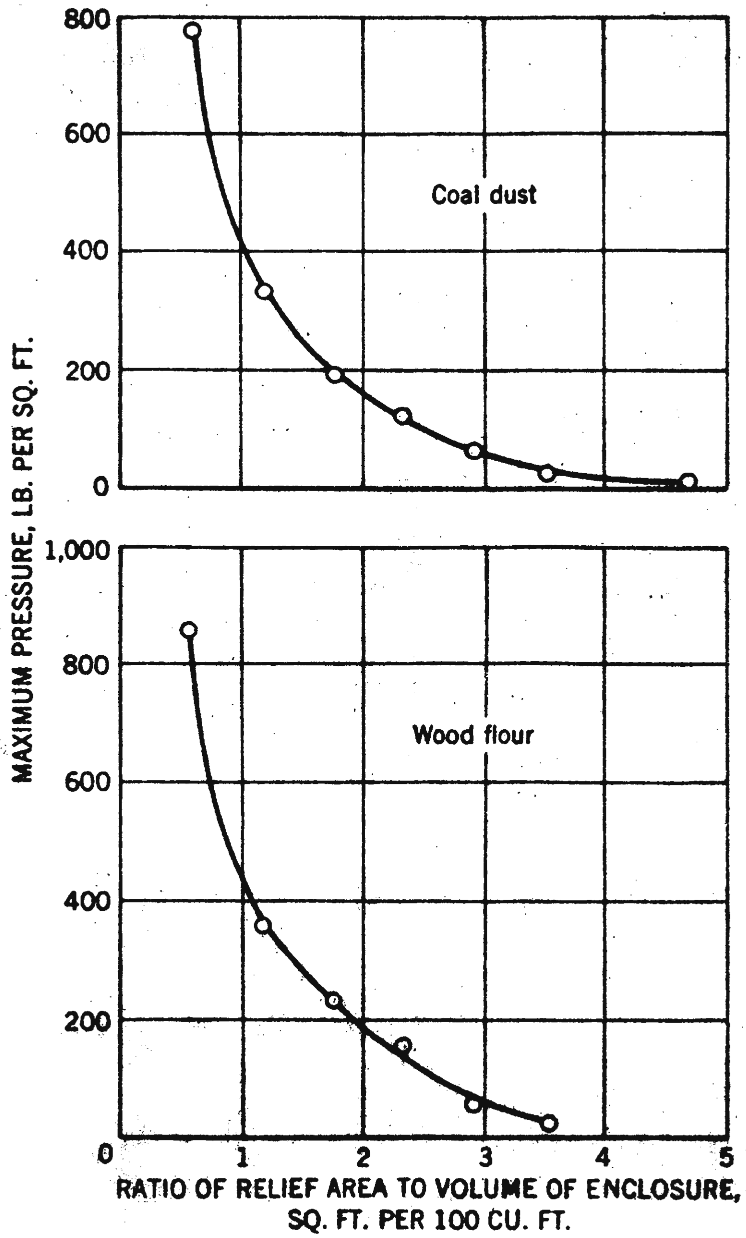


FIGURE 59

From United States Bureau of Mines R.I. 3924, 1946.

poppet-type vent closures; and (g) paper, metal foil, or other diaphragms that burst at low pressures.

Much additional development work is needed on the subject and it is hoped that it will be possible to recommend correct venting for most operations in which a dust explosion hazard exists.

SUMMARY AND CONCLUSIONS

It is practically impossible to prevent the formation of coal dust at the working faces of a coal mine, particularly in modern mechanized mines where production and dissemination of fine coal dust in mine entries have greatly increased. The distribution and dispersion of dust, however, can be considerably reduced if proper preventive measures are adopted.

The properties of coal dust which have an important influence on its explosibility are; fineness, purity, percentage of volatile matter, dryness, age and degree of oxidation, lifting velocities and dispersability, and its static electrification.

Explosibility of coal dust and methane has been studied in laboratories, in explosion galleries, and in an experimental coal mine. The data and results obtained from the tests are the following:

1. The upper and lower ignition limits of methane are 14.8 and 5.3 percent, respectively, and an explosion of 150 cubic feet of a mixture of gas and air is sufficient to initiate an explosion of Pittsburgh coal dust.
2. Coal dust has a lower explosive limit of 0.035 to 0.08 ounces per cubic foot.
3. Explosibility of coal dust increases with the increase in its volatile content.
4. Considering the very small amount of coal dust necessary to cause an explosion, and because it is not possible to prevent formation of the dust, every part of a coal mine should be considered as a potential source of coal dust explosion.

Due to the varied mining conditions and techniques, it is not possible to set a general safety rule which will be applicable to all mines. Each problem has to be studied carefully and the most applicable preventive measures should be strictly followed.

Application of water during the various phases of mining is an effective means in many instances for reducing the quantity of fine coal dust produced during mining processes. Among the different methods of using water, its injection into the coal seam before mining is successfully applied in Australian and European coal fields but the results obtained from the tests in the Experimental Mine show that water infusion in the Pittsburgh coal seam has little, if any, beneficial effect on dust reduction.

The most practical measure for preventing the propagation of coal dust explosions in mines is to rock dust the mine workings. The requirement of 65 percent of incombustible in mine dust provides a considerable factor of safety against coal dust explosions. It gives no guarantee that an explosion will never occur but it does give assurance that if the rock dust has been properly applied and maintained, wide-spread explosions will not develop. Apart from the foregoing methods, some other preventive measures which can be adopted include rock dust barriers, concrete stoppings, and relief vents in their various forms and modifications.

Although intensive studies and research have been and are presently being conducted in this and many other countries, there is still a vital need of further research to improve present measures for controlling explosions.

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VITA

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In August, 1952, he joined the Banaras Hindu University as a Lecturer in Mining Engineering and is still continuing in the same capacity. In 1956 he was awarded the Mine Managers' Certificate of Competency, Class II, by the Department of Mines, Government of India. The Banaras Hindu University granted him study leave and, being awarded a scholarship under the Technical Cooperation Mission program, he came to the United States in September, 1959 to join the School of Mines and Metallurgy, University of Missouri, as a graduate student in Mining Engineering.