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UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

District C

FINAL REPORT ON MAJOR EXPLOSION AND FIRE DISASTER
NO. 9 MINE
JAMISON COAL AND COKE COMPANY
FARMINGTON, MARION COUNTY, WEST VIRGINIA

November 13, 1954

By

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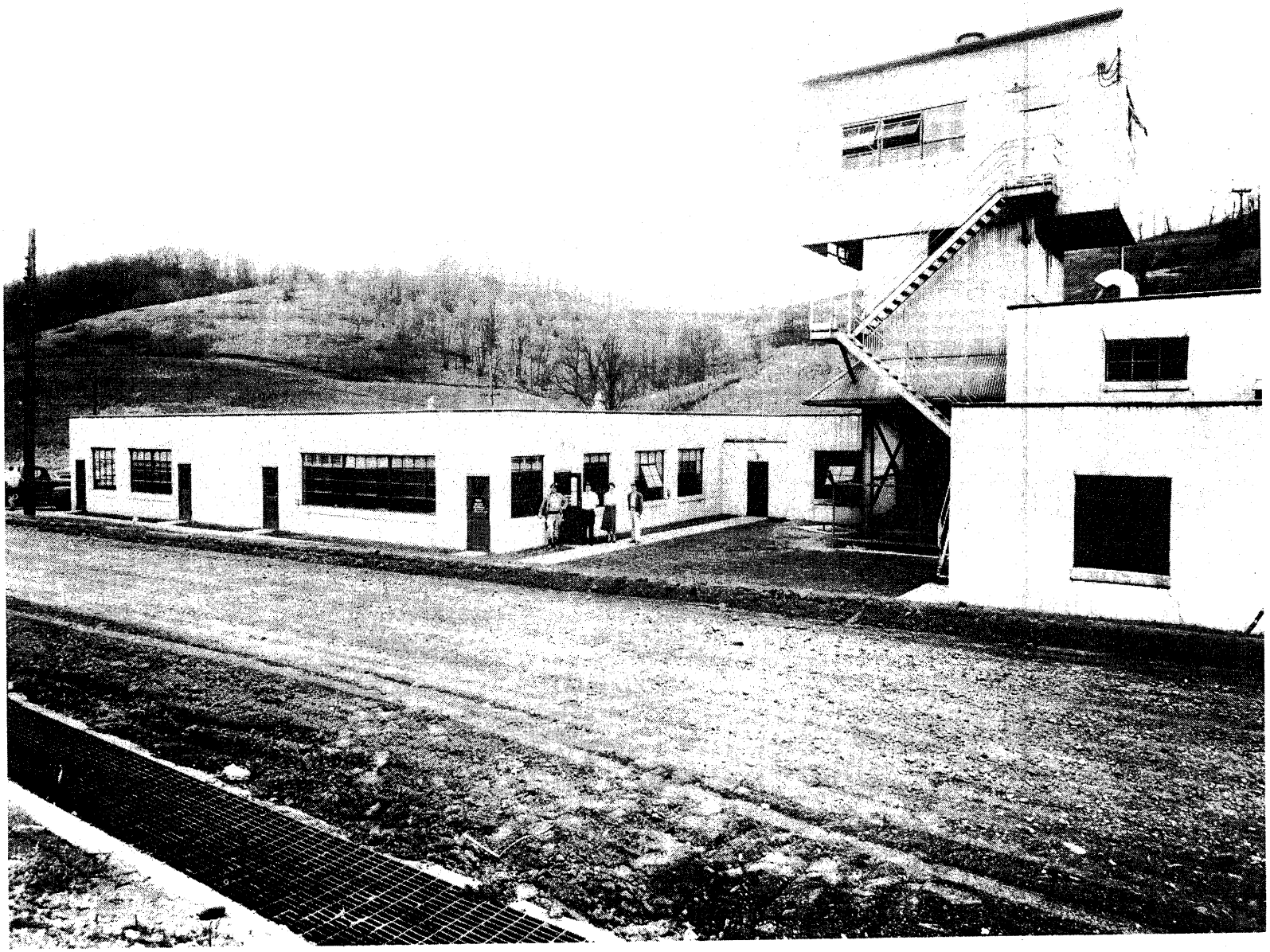

Figure 1. - Surface buildings at man shaft before explosion.

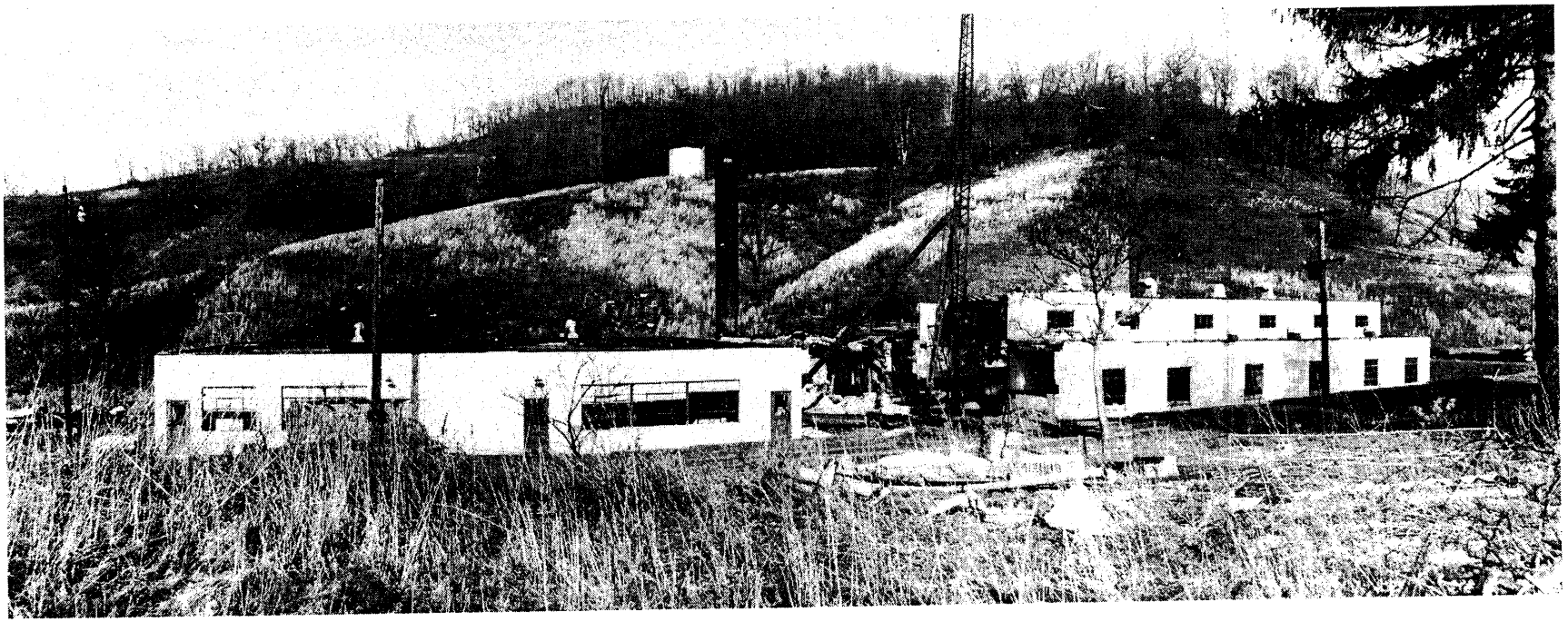

Figure 2. - Surface buildings at man shaft after explosion.

Figure 3. - No. 2 fan after explosion.

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INTRODUCTION

A gas and dust explosion followed by a series of mine fires and explosions occurred in the No. 9 mine, Jamison Coal and Coke Company, near Farmington, West Virginia, about 1:45 p.m., Saturday, November 13, 1954, and caused the death of 16 persons. Two of the seventeen men in the mine at the time escaped injury and made their way to the surface unaided; 12 were killed instantly by burns and violence; and 3, who lived for perhaps 90 minutes after the initial explosion but made no attempt to erect a barricade, died from the effects of afterdamp. The sixteenth victim, who was employed as a lampman in a room adjacent to the man-shaft portal, was killed instantly when struck by a steel structural member dislodged by the explosion.

The names of the victims, their ages, marital status, occupations, and the number of their dependents are listed in Appendix A of this report.

A mine rescue team accompanied by company, State, and Federal officials entered the mine about 4 hours after the initial explosion, and a second team entered within the next 30 minutes. Two hours later, all the men were withdrawn when the methane content of return air at the one fan that remained in operation indicated a dangerous condition underground.

A second explosion occurred in the mine about 10:30 p.m., the same day; a third, at 4:10 a.m., November 15; and a fourth explosion of somewhat less violence, about 5 minutes later. Minor explosions, forces of which were not noticeable on the surface, may have occurred underground

during the same period. Except for the 15 men entrapped and/or killed by the first explosion, no one was underground at the time nor was anyone on the surface injured by the subsequent explosions.

Hoisting equipment in and above the man shaft and one of the two main fans (No. 2 fan) were destroyed by the first explosion, and thereafter dense smoke continued to pour from this fan shaft. Following the second explosion, dense smoke poured from either or both of these shafts and was being drawn from the mine by the No. 1 fan, and smoke and fumes containing carbon monoxide began to flow from the slope portal, the only means of ingress. In a conference of responsible officials of the company, the United Mine Workers of America, the West Virginia Department of Mines, and the Bureau of Mines, convened at 12:30 a.m., November 14, it was decided to seal the mine at the surface openings.

Sealing of the mine at 5 surface openings was first completed at 1:30 a.m., November 15, but 2 of the seals were broken by the third and fourth explosions, and final sealing was effected at 2:00 p.m., November 17, 1954. The first seal, at the No. 1 fan shaft, was removed at 12:15 p.m., March 10, 1955. The No. 1 fan was put in operation shortly thereafter, and the No. 2 fan was started at 9:38 p.m., March 12. Underground recovery operations, begun March 14, were retarded by a fire which rekindled near the man shaft; the fire was discovered March 19, and required sealing of the affected area. Recovery work had not been completed at the time this report was released; however, production of coal was first resumed, in 2 of the 6 working sections, July 24, 1955.

Bureau of Mines investigators believe that the initial explosion originated in 4 left section of 2 north when an explosive mixture of methane-air was ignited by blasting in a nonpermissible manner and that the explosion was propagated by methane and coal dust. Forces of the first explosion were the greatest; this was the only one that was recorded by a seismograph at the University of West Virginia in Morgantown, about 17 air-miles distant. The duration of the tremor is recorded as 17 seconds. Evidence of forces of this explosion extended to the surface through the man shaft, No. 2 fan shaft, the slope, and Plum Run borehole, and windows were broken and pictures jarred from the walls in houses on the surface in Plum Run.

GENERAL INFORMATION

The No. 9 mine of the Jamison Coal and Coke Company is about 2 miles north of Farmington, Marion County, West Virginia, and it is served by the Baltimore and Ohio Railroad. The majority interest of the capital stock of the Jamison Coal and Coke Company was purchased by the Pittsburgh Consolidation Coal Company shortly before the date of the disaster. The operating officials of the Jamison Coal and Coke Company on November 13, 1954 were:

R. E. Jamison	President	Greensburg, Pennsylvania
W. B. Jamison	Vice President	Greensburg, Pennsylvania
George W. McCaa	General Manager	Farmington, West Virginia
George Cain	Superintendent	Farmington, West Virginia
John M. Neer	General Mine Foreman	Farmington, West Virginia

A total of 443 men was employed; 355 of them worked underground 3 shifts a day and produced an average of 6,000 tons of coal daily. Most of the 88 surface employees were engaged in construction work. Production for the year 1954 was 970,415 tons of coal. The last Federal inspection of this mine prior to the disaster was made October 6, 8, 11-14, and 18-20, 1954.

The mine was opened in 1910 by the Jamison Coal and Coke Company into the Pittsburgh coal bed, which averages 96 inches in thickness in the area being mined and dips about 1.5 percent to the northwest. Four shafts and a slope provide access to the mine workings. A fifth shaft, an original opening into the mine, was abandoned and filled in 1953. The shafts are 334 to 591 feet in depth, and the slope is 1,468 feet in length on 16° dip. Two of the shafts, near which the main fans are located, serve only as return airways; one serves only as an intake airway, and the other, nearest the active workings, serves as a man shaft and an intake airway. All coal is transported to the surface by means of a conveyor belt installed in the slope. Necessary use of the slope for travel into and out of the mine slowed initial entrance by recovery workers immediately following the explosion and for a few days in March 1955, when final recovery operations were begun.

The immediate roof is usually 12 to 14 inches of coal left to support 10 to 30 inches of draw rock; the main roof is shale and sandstone. Occasional slips, rolls, horsebacks, and clay veins are encountered. The cover ranges from 300 to 800 feet in thickness. The floor is medium soft fire clay, 2 to 6 feet in thickness. The analysis of a coal sample from the Pittsburgh coal bed in this mine. as listed in Technical Paper 626, "Analyses of West Virginia Coals," published by the United States Department of the Interior, Bureau of Mines, is as follows:

	Percent
Moisture	2.1
Volatile Matter	36.8
Fixed Carbon	54.4
Ash	6.7
	100.0

Numerous tests by the Bureau of Mines have shown that coal dust having a volatile ratio of 0.12 is explosive and that the explosibility increases with an increase in the volatile ratio. The volatile ratio of the coal in this mine as determined from the above-mentioned analysis is 0.40, indicating that the dust from this coal is highly explosive.

A fire or an explosion involving loss of life had not occurred previously in this mine; however, a gas explosion, said to have been caused by firing an unconfined shot in loose roof material, happened therein on October 11, 1951; and an explosion resulting in the death of 19 men occurred January 14, 1926, in the Jamison No. 8 mine, which adjoins and had been connected to the No. 9 mine. Other major mine explosions that have occurred in nearby mines include:

Mine	Date	Location	Lives Lost
Chatham	May 15, 1901	Farmington, W. Va.	10
Monongah Nos. 6 & 8	December 6, 1907	Monongah, W. Va.	361
Jamison No. 7	October 19, 1916	Barrackville, W. Va.	10
Barrackville	March 17, 1925	Barrackville, W. Va.	33
Federal No. 3	April 30, 1927	Everettville, W. Va.	97
No. 1	June 20, 1928	National, W. Va.	7
Yukon	March 26, 1930	Arnettsville, W. Va.	12
Christopher No. 3	May 12, 1942	Osage, W. Va.	56
No. 2	July 9, 1942	Pursglove, W. Va.	20
Katherine No. 4	March 25, 1944	Shinnston, W. Va.	16
Bunker	October 15, 1951	Cassville, W. Va.	10

MINING METHODS, CONDITIONS, AND EQUIPMENT

Mining Methods. A block system of mining was followed. Multiple entries in sets of 7 to 9, turned at various intervals, were driven 14 to 16 feet wide on approximately 70-foot centers, and crosscuts were about 80 feet apart. Pillars were generally recovered by a pocket-and-fender method. Pillar lifts were 18 to 21 feet in width and each 8- to 8-1/2-foot cut was double-sheared, one shear in the middle and the other angled toward and cut through to the gob. With the first cut in each lift taken from the entry side of the pillar, a triangular stump about 6 feet wide at the outby end was left for support at the corner nearest the gob. With each succeeding cut except the last, the triangular stump left was 3 to 4 feet in width at the outby end, and a rectangular stump 3 feet in width was left in the middle of the last cut. Shot holes in these 6 or 7 stumps were drilled generally as the face was advanced, they were blasted on retreat, and the coal therefrom was not recovered.

Bolts were used for roof support in face areas and along recently developed haulageways and air courses; they had also been installed at various places along older haulageways, air courses, and other places in the mine. With few exceptions, bolts were installed in full compliance with the recommendations of the Bureau's roof-control representative and State roof-bolting permit No. 619. Wooden timbers were used for breaker posts in pillar sections and to supplement roof bolts where abnormal roof conditions were encountered in other face areas. Wooden timbers and steel beams were also used for support along haulageways and return air courses, but roof along the greater part of the haulageways and air courses was not supported artificially. Coal was

topcut and sheared by rubber-tired universal mining machines and loaded into rubber-tired shuttle cars with tractor-mounted loading machines.

Explosives and Blasting. Permissible-type explosives were used for blasting, and the blasting supplies were transported underground in a specially constructed explosives car and stored temporarily in suitable section boxes. Coal faces were topcut and sheared to a depth of 8 to 8-1/2 feet, then blasted on shift by authorized shot firers. The cuts were sheared off center, two shot holes were drilled on the wide side and one on the narrow side, and not more than 5-1/2 cartridges of explosive, less than 3 pounds, was supposed to be used in each hole. However, in one instance observed during the October 1954 Federal inspection, a shot hole was charged with more than 3 pounds of explosive. Incombustible material was used for stemming. Shot holes, except those used in blasting stumps, were blasted singly, beginning with the holes next to the shear. At the time of the October 1954 Federal inspection, shots were fired promptly after charging and suitable roof and gas tests were made before and after blasting.

Immediately after the disaster parts of 8 cases of the explosive used in the mine, Monobel AA, from 7 of the manufacturer's lots, were taken from the surface storage magazine at the mine and tested for permissibility requirements in the Bureau of Mines laboratories at Bruceton, Pennsylvania. The explosives from lots No. 11TI0623 and No. 24TI0629 failed to pass the gallery test. After the failures, the manufacturer was requested to remove from the market all the remaining explosives from these lots. One case of the same brand of explosive from each of 3 different lots, taken in March 1955 from the explosives car that had been left on the slope bottom, and a selected amount of the same explosive (lot number unknown) removed from the section storage magazine in 4 left off 2 north were tested in the Bureau laboratories and passed all tests for permissibility. Explosives were not found in the immediate face areas of 4 left entries off 2 north; therefore, it will never be known whether the explosives used for blasting in 4 left section on November 13, 1954, were part of any lot tested.

Insofar as could be ascertained each of the smaller pillar stumps (triangular base measurements about 3.5 x 6 x 7 feet) was generally blasted with one shot hole 6 to 7 feet in depth, drilled through the stump and head-coal roof into the overlying shale to a depth of 8 to 10 inches, and charged with 5 cartridges of explosive. At times, however, when the weakened condition of the stump so warranted, the one shot hole was drilled to a depth of 2 to 3 feet within the stump, or to a depth of less than 5 feet through and over or under the stump, and blasted with 2 or 3 cartridges. The larger corner stumps were usually blasted with 2 shot holes within and/or over the stump, each hole 6 feet or more in depth and charged with not more than 5-1/2 explosive cartridges. Reportedly, such stumps had been blasted occasionally with a single shot hole as much as 9 feet in depth and charged with as much as 12 explosive cartridges.

Although company safety rules required that only one shot be fired at a time, and multiple-blasting units were not provided, it was learned during the investigation that: (1) Shots in pillar stumps were usually fired two at a time, nonpermissibly, with a single-shot blasting unit; (2) 3 or 4 shot-firing cables, each attached to 1 or 2 of the 6 to 8 charges in a series of stumps, were employed in such blasting; (3) frequently tests for gas were not or could not be made, or could not be made safely close to the point of blasting, immediately before shots other than those fired first were blasted; (4) at times charges in stumps were not fired and were lost because roof falls resulting from firing previous shots "rode out" these stumps; and (5) it was not an altogether uncommon practice to connect all the shots in a series of stumps and fire them simultaneously by means of electric power obtained from the 275-volt mine electric circuit, either through an opening in face electrical equipment, nails driven into the trailing cable on such equipment, or the power line in the working section. The last-mentioned information was partly confirmed on September 9, 1955, when a three-conductor trailing cable on a shuttle car, with two 8-penny nails driven through the positive and negative conductors of the cable, was uncovered in cleaning up the large roof fall in No. 8 entry of 4 left off 2 north (See figure 4). The untwisted ends of the blasting cable were within 2 feet of the nails in the trailing cable, and a spool of cotton-insulated copper wire for connecting shots had been found nearby. Examination of trailing cables on the face electrical equipment indicated that shots had been fired similarly in 2 left off 1 south section, as nail holes and a part of a nail were found in the loading-machine cable.

Ventilation and Mine Gases. Ventilation was induced by two electrically driven axial-flow fans, operated exhausting and circulating through the mine approximately 347,800 cubic feet of air a minute. The fans were operated continuously. Each fan was installed in a fireproof housing on the surface, connected by a fireproof air duct to and offset from its air shaft, and provided with explosion doors, a recording pressure gage, and a device to give alarm should the fan slow or stop. Overcasts, stoppings, and regulators were constructed of substantial incombustible material. Doors were used only on the supply track near the slope bottom and for the man-trip station at the foot of the man shaft, and these doors were installed in pairs to form air locks. Check curtains and line brattice were installed to conduct air in the face regions, but the check curtains were rolled up and tied in place to facilitate shuttle-car haulage, and after the explosion the inby ends of unaffected line brattice in 5 left off C face and in 4 right off 3 north were 15 to 50 feet from the faces. Reportedly, line brattice was not used in pillar lifts except when gas was encountered. A fan of 24-inch diameter, the only fan used underground, was installed in fresh air near the foot of the slope and was employed to force about 4,000 cubic feet of air a minute through the slope to the surface. Company officials stated this was done to help prevent roof spalling in the slope, to minimize icing in the slope during the winter, and to insure against fumes or smoke entering the mine in case of a conveyor belt fire.

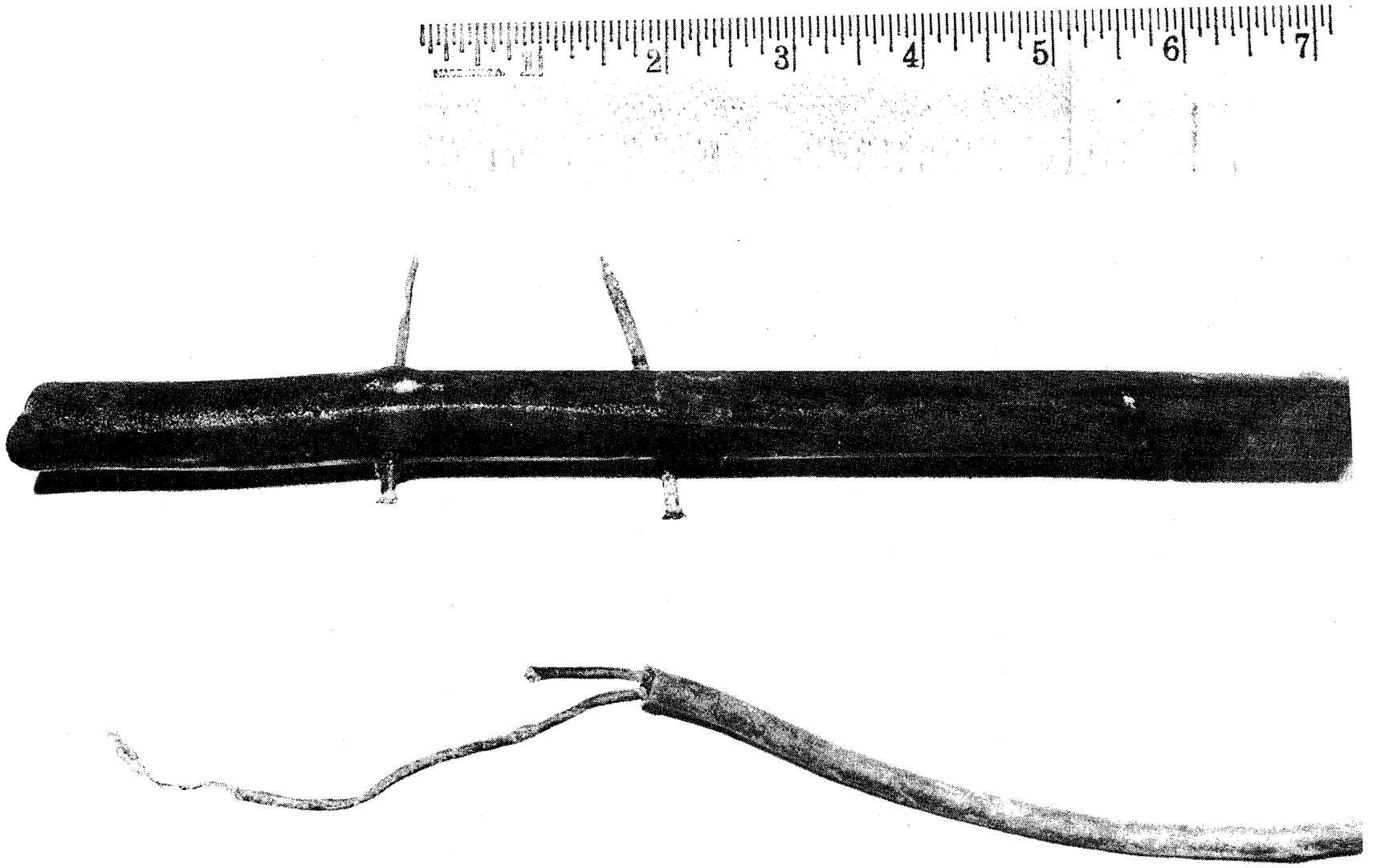

Figure 4. - Battery end of blasting cable and part of shuttle-car trailing cable, showing nails through cable, used for firing shots in No. 7 pillar place.

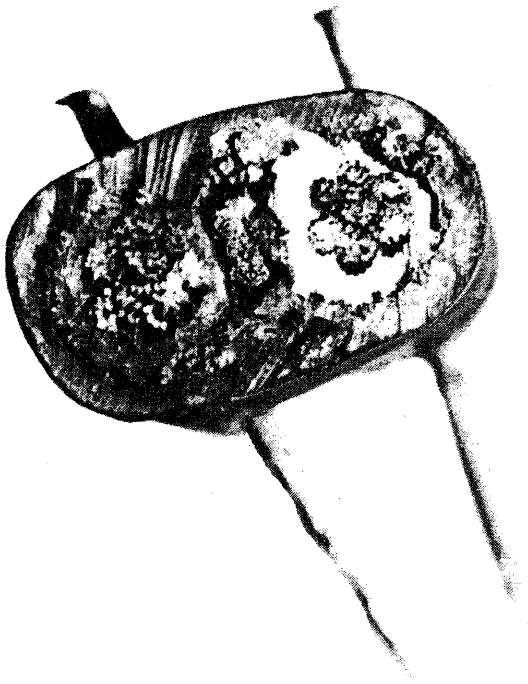

Figure 5. - End view of trailing cable with nails through positive and negative conductors.

The main west intake and return airways (separated by a 400-foot-wide pillar) and all the workings off these airways were ventilated with air that entered the mine through the man shaft and returned to the No. 2 fan. Part of the air from this intake, the 9 right split and the main north split off 10 right, returned to the No. 1 fan. All the air that entered the mine through the No. 1 air shaft, except for the slope split and a small split that was coursed through the pillared area off and west of C face mains, returned to the No. 1 fan. Reportedly, that part of C face main entries from 5 left to 9 right entries was ventilated at times by intake air that entered the mine through either of the 2 intake air shafts.

Each working section was ventilated by a separate split of intake air. The quantities of air reaching the last open entry crosscuts and the intake ends of pillar lines during the October 1954 Federal inspection ranged from 8,000 to 20,000 cubic feet a minute. In the working sections, intake air was coursed through the 4 center entries of each set, split right and left near the faces, and returned by way of the 3 or 4 outside entries and/or adjacent pillared areas.

The following air measurements and methane determinations for the 2 north split were obtained from the company's 1954 records:

	November 13		November 6		October 25	
Location	Air volume c.f.m.	Methane, percent	Air volume c.f.m.	Methane, percent	Air volume c.f.m.	Methane, percent
2 north intake, outby substation	58,132		60,984		60,006	
2 north intake, outby 4 left	47,232		47,040		41,164	
4 left, 2 north return, right side	17,108	0.10	14,100	0.10	16,160	0.10
4 left, 2 north return, left side	19,224	0.30	16,900	0.10	18,118	0.05
2 north return, left side outby 1 left	Not measured		9,940	1.70	9,886	1.65
2 north return, right side, No. 6 heading	do	do	28,756	0.55	29,006	0.45
2 north return, right side, No. 7 heading	do	do	27,840	0.55	25,112	0.45

According to measurements made by the general assistant mine foreman within an hour before the explosion, the quantity of air entering 2 north entries just outby 4 left entries was 47,232 cubic feet a minute. The volumes entering 4 left entries and passing through the 2 north regulator inby 4 left were not measured; however, air volumes measured in the 4 left return entries totalled 36,332 cubic feet a minute. Assuming that the quantity of air passing through the 2 north regulator was 3,000 to 5,000 cubic feet a minute, only a relatively small volume of air could have passed through the 4 left gob area toward the worked-out 3 left section, and, since a bleeder opening was not provided and the entries between the worked-out sections and the 2 north returns were not closed, it is certain that an air current did not pass through the vast gob area on the west side of 2 north entries and that, at best, only the east edge of this area was ventilated. Comparable air measurements were recorded by the officials for each of two previous examinations, October 25 and November 6, and methane in the amount of 1.70 and 1.65 percent was measured in a partial return (9,940 and 9,886 c.f.m.) outby the gob area on these occasions. Furthermore, line brattice was used to ventilate the outby 150 feet of the so-called bleeder entry south of 1 left off 2 north entries, and methane in at least near explosive proportions was found at the end of the brattice before the explosion. Check curtains, stoppings, and line brattice had to be used to keep the east edge of the 2 north gob area clear of gas during and after recovery operations. Therefore, it is believed that the gob area was filled with methane gas within or above the explosive limits.

Air samples were not collected in the split return airways during the October 1954 Federal inspection; however, tests made with an approved electric methane tester on the return side of the last active working place on each split showed methane contents ranging from 0.0 to 0.6 percent.

Air measurements made at the main fans during the four Federal inspections prior to the explosion and during the July 1955 special inspection and the corresponding rate of methane liberation are as follows:

Date	Fan	Cu. ft. air per min.	Cu. ft. methane per 24 hrs.	Total rate of methane liberation
August 1953	No. 1	120,960	566,646	
	No. 2	223,905	2,358,601	2,925,247
January 1954	No. 1	126,200	551,780	
	No. 2	235,300	2,443,348	2,995,128
May 1954	No. 1	113,100	430,099	
	No. 2	234,600	2,939,788	3,369,887
October 1954	No. 1	109,400	415,613	
	No. 2	238,400	2,832,955	3,248,568
*July 1955	No. 1	118,000	303,552	
	No. 2	190,000	2,018,304	2,321,856

*No coal produced at time of special inspection.

The mine is classed gassy by the West Virginia Department of Mines and by the Bureau of Mines. Fire bosses, who traveled by electrically driven jeeps from one working section to another, made preshift examinations for gas and other hazards only on Sunday evenings and at similar times prior to the resumption of operations following a shutdown of more than 4 hours; other preshift examinations for succeeding shifts were made on shift by the section foremen during their regular tour of duty. Onshift and weekly examinations for gas and other hazards were made by fire bosses, section foremen, assistant foremen, and the general mine foreman; however, the finding of gas that was removed during the same work shift, which occurred quite frequently in the 2 north section, was seldom included in the daily mine record book. Also, at the time of the last Federal inspection the records indicated that the return airways of some sections were not examined for intervals of as much as 2 weeks. Many falls were in the return airways before the explosion, and interrogation of employees revealed that perhaps only 1 and certainly not more than 2 of the multiple returns were traveled during such examinations, consequently many of these falls could not have been examined for gas at such times. Operators of electrical face equipment were instructed to make suitable tests for gas, but testimony submitted indicated that tests for gas were not always made immediately before electrical equipment was taken to the working faces. Tests made with a permissible flame safety lamp on pillar falls in active pillar sections during the October 1954 inspection did not indicate any accumulation of methane. Methane detected in one place in 3 north entries, a development section, by the section foreman when making a routine examination during this Federal inspection, was removed promptly by repairing the line brattice. No one was working in the place at the time, and it was the last working place on the air split. Numerous active and inactive gas and oil wells penetrated the coal bed in active and worked-out areas of the mine (and in virgin territory adjacent thereto); however. the mine map indicated that suitable blocks of coal were left around the wells. Tests made with W-8 methane detectors and air measurements made in split returns and at bottom of upcast air shafts by mine officials on November 13, 1954, and completed shortly before the explosion occurred, indicated normal air quantities and percentages of methane at these locations.

Coal and Rock Dust. The following information was obtained from reports on the three Federal inspections completed in 1954 and from company officials and employees. The greater part of the mine surfaces was dry. Water sprays were mounted on mining machines to allay dust; water was used when necessary to allay dust during loading operations, at shuttle-car discharging stations, and at the rotary dump near the slope bottom; and shuttle-car roadways were wetted down occasionally. Dangerous accumulations of loose coal and coal dust were reported at two locations, and these accumulations were removed from the mine promptly. Rock dust had been applied to within 40 feet of all but one of the working faces, and this working place was rock-dusted immediately. Small high-pressure rock-dust distributors transported in shuttle cars were used to rock dust the active sections, and larger rock-dusting

machines were used on off shifts for generalized rock-dusting. Rock dust was scattered manually along shuttle-car roadways on shift, and the shuttle-car discharging stations were cleaned up and the areas rerock-dusted as the mine-car loading points were moved. As the working sections were advanced, the floor in the return airways (back entries) was blanket rock-dusted; i.e., covered with a layer of rock dust one to several inches in depth. In addition, parts of the back entries and parallel entries developed several years ago had been rock-dusted recently by hand, and heavy blanket rock-dusting was done at some locations in such areas.

During the period October 1952 to October 1954, 6 Federal inspections of the mine were made, and 72 dust samples were collected at 48 locations in the face regions and along the haulageways. The incombustible content of the samples ranged from 70 percent to 99.9 percent. Federal and State reports on inspections before the explosion indicated the mine was generally well rock-dusted, and a rock-dusting survey completed August 25, 1954, by a representative of the West Virginia Department of Mines indicated adequate rock-dusting at most locations, as only 13 of 115 samples collected contained less than 65 percent incombustible. State and Federal inspectors did not collect dust samples in back entries or in open, trackless parallel entries except near the active working faces.

Company records show that 2,955 tons of rock dust was applied and 970,415 tons of coal was produced in 1954; this amounts to 6.1 pounds of rock dust used per ton of coal produced. During the 4 preceding years, the rate of rock-dust application was at a similar level, which is above the West Virginia average and probably above the National average. The amount of rock dust applied, the high incombustible content of the dust samples collected, the reasonably dust-free condition and good state of rock-dusting on haulageways, as well as the blanket rock-dusting in advancing back entries and in parts of older back and parallel entries, led company officials and employees and many others to believe that the entire mine was well rock-dusted.

If this rock dust was applied fairly uniformly by normal rock-dusting by machines, it would be distributed in the 7-foot by 15-foot entries (from which about 4.3 tons of coal is produced per linear foot of advance) at an average rate of 26 pounds per linear foot. Since about 30 percent of the rock dust applied in this manner adheres to the ribs and roof and 70 percent (18 pounds per foot) falls on the floor, this would provide on the floor of the entries a layer of rock dust with an average thickness of about 1/5-inch. Such rock-dust application on the ribs, roof, and floor would provide good protection against the propagation of an explosion in the area mined in 1954, if such area were reasonably free of fine coal and coal dust.

Despite the large amounts of rock dust used, rock-dusting practices in effect, samples collected, and statements of company officials, employees, and others regarding conditions prior to the explosion, information obtained during the investigation revealed

certain deficiencies. A very high proportion of the rock dust was used in blanket rock-dusting on the floor of back entries. In some sections of back entries the thickness of the rock-dust blanket was as much as 4 inches. According to testimony by a company official, it was normal practice to use 100 to 115 eighty-pound bags of rock dust on the floor of a 95-foot length of back entry; this is equivalent to 90 pounds of rock dust per linear foot, giving a layer with an average thickness of slightly more than 1 inch.

According to the mine map, development entries and sections were advanced at least 11,000 linear feet in 1954; this development included about 44,000 feet of intake entries, 39,700 feet of return (back) entries, and 41,000 feet of crosscuts. About 55 percent of the total coal production was obtained from these development entries, and the balance from pillar retreat. Blanket rock-dusting of the 39,700 feet of back entries (from which 170,000 tons or about 18 percent of the total coal production was won) at the indicated rate required 1,800 tons of rock dust or about 60 percent of the total used during the year. This means that only 1,155 tons of rock dust was available for rerock-dusting old entries, for application in newly developed haulage entries, parallels, crosscuts, and pillar sections--from which 800,000 tons of coal (82 percent of total production) was won, as well as for rock-dusting the rib-roof surfaces of the back entries.

Thus during 1954, when a high rate of rock-dusting was performed, the use of large quantities of rock dust for blanketing apparently resulted in deficiencies of rock dust and inadequate protection of some workings. Furthermore, even heavily blanket-rock-dusted entries are not protected properly unless the coal dust on the ribs and roof is also neutralized by adequate application of rock dust thereon.

Judging by the analyses of dust samples collected after the explosions from four of the working sections that were only slightly, if at all, affected by forces of the explosions, it is evident that the incombustible content was high (in the floor dust) in blanket-rock-dusted back entries (all 19 samples contained more than 67 percent incombustible); it was fairly satisfactory in haulage entries (9 samples out of 18 contained more than 65 percent incombustible); but it was entirely too low in parallel intake entries (18 samples out of 19 contained less than 60 percent incombustible).

After the explosions, examination of sections of the mine affected by strong forces, as well as sections affected only slightly by forces, revealed that loose coal and coal dust 3 to 8 inches in depth had been left along the greater part of the ribs, and there was more loose coal and coal dust in the pillar sections than in advancing sections. Larger amounts of loose coal and coal dust (ranging from 1 to 10 inches in depth) were on the floor along the greater part of the older back entries and entries parallel to the haulageways and connected thereto by open crosscuts. In many places rock dust covered the loose coal and coal dust. The loose coal and coal dust on the floor of the parallel and back entries were generally of the same type and texture and were lying

on the floor at a generally uniform thickness across the entry. There was little evidence of sloughing of new coal from the roof and ribs. Visual observation indicated that sufficient rock dust had not been applied on the roof and ribs at several places in advancing entries, notwithstanding the large amount of rock dust on the floor at such locations. Furthermore, careful examination showed little evidence of rock dust having been applied at many places in the parallel and older back entries, even though other parts of these entries were blanket-rock-dusted. These observations and the analytical results of dust samples collected after the explosions show clearly that the mine dust in the greater part of the parallel and older back entries did not contain sufficient incombustible to prevent propagation of an explosion.

Transportation. Cable-reel-type shuttle cars were used for face haulage; they discharged the coal directly into 20-ton-capacity steel mine cars, which were hauled by trolley locomotives to the rotary dump at the slope bottom. A well-installed belt conveyor transported the coal from the slope bottom to the coal-storage bins on the surface. A track was also installed in the slope for hauling supplies. Tracks and rolling stock were in reasonably good condition. The clearance space along haulageways was well maintained, and shelter holes were available at frequent intervals except along the slope. Ordinarily, men entered and left the mine by way of the automatic elevator installed in the man shaft and were transported underground in specially constructed covered man cars. In some instances when only 5 or 6 men were transported, they were hauled in a mine jeep. The man-trips were well conducted.

Electricity. Electric power, 110, 220, 440, 2,200, 4,160, and 22,000 volts alternating current, was used on the surface; and 275 volts direct current and 110, 220, 440, 2,200, and 4,160 volts alternating current were used underground. The substation equipment underground was in well-ventilated fireproof rooms, each on a separate air split. The electric face equipment, trolley locomotives, and some pumps were operated from the 275-volt direct-current system. Pumps were operated only in intake airways. Permissible and nonpermissible electric face equipment was used. Two permissible-type cutting machines, two loading machines, and a hand-held electric drill were not in permissible condition at the time of the October 1954 Federal inspection, due to three openings in excess of 0.004-inch into vital electric compartments, four defective conduits, a missing bolt, and a loose boom-light lens. This equipment was placed in permissible condition promptly. Trolley, feeder, and power wires were installed on insulators and sectionalized with cutout switches and circuit breakers. Temporary splices in trailing cables were made with splicing rings and insulated with friction tape. Short-circuit protection was provided. At the time of the October 1954 Federal inspection, operators of electrically driven equipment made suitable tests for gas before taking the equipment into the face regions and at frequent intervals thereafter.

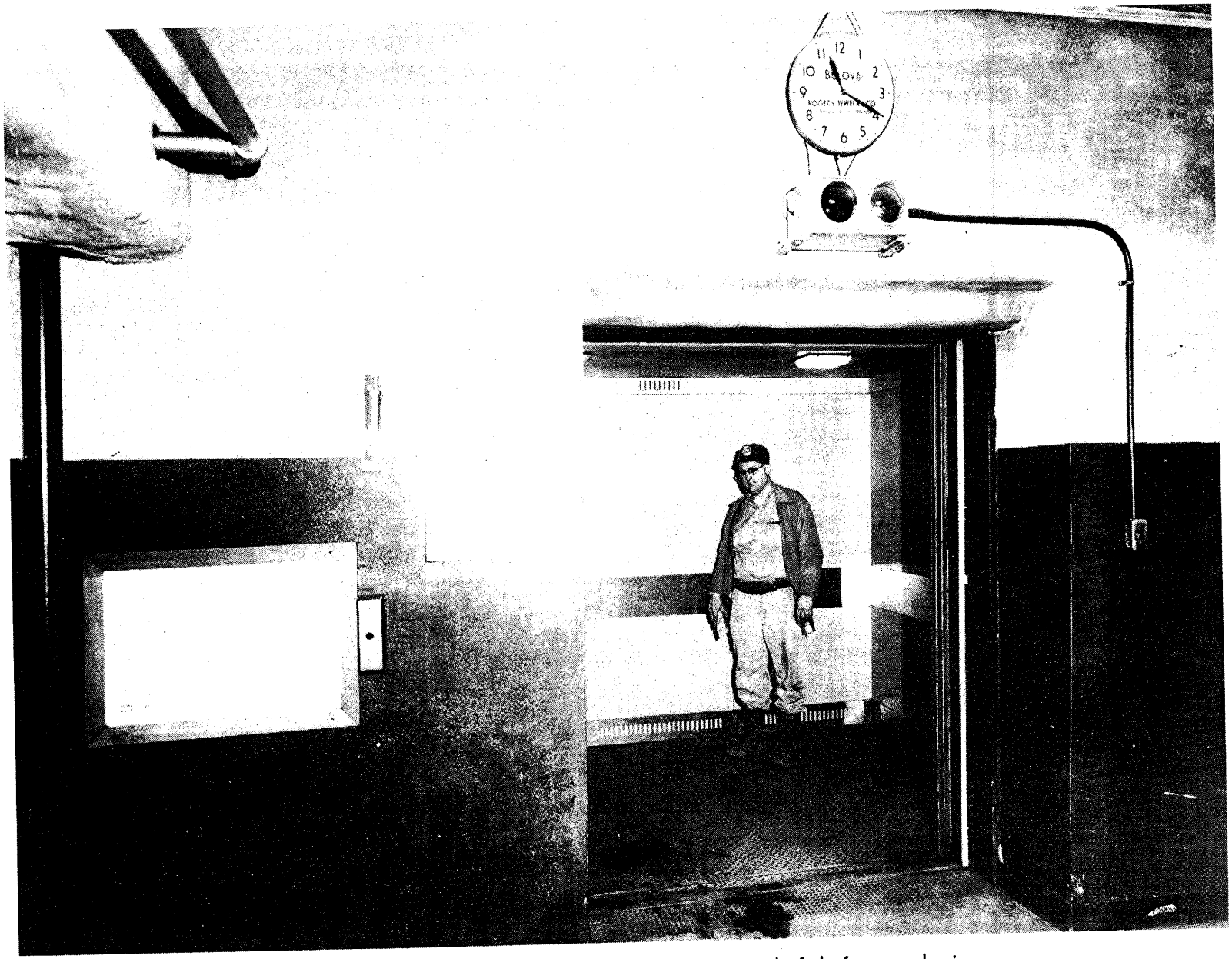

Figure 6. - Waiting room and elevator at top of man shaft before explosion.

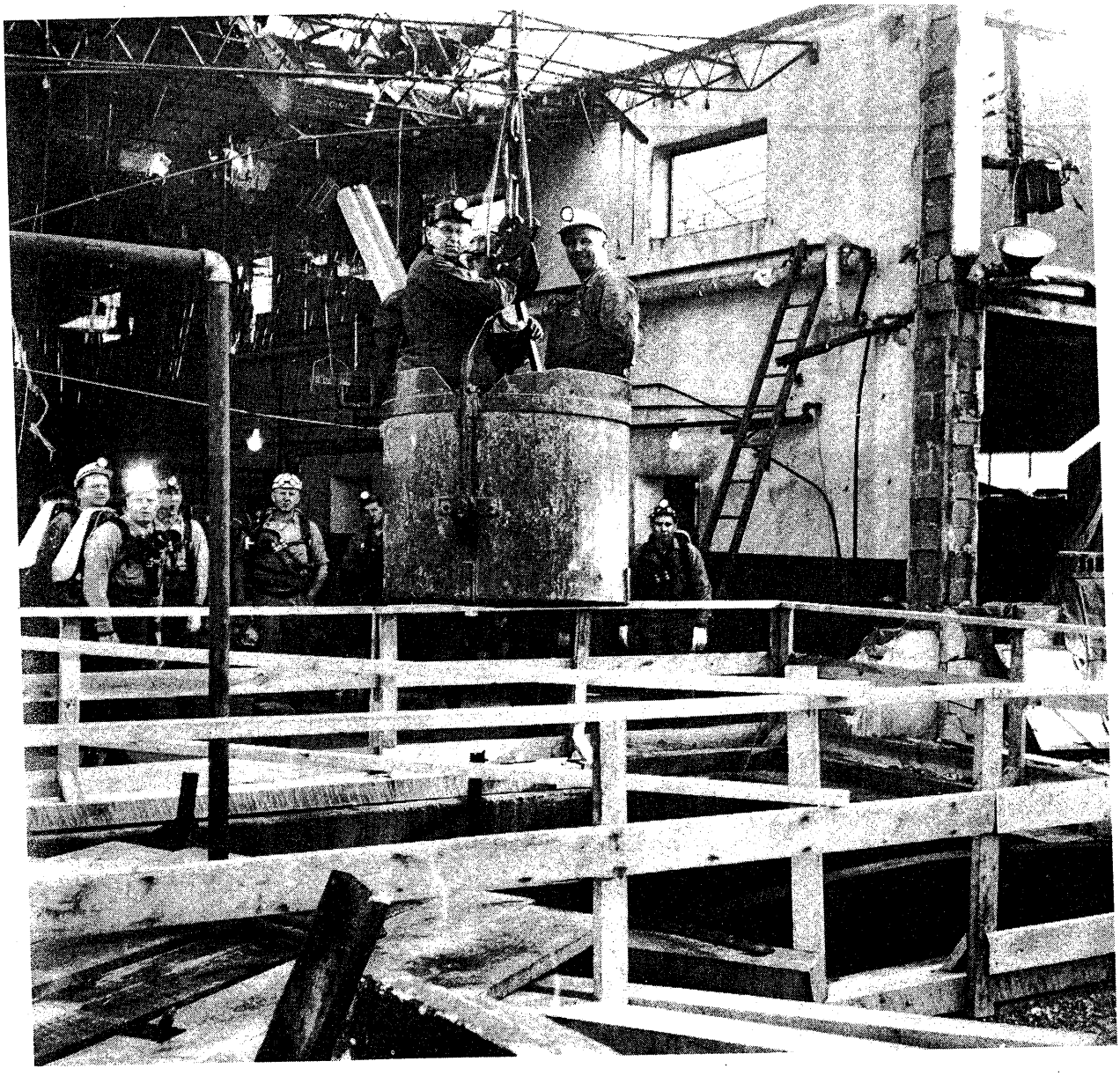

Figure 7. - Top of man shaft and bucket used for handling men during recovery operations.

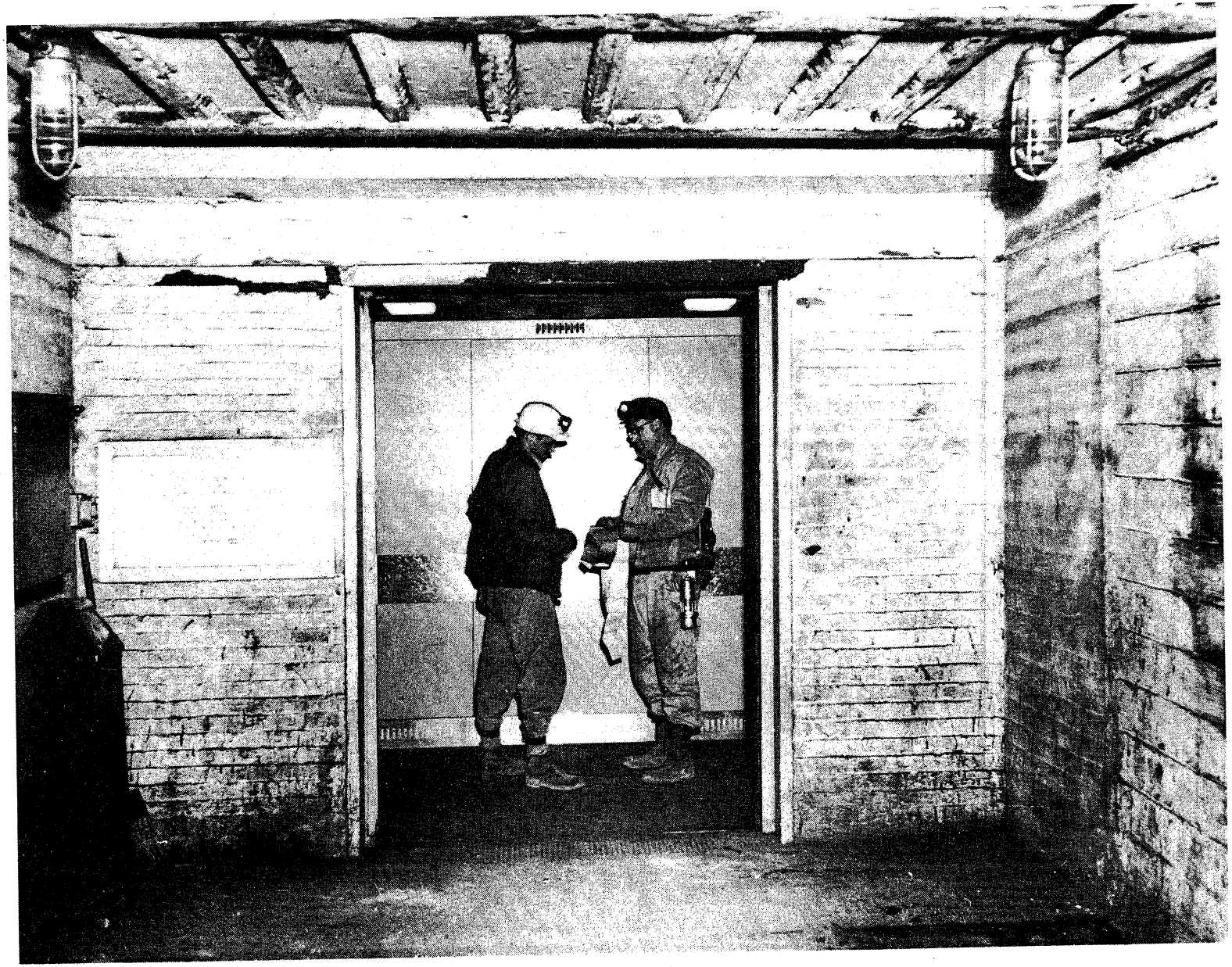

Figure 8. - Elevator and man-trip station at bottom of man shaft before explosion.