A minerals research contract report

March 1982

SURVEY OF ELECTROSTATIC HAZARDS DETECTION AND MINIMIZATION

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16. Abstract (Limit 200 words)

The objective of this study is to make a critical evaluation of the magnitude and extent of electrostatic hazards in typical U.S. mining operations and to assess the adequacy of techniques, materials, and equipment for detecting and minimizing these hazards. This was accomplished with an incident-accident survey, mine visits, literature search, and engineering analysis. It was concluded that it would be desirable to (1) utilize ground fault interrupters to minimize stray current from electrical utilities, (2) decrease the current and electrostatic sensitivity of electric detonators used in areas with a high incidence of lightning strikes or troublesome stray currents, (3) institute quality control and test standards for blasting material and protective devices, (4) establish an electrostatic monitoring program for loading ammonium nitrate and fuel oil in plastic-lined holes, and (5) establish a risk criteria for operator exposure in mines.

17. Document Analysis   a. Descriptors

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  Electrostatics   Detonators
  Hazards   Safe Handling

b. Identifiers/Open-Ended Terms

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Stray currents
Explosive handling

c. COSATI Field/Group   08I

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FOREWORD

This report was prepared by Hercules Incorporated, Allegany
Ballistics Laboratory, Cumberland, MD, under USBM Contract Number
J0308113. The contract was initiated under the Mineral Health and
Safety Technology Program. It was administered under the technical
direction of the USBM Pittsburgh Research Center with Mr. Karl Becker
acting as Technical Project Officer. Mr. Patrick J. Neary was the
contract administrator for the Bureau of Mines. This report is a
summary of the work recently completed as a part of this contract
during the period September 1980 to March 1982. This report was
submitted by the authors on March 15, 1982.

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INTRODUCTION

BACKGROUND

The Bureau of Mines was established in the public interest with part of their charter to conduct inquiries and scientific and technologic investigations concerning mining and the preparation, treatment, and utilization of mineral substances, as well as to promote health and safety in the mineral industries. Federal Safety Standards for both metal and nonmetal, open pit and underground mines as well as for sand, gravel, and crushed stone operations and surface coal mines reflect the results of this work. One requirement that is now law is that when electric detonators, delay connecters, blasting caps, etc., are used, charging shall be stopped immediately when the presence of static electricity or stray current is detected. Charging operations will not be resumed while these conditions persist.

The electrostatic sources may be, but are not restricted to, electrical storms, loaders, plastic loading hose, plastic liners, and electrostatically charged persons. A variety of procedures, devices, and accessories is currently in use to direct and/or minimize the hazard.

A review of the sensitivity, reliability, effectiveness, and utilization of these devices and procedures in typical mining operations was required.

OBJECTIVE

The contractor was authorized to make a critical evaluation of the magnitude and extent of electrostatic hazards as well as their role in past accidents/incidents in blasting operations and to assess the adequacy of techniques, materials and equipment for detecting and minimizing these hazards.

SCOPE OF WORK

The objective was accomplished in four tasks as follows:

1. Review of Accident Reports

Accident reports for the five years (1976 to 1980) were reviewed for incidents or accidents that can be directly attributed to, or are suspected of having been caused by, electrostatic or electrical current phenomenon. These were reviewed for frequency of similar accidents or incidents and for various common factors.

2. On-Site Visits

A total of 16 on-site visits were made to review mining methods and the measures presently employed for detecting and assessing electrostatic hazards. These visits included a cross-section of the U. S. Mining Industry.

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3.   Engineering Analysis

This task was undertaken to review the hazards of mining practices based on the sensitivity of blasting materials to inadvertent activation by electrostatic discharge or stray currents. The sensitivity data were obtained from a survey of technical data published by the Bureau of Mines, the IME, Explosive Manufacturers and other sources.

4.   Evaluation of Techniques to Minimize Hazards

A variety of materials, techniques, or instruments have been developed to minimize the hazards associated with static electricity in mines. This task included an evaluation of these approaches to minimizing the hazards identified in the engineering analysis and provides recommendations for additional development or regulation to improve mine safety where problems still exist.

I. ACCIDENT/INCIDENT SURVEY

A survey was performed of accidents and incidents which occurred during the years 1976-1980 involving static electricity and stray currents in coal and metal/nonmetal mining operations in the United States, including both surface and underground mines. The contractor sought reports on accidents or incidents involving explosives, blasting agents, and detonators with one of the following possible causative factors: (1) lightning/electrical storms, (2) static electricity (associated with dust storms, snow storms, pneumatic loading of blasting agents, electrostatic discharges from human operators, etc.), (3) induced currents from radio frequency transmitters or nearby power lines, or (4) stray electrical currents (fault currents from electrically-powered machinery or equipment, galvanic currents, ground currents through conductive rock strata, exposed battery terminals, etc.). A "stray current" was loosely defined as any electrical current entering a blasting circuit or conducting through blasting-related materials (such as safety fuse) other than that which was intended for electrical initiation of detonators at the proper time.

A. Obtaining Reports

Accident reports were obtained from several locations within the structure of the U. S. Department of Labor, Mine Safety and Health Administration (MSHA).

The survey was initiated by obtaining a computer printout of all accidents having "explosives" or "explosions" as a key word, with a brief narrative description for each. This listing was obtained from the MSHA Health and Safety Analysis Center (HSAC) in Denver, Colorado, through the Bureau of Mines liaison office in Denver. A total of 893 accidents or incidents was listed in this compilation which included the years 1978-1980. (Listings with narrative descriptions were not computerized prior to 1978.) Of the 572 coal mine listings, many were minor methane ignitions of no consequence.

From a review of the HSAC list, 27 metal/nonmetal mine and 18 coal mine listings were identified which might possibly have had static electricity or stray currents as a causative factor. Due to the brevity of the narrative descriptions and the fact that the listing did not cover 1976 and 1977, it was evident that the survey could not be based solely upon a review of available information at HSAC.

During one of the on-site mine visits, the MSHA Metal/Nonmetal Subdistrict Office in Salt Lake City, Utah was visited. It was clear from this visit that the most productive place to obtain the accident/incident reports would be at the MSHA district/subdistrict offices. Since it was not feasible to visit each office, a letter of request was sent to each

MSHA metal/nonmetal and coal subdistrict office (or district office, for those districts not divided into subdistricts) asking the office to review their files for the years 1976-1980 and send copies of reports for accidents having stray currents or static electricity as a possible causative factor. This request produced 69 reports, 22 of which were applicable to the present analysis.

The compilation of reports was completed by requesting and receiving from HSAC, via the Bureau of Mines liaison office in Denver, copies of microfiche documentation for accidents identified from the original HSAC data and not obtained from the district/subdistrict offices. Eight documents were obtained from this request, seven of which were applicable to the present study.

The above procedure resulted in records for 29 accidents or incidents in the U. S. coal and metal/nonmetal mining industries for the five-year period from 1976 to 1980 which have static electricity or stray currents as a possible causative factor. This is a rather small number when compared to the size of the U. S. mining industry but it nevertheless is a significant number. These 29 accidents resulted in six fatalities and 31 injuries.

B.   Review of the Data

The accident survey pointed out cases that were attributable to (1) lightning, (2) stray current, or (3) unknown causes. No cases were identified as being caused by static electricity; however, some of those listed as "unknown" could have resulted from electrostatic causes.

Lightning was a causative factor in four underground and five surface mine cases.

In the area of stray current, 12 cases were reviewed, eight were underground and four were in surface mines. Of these, only three were strictly related to stray current. These include (1) a shot that occurred when the only suspect source was a 7200 VAC single phase overhead transmission line 150 ft from the site, (2) a shot that occurred in a situation where poor power cable connections existed with highly conductive rock conditions, and (3) a condition where uninsulated splices were close to but not touching electrical circuits.

Several other cases were listed in the reported summary as being attributed to "stray current." These involved direct contact with live circuits due to poor working conditions/habits and one due to a faulty blasting machine. These do not require a specific analysis and can be corrected by operating procedure changes.

Typical cases where static electricity is suspected as a causative factor are (1) a situation where an electric detonator exploded in a rock picker's hand, and (2) a case where a bore hole shot while a blaster's helper was pushing powder into the hole.

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The accident survey was not as productive as anticipated. The general type of information available is summarized in this report in Table I-1. The nine cases directly associated with lightning, three cases of potential stray current, and the lack of a clear case where static electricity was identified as a causative factor did not present sufficient data for statistical analysis of controlling factors.

The review did point out the following areas of concern:

1. Warning of Incipient Lightning

Sufficient warning is neither available nor is it being utilized to remove the personnel from the blast site when lightning hazards are present. In the nine cases reviewed where lightning was a causative factor, only three times were the personnel evacuated before the premature detonation occurred.

2. Likelihood of Premature Detonation by Lightning

In the cases reviewed where lightning was involved, the premature detonation did not involve activation of the complete system. This indicates that the current introduced into the firing circuit is marginal for electric detonators and that less sensitive caps may improve the safety of the operation. The following is representative of the charges detonated in a round when it is subjected to lightning:

- o   One of 14 charges detonated
- o   Partial round detonated
- o   14 of 20 charges detonated
- o   Three of 12 charges detonated
- o   73 of 114 charges detonated

Laboratory studies should be undertaken to determine why the undetonated material was not susceptable to lightning.

3. Accident Reports

Accident reporting could be made more effective if the format were standardized and required to contain additional technical data such as:

- Detailed description of the blasting arrangement and the blasting materials
- Soil conductivity
- Representative conductivity of blasting agents, hose, etc.

The reports surveyed are summarized in Table I-1.

TABLE I-1

SUMMARY OF ACCIDENT SURVEY

Accidents Attributable to Lightning, Stray Current, or
Static Electricity in the United States During 1976-1980

Mine Information	Primary Cause	Information Pertinent to the Accident				
		Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District – Northeastern Area City – Johnson County – Lamoille State – Vermont Date – 24 Sept. 1979 Mine Type – Underground talc Mine Method – Sublevel stope	Human error	A scraper man carried ~50 each 36-inch long, fused, nonelectric, No. 6 blasting caps into the changehouse section of the mine. He did not use a covered nonconductive container and held the detonators in his hand. He stood adjacent to a miner's cap lamp charger and apparently contacted the terminal, resulting in an explosion.	One man was treated for metal fragment in his right hand and minor burns.	Fused nonelectric No. 6 blasting caps	Three shelf Koehler, Model 5400, 1800 watt/225 amp 4 volt miner cap lamp charger.	o Operator did not have caps in proper container. o Operator carried blasting caps into wrong area.
District – Northeastern Area City – Adams County – Berkshire State – Massachusetts Date – 2 Feb. 1977 Mine Type – Surface limestone Mine Method – Open pit multiple bench	Stray current from faulty blasting machine	An operator had positioned himself behind a pickup truck to hook up the blasting leads to a condenser discharge machine. He tightened one lead to the negative terminal with the second lead hanging free. The shot then went off.	None	Load Atlas bagged Pellite® blasting agent Priming Atlas Power primer with a single MS delay cap Arrangement Single row of 14- 3 inch diameter, 35 foot deep vertical blast holes.	VME condenser discharge blasting machine	A check of the blasting machine after the incident indicated a direct short between the negative terminal and the machine case.

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TABLE I-1 (CONT'D.)

Information Pertinent to the Accident

Mine Information	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District - Northeastern Area City - Austinville County - Wytne State - Virginia Date - 9 July 1980 Mine Type - Underground zinc Mine - Open Method stopes and random pillars	Lightning with the current flow probably entering the mine via the wire braded air hose and water lines that were located throughout the mine.	A heading round consisting of 44, 1 1/8 inch diameter holes, 8 ft deep were charged in the lowermost section of the B decline. The caps were wired in a series circuit with a leg wire remaining from a top hole on each side of the round. These had not been shunted to protect from stray current. The shot fired prematurely approximately one hour after loading was completed and 1 1/2 to 2 minutes after the operators left the loaded round.	None	Load Hercules Gelimite® No. 2 Priming Hercules Superdets®	o Air and water lines installed to within 30 ft of the face. o A de-energized power cable was installed with the air and water pipes.	o Electric blasting caps were left unshunted. o Other mine operators indicated that lightning strikes were evident. This produced situations such that as (1) a jack leg drill operator received a shock off the drill hose, and (2) the jumbo drill operator said the entire floor area lit up with a glowing appearance three times. o No attempt was made to warn the miners underground of the electrical storm.

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TABLE I-1 (CONT'D.)

Information Pertinent to the Accident

Mine Information	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District – Northeast-ern Area – Austinville City – Wythe County – Virginia State – Date – 21 June 1978 Mine Type – Underground lead/zinc Mine – Open Method – stopes and random pillars	Lightning	A 14-hole round was loaded and primed in the mine while the weather condi-tion was clear. A single series circuit was uti-lized with the end of the firing line shunted and located 6 feet from the blasting machine. The weather condition changed during the time when the operators checked at 12:30 p.m. and the pre-mature detonation occur-red at 2:08 p.m.	None	Load ANFO Priming Electric detonator with 8 ft copper leg wire Arrangement Single series circuit	Electric slusher in-stalled near the ore pass.	o Area was monitored for stray current on the day of the inves-tigation and no stray currents were found. o Hoist man reported that a severe elec-trical storm ap-proached from the east and that light-ning struck at the time of the blast. o An electrician re-ported unexplained arcing which coin-cided with the light-ning and the blast.

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TABLE I-1 (CONT'D.)

Information Pertinent to the Accident

Mine Information	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District - Rocky Mountain Area - ; City - Thornton County - Not available State - Illinois Date - 17 August 1979 Mine Type - Surface limestone quarry Mine - Multiple pits	Lightning	A premature detonation of a connected blast round occurred as an electric storm passed through the area. At the time of the premature detonation, all of the holes were loaded, stemmed, and the wires from about 5 or 6 of the holes were attached to the blasting machine. All personnel had been accounted for as the storm approached.	None	Load ANFO Priming Double primed with cast primers and electric blasting caps. Arrangement A typical round was a single row containing 20 holes. (Ten 6 1/2 inch diameter primary holes and ten 5 1/2 inch diameter secondary holes.) The primary holes contained six separate deck charges.	Trojan sequential blasting machine	o Only some shots of a given series detonated while others remained intact indicating a marginal firing current. o The shot was cleaned up prior to the investigation. o This was reportedly the first premature detonation by lightning in 40 or 50 yrs of operation by this company.

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TABLE I-1 (CONT'D.)

Information Pertinent to the Accident

Mine Information	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District - Rocky Mountain Area - Lead City - County - Lawrence State - South Dakota Date - 23 June 1980 Mine Type - Underground gold Mine Method - (1) Cut and fill square set (2) Vertical crater retreat (VCR)	Stray current or static electricity	Three VCR stope blasting loaders were preparing a ledge for blasting. The loading was complete and two No. "0" delay detonators were placed parallel to the trunk line and secured with black electrical tape. They tied the detonators to the blasting line. During this operation, they found a break in one line and repaired it. They then decided to hook up one lead to the blasting machine. This was done and while an operator was scraping the other lead with a knife, the shot went off.	One operator was injured by flying rock from the blast.	Load DuPont Tovex® water gel Priming o Ensign-Bickford strip mine special primer cord. o Atlas 1-pound Deck master® o DuPont instant caps Arrangement Six-inch diameter down holes were drilled to the drift below. The holes were spaced on an 8 x 9 ft pattern with extra holes as needed.	o DuPont CD-24 blasting machine o An Eimco 12-B mucking machine and eight rocker cars were close to the blasting machine	o Operators hooked up the one lead to the blasting machine prematurely. o The lead wires that the miners had repaired earlier had not been insulated by taping. (a) The bare spot in one wire was separated by a distance of 1 ft from power cables and pipes. (b) The lead wires were hung at least three ft above a 440 volt switch box. o After the accident, the blasting machine was found to have some voltage leakage. It could not be demonstrated that the voltage would fire two No. 6 detonators like those involved in the incident.

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TABLE I-1 (CONT'D.)

Information Pertinent to the Accident

Mine Information	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District – Rocky Mountain Area City – Silverton County – San Juan State – Colorado Date – 22 December 1980 Mine Type – Underground gold Mine Method – (1) Square set cut and fill (2) Vertical crater retreat (VCR)	Lightning or stray current	A premature detonation occurred during a loading operation when a lightning storm was in progress in the area and the electrical power conductors were in a condition that could cause stray currents. All of the holes were primed with electric detonators and one stick of 65% dynamite. Eleven holes were loaded with ANFO and as the operator started to load the twelfth hole, a detonation occurred.	Victim lost his left hand and forearm and damaged both eyes.	Load ANFO Priming o Hercules 1 1/8 in. Unigel® dynamite o Electric detonators Arrangement 30 1 3/8 in. diameter holes by 6 ft deep	o Condenser discharge blasting machine o Jack leg drill o Jarvis Clark 220X loader o Pneumatic loading gun	o The electrical ground and single 480 volt – three phase conductors had been damaged by fly rock and were resting on a 2 in. metal compressed air line. o The area was wet and condensate had formed around the electrical conductor. o The conductors were undersized. o The grounding cable was ineffective. o The compressed air line was connected to the prill loading device. o The 2 in. bull hose and static resistant hose on the loading device were in ~3 to 4 in. of water. o The leads from the electrical detonators were not rolled up and they dangled in the water.

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TABLE I-1 (CONT'D.)

		Information Pertinent to the Accident				
Mine Information	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District - Rocky Mountain Area City - Not available County - Not available State - Colorado Date - 14 March 1978 Mine Type - Underground Uranium Mine Method - Conventional stoping	Undetermined	A man walked into the primer room to talk to a man who was making up cap and fuse primers. They stopped to talk when a quantity of caps in a tray exploded.	One man suffered multiple cuts and burns on his face and arms.	o Cap and fuse	Undefined	Undefined
District - Rocky Mountain Area City - Not available County - Not available State - Colorado Date - 9 March 1979 Mine Type - Above ground sandstone Mine Method - Open pit	Undetermined	An employee was destroying electric dynamite caps when one of the caps went off.	A man was injured by fragments hitting both knees.	o Dynamite caps	Undefined	Undefined

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TABLE I-1 (CONT'D.)

Mine Information	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District – Rocky Mountain Area – City – La Sal County – San Juan State – Utah Date – 23 November 1976 Mine Type – Underground Uranium Mine – Method – Random pillar	Electrical current due to failure to transport capped fuses in non-conductive container.	A man was carrying 50 capped fuses (primers) over his shoulder and passing a haulage truck. One or more of the un-contained detonators contacted the bare wire connections on the vehicle alternator.	Two men were injured by shrapnel	Capped fuse	Undefined	o The voltage regulator on the haulage buggy had a partially bare wire and bare wire connections. o Employee was carrying capped fuses uncovered and uncontained.
District – Rocky Mountain Area – City – La Sal County – San Juan State – Colorado Date – 10 October 1980 Mine Type – Underground Uranium Mine – Method – Open stope random pillar	Possible stray current from shorted headlight on haulage truck.	A man picked up two bundles of primers (30 capped fuses per bundle) and placed them beside the driver's seat in the operator's compartment of a haulage truck. He traveled 10 to 15 ft when all of the caps detonated.	Operator was injured by fragments	Capped fuses	10 ton Elmac mine haulage truck	o Employee was carrying capped fuses uncovered and uncontained. o Twelve-volt wiring system on truck could have been shorted.

Information Pertinent to the Accident

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TABLE I-1 (CONT'D.)

Mine Information	Primary Cause	Accident Summary	Injury	Information Pertinent to the Accident		
				Explosive Materials	Equipment/Utilities	Related Factors
District – Rocky Mountain Area – City – Callao County – Tooele State – Utah Date – 29 July 1980 Mine Type – Underground gold Mine – Method – Drift mining	Lightning	The two victims were in the process of wiring the lead wires of a 20 hole face round. The mine was a new adit 8 x 10 ft cross-section and 40 ft deep. A third operator outside the portal saw lightning strike in the area. He walked into the portal and told the men to get out. They responded but the premature detonation occurred before they could leave the area.	Two men were killed	Load Hand mixed ammonium nitrate and diesel fuel Priming Atlas Power primer® dynamite and duPont Mark 5 blasting caps	o 110 volt AC safety switch and a 7000 watt, 110/720 volt AC gasoline powered electric generator was used to fire the explosives.	o Blasting line was No. 2/14 AWG wire.
District – North Central Area – City – St. Augustine County – Warren State – Illinois Date – 23 September 1976 Mine Type – Surface limestone Mine – Open pit Method – high wall	Stray current	A blast hole round consisting of 32 holes, 62 feet deep and 3 1/2 inches in diameter was being prepared in adverse weather conditions and with an undesirable condition and location of power cable going to the pit water pump. An air track drill was being moved into position to clean the holes when the premature blast occurred.	One operator received a fractured skull, severe cuts and bruises.	Load Hercules A-3 Gel Powder® Priming o DuPont 18 gm Deta prime® o Hercules Millidet® 0-5 millisecond delay caps.	o Detailed discussion included on Page 30 of this report.	

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TABLE I-1 (CONT'D.)

| | | Information Pertinent to the Accident | | | | |
Mine Information	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District - Southeastern Area City - Smithland County - Livingstone State - Kentucky Date - 3 April 1980 Mine Type - Surface limestone Mine - Quarry	Horseplay resulting in stray current	An employee was attempting to wire an electric blasting cap to his foreman's truck as a joke when the cap detonated.	Employee received severe cuts to his hand	Atlas electric cap	No additional details were available.	
District - Southwestern Area City - Ambrosia County - McKinley State - New Mexico Date - 13 August 1979 Mine Type - Underground (Uranium) Mine - Conventional stoping	Stray current or static electricity	A track foreman found a stick of 45% powder with a No. 9 delay detonator. He picked up the powder, took two or three steps and the powder detonated.	Foreman lost his left hand	Stick powder No. 9 cap	None	o Leg wires unshunted. o Six volt dry cell telephone battery nearby.

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TABLE I-1 (CONT'D.)

Mine Information	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
				Information Pertinent to the Accident		
District - Southwestern Area - City - Grant County - Valencia State - New Mexico Date - 27 May 1980 Mine Type - Underground (Uranium) Mine - Room and pillar	Unknown	An employee was standing by a single blasting cap when it detonated.	Minor injury to the employee's leg.	Electric blasting cap	None	No details in the report.
District - South Central Area - City - Annapolis County - Iron State - Missouri Date - 30 June 1977 Mine Type - Surface (granite) Mine - Quarry Method - Open pit	Lightning	A crew was charging a blasting round when an electrical storm was observed in the distance. They decided to evacuate the area and after a period of time, 73 of the 114 holes detonated. During the drilling cycle, a drill became stuck in a hole and was left protruding 6 feet vertically. This could have received a lightning strike and be a contributing factor.	No injuries	Load Aquaflow blasting agent Priming Powerdyn® dynamite and electric detonators Pattern 114 vertical 3-inch diameter holes averaging 52 feet deep on an 8 x 10 feet pattern.	None	None

TABLE I-1 (CONT'D.)

Mine Information	Information Pertinent to the Accident					
	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District - No. 1 Area City - Not available County - Not available State - Pennsylvania Date - 24 January 1980 Mine Type - (Coal) Mine - Method -	Unknown	The report indicates that a cap exploded when an employee had it in his hand. Poor details in report.	Employee's hand was injured.	Details are not available.	Undefined	
District - No. 1 Area City - Lincoln County - Schuylkill State - Pennsylvania Date - 1 September 1980 Mine Type - Underground (coal) Mine Method - Drift	It was concluded that the blasting caps were unknowingly attached to an energized power supply.	A fatal explosives accident occurred at the face of the advancing slope of a coal mine. The operator was reportedly starting to connect the leg wires of the blasting caps when the explosives prematurely detonated.	One man was killed.	Undefined	Undefined	o Employees were doing other work in the immediate area while the holes were being charged. o There was no permissible shot firing device available at the mine. o All charged holes detonated.

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TABLE I-1 (CONT'D.)

Mine Information	Primary Cause	Accident Summary	Injury	Information Pertinent to the Accident Explosive Materials	Equipment/Utilities	Related Factors
District – No. 2 Area City – Morris Run County – Tioga State – Pennsyl- vania Date – 1 August 1977 Mine Type – Surface (coal) Mine Method – Strip	Lightning	Following a 6 a.m. rain shower with lightning, charging a large number of holes was taking place during intermittent rain showers. At approximately 2:30 p.m., an open circuit was found in the firing system. While they were attempting to test portions of a circuit, a five-minute rain shower occurred, followed by lightning, which detonated all but three of the charged holes.	No additional details available.			
District – No. 2 Area City – Ebensburg County – Cambria State – Pennsyl- vania Date – 19 July 1978 Mine Type – Underground (coal) Mine Method – Stope	Improper disposal method	A worker and his foreman found a blasting cap that had not exploded while inspecting a rock shot. The worker was instructed to touch the cap leads to the foreman's cap lamp battery. The cap detonated, causing injury.	The injured person received multiple abrasions and puncture wounds.	No additional details are available.		

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TABLE I-1 (CONT'D.)

	Information Pertinent to the Accident				
Mine Information	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District – No. 4 Area City – Keystone County – McDowell State – West Virginia Date – 3 March 1977 Mine Type – Underground (coal) Mine Method – Stope	Stray current	The accident occurred when six employees were preparing blast holes in the mine floor. Eighty-eight shots were prepared and 58 shots were wired when 24 of the shots were prematurely detonated. Contact of cap leg wires with a 300 volt DC power line was suspected as the cause of the premature detonation.	Two men were seriously injured.	DuPont Tovex®	Electric trolley	o The operating conditions were such that the trolley cable could have been active at the time of the accident.
District – No. 6 Area City – Van Lear County – Johnson State – Kentucky Date – 10 August 1977 Mine Type – Surface (coal) Mine Method – Strip	Lightning	On the day of the accident, preparatory blasting operators were halted due to rain and electrical storm conditions. After a period of time, explosive charging was resumed. Lightning then struck ~800 feet from the pit resulting in premature detonation.	Two men were killed and one man seriously injured.	Undefined	Undefined	o Explosives were not handled and holes were not charged in a safe manner.

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TABLE I-1 (CONT'D.)

Mine Information	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District – No. 6 Area – City – County – State – Kentucky Date – 6 November 1976 Mine Type – Underground (coal) Mine Method – Stope	Unknown	A shot helper was pushing powder in a hole when it exploded.	The face and chest of the helper received lacerations and burns.	Undefined	No additional data are available	
District – No. 7 Area – City – Graysville County – Jefferson State – Alabama Date – 7 May 1976 Mine Type – Surface (coal) Mine Method – Strip	Lightning	Personnel had evacuated the area due to the approach of a thunderstorm. Lightning apparently caused a premature explosion of the charged holes on a nigh wall.	Undefined injuries.	Electric blasting caps.	Undefined	Undefined

Information Pertinent to the Accident

TABLE I-1 (CONT'D.)

Mine Information	Primary Cause	Accident Summary	Information Pertinent to the Accident			
			Injury	Explosive Materials	Equipment/Utilities	Related Factors
District – No. 7 Area City – Mentone County – Cherokee State – Alabama Date – 12 July 1978 Mine Type – Surface (coal) Mine Method – Strip	Improper manner of checking continuity.	A misfire was experienced in the mining operation. After the misfire, the blasting cable was disconnected from the cap leg wires. The following method was utilized to check the circuit continuity in the charged holes. Two flashlight batteries were connected to one detonator leg wire. The other leg wire was wrapped around a flashlight bulb. The bulb was then touched to the batteries. The second hole shot when this technique was used.	Undefined injuries.	Undefined	Undefined	o The galvanometer normally used was inoperative. o A certified blaster was not supplied.

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TABLE I-1 (CONT'D.)

Information Pertinent to the Accident

Mine Information	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District - No. 7 Area - Boothton City - Shelby County - State - Alabama Date - 22 June 1978 Mine Type - Surface (coal) Mine Method - Strip	Stray current	The vertical auger on an ANFO truck was malfunctioning such that the truck had to straddle the holes to accomplish loading. It appears that a source emanating from the truck initiated the charge in one of the holes.	Four men were injured.	Explosive o E-2 crushed AN prill o Sierra Chemical Company's cast primer containing an Atlas "Rockmaster" delay EB cap o HX-30 ANFO	Vertical auger bulk explosives truck	o The truck was being driven over charged holes and leg wires from other charged holes.
District - No. 8 Area - New Concord City - New Concord County - Muskingum State - Ohio Date - 16 May 1978 Mine Type - Surface (coal) Mine Method - Strip	Assumed to be static or stray current.	Three vertical holes had been drilled in the over burden of the mine. Two of the three holes were charged in preparation for blasting. The entire crew was away from the first hole when it detonated.	No injuries.	Load Nitro-carbo-nitrate Priming Hercules dynamite and 12 millisecond electric detonators with 40 foot leg wires. Pattern Holes were 25 feet deep, 6 inches in diameter, and on 20 foot centers.	Chicago Pneumatic diesel-powered drill mounted on the bed of a 190 International truck.	o A 7,200 volt AC single phase distribution power circuit was installed on 40 ft high poles ~150 ft from the blast area. o A microwave transmission tower operating at 48.66 KHz was about 1 mile south of the area. It was reported to be idle at the time of the blast. It could not be determined if C.B. or shortwave radios were being operated.

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TABLE I-1 (CONT'D.)

Information Pertinent to the Accident

Mine Information	Primary Cause	Accident Summary	Injury	Explosive Materials	Equipment/Utilities	Related Factors
District – No. 10 Area City – Unknown County – Unknown State – Missouri Date – Mine Type – Above ground Mine Method – Strip	Unknown	The only information available is that 18 holes had been wired and shunted when hole No. 17 and cap No. 16 "discharged." ("Discharged" was the word used in the report.)	No additional information is available.			

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C. Specific Cases

Several of the cases are typical examples of specific hazards that can occur in a mining operation and are presented in detail as an illustration of the problem.

1. Accident Due to Stray Current from Electrical Equipment[1]

An accident occurred on September 23, 1976, at a limestone quarry. The most likely cause of the accident was stray current from a water pump. The situation at the mine is illustrated in Figure I-1.

At 9:00 a.m. on the day of the accident a cold front approached from the southwest with 14 to 17 mph winds. The sky was overcast and light rain showers fell; however, no atmospheric electrical activity was evident.

A 6-inch centrifugal water pump, "Flyt" brand, powered by a 75-hp, 440-volt 3-phase motor, was being used to pump water from the pit sump. The pump was located approximately 275 feet to the southeast of the face being loaded. The electrical circuit to the pump was comprised of a 100-ampere, 500 VAC, Westinghouse safety switch, located in the main plant distribution center which fed one-half mile of overhead powerlines to a 400-ampere, 600 VAC, Square D switch. The power was then fed through a 170-ampere, 440-volt, pump control box. Neither the control box nor the switch was connected to a low resistance ground; i.e., power company ground line.

A General Electric Geoprene 2/0 A.W.G. four conductor, Type W, power cable supplied power from the pump control box to the pump. The power cable was 200 feet long and contained two poorly constructed, insulated splices. There was also evidence of dry rot within the cable. Approximately 50 feet of the cable was constantly showered with water from a leak in the pump discharge line. The last 15 feet of the power cable leading to the pump was underwater.

The only overload protection afforded the pump was three 100-ampere, super lag renewable fuses located in the Westinghouse safety switch. The fuses had blown frequently. On the morning of the accident, one fuse had to be renewed and the pump control box was reset.

The limestone ledge where the accident occurred had a water course running through it, with the rock strata being separated by several narrow horizontal seams of chert and flint. Tests revealed that the ground resistance between the quarry face and the water pump area (~275 ft) was 400 ohms.

The blasthole round consisted of 32 holes, 62 feet deep and 3-1/2 inches in diameter. The shot was drilled in a multiple row pattern with holes and rows fired in sequence from one end. Stemming, (fine rock materials) 3 to 4 feet in depth, was used in each hole. Each hole was double
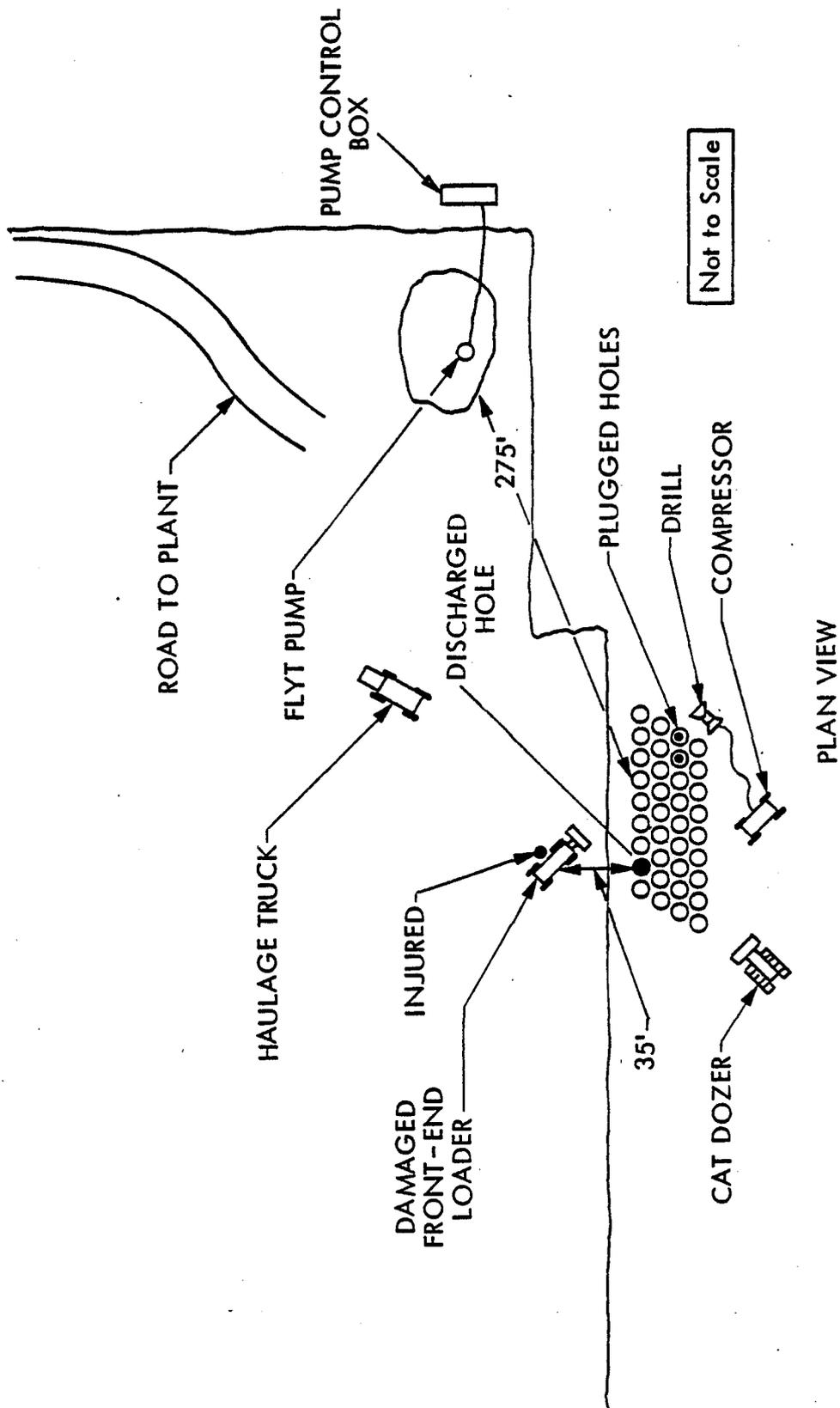
Figure I-1. Scene of Nonfatal (Premature) Blasting Accident

ROAD TO PLANT

PUMP CONTROL BOX

FLYT PUMP

DISCHARGED HOLE

275'

PLUGGED HOLES

DRILL

COMPRESSOR

Not to Scale

PLAN VIEW

HAULAGE TRUCK

INJURED

DAMAGED FRONT-END LOADER

35'

CAT DOZER

primed, using 18 gm duPont Detaprime® primers with each cap; the caps were the Hercules Millidet® delay series (periods 0 through 5) with 40- and 80-foot legwires. Each hole was loaded with 31 sticks of Hercules A-3 Gel Power. The caps were shunted with an aluminum foil shunt with a polyurethane exterior insulation. At the time of the investigation, three caps were found not shunted and their wires were in contact with the ground.

Thirty of the holes were loaded and preparations were underway to clean two holes that had obstructions in them. The obstructions had occurred 50 feet below the collar of each hole where a water course intersected the holes.

Description of Accident

A maintenance man, arrived at the operation at approximately 5:45 a.m. on the day of the accident. His normal duties included starting the pit water pump and checking and performing general maintenance on the plant. At 6:15 a.m., the Westinghouse safety switch which serviced the pump began to smoke. He turned off the power to the switch and continued his maintenance duties. At 7:10 a.m., he began to work on the switch. He found that one of the 100-amp renewable fuses in the switchbox had blown and that two of the incoming phase wires had arced and shorted across. At 8:10 a.m., the wiring and fuse were repaired and he turned the power back on. He then drove over to the pump control box, reset the breakers, and started the pump. The pump appeared to operate normally so he returned to the plant.

The normal shift employees arrived at the operation at 7:00 a.m. - their regular starting time. The assistant superintendent, made his assignments for the day. Two drillers were assigned to accompany and assist the assistant superintendent in loading 32 previously drilled holes. A haul truck operator and another individual were to haul rock from the pit to the plant. Another operator was assigned to operate a front-end loader in the pit. The loading operations were conducted directly in front of the round being charged. Reportedly, on a past occasion, the loader operator requested that he not be required to operate the loader under these conditions. He had also contacted the business agent of the Operating Engineers Local, and repeated this request. Company officials felt there was no hazard and this practice continued.

At about 8:30 a.m., a sales distributor arrived on the property to assist in the charging of the round. Work was not progressing very well due to adverse weather conditions. Most of the holes had 6 to 8 feet of water in them, plus material from the water course. About 10:30 a.m., 30 of the 32 holes had been charged. The two remaining holes had to be cleaned due to the fact that they were completely plugged. While loading was in progress, a storm front passed through and the loading personnel sat in their vehicles on several occasions until the wind and rain subsided.

A mechanic, was instructed by a supervisor to get the dozer, hook up the jumper cables and start the compressor motor. This work was performed and he backed the dozer away from the loading area and shut off the

engine. He then helped the others connect the air hose to the drill. The air track drill was being moved in position to clean the holes when the premature blast occurred.

At the time of the blast, the loading operator was performing his assigned duties - loading haulage trucks in the pit. He was about 35 feet away, facing the pit wall, when one of the front holes detonated. Flyrock from the blast seriously injured him and did extensive damage to the front-end loader. He was found in a semi-conscious condition, attempting to climb out of the loader, with apparent severe head injuries.

The shunted leg wires from the two caps that detonated were found in the rockpile created by the blast.

Cause of Accident

The cause of the accident was premature detonation of a charged blasthole by stray or static current induced into the blasting circuit. Contributing causes included, (1) charging of holes under adverse weather conditions, (2) operating the pit water pump in the immediate area of the blast site, and (3) the undesirable condition and location of the power cable going to the pit water pump.

2. **Accident Due to Electrostatic Discharge or Stray Current**[2]

An accident occurred on June 23, 1980, at an underground gold mine. The cause was not clearly defined but could have resulted from electrostatic discharge from the operator.

The VCR (Vertical Crater Retreat) Stope mining method was being utilized. Six-inch diameter down holes were drilled to the drift below from the top drift. Holes were spaced on an 8 foot by 9 foot pattern with extra holes drilled as needed. After the entire stope was drilled, all equipment was removed from the stope and the area was made ready for blasting.

Each down hole was measured to determine the exact depth of the hole. Holes were loaded from the top and the explosives were initiated at the bottom. Single or double decking of explosives was used, depending upon the thickness needed to be blasted. A single deck of explosives was loaded in the following manner: two wedges were cut from a 4 by 4 by 12 inch pine block with each wedge having a 1/2-inch hole drilled into the tapered end. Ensign-Bickford strip mine special Primacord was fed through the hole in the wedge and tied. The wedge was lowered into the hole to within 5 feet of the bottom of the hole. The other wedge was dropped into the hole, and the first wedge pulled tight blocking the hole. A foot of loose muck was pushed down the hole, blocking the bottom. Two 30-pound cartridges of DuPont Tovex WS-7(F) Water Gel were cut and dropped into the hole. An Atlas 1-pound Deckmaster primer was placed on the primer cord and slid down the hole. A foot of loose muck or sand and 10 to 12 feet of water was used as stemming. A double deck charge, another charge loaded and stemmed over the first one.

In the stope area, where the blasting accident occurred, 23 single deck charges and 5 double deck charges were loaded, making a total of 28 loaded holes, with 28 1-pound Deckmaster primers and 1,980 pounds of water gel. The 28 down hole strip mine detonating cords were tied into 300 feet of Ensign-Bickford E cord trunk line. Two duPont instant caps were taped to the double looped truck line for connection to the lead lines leading to the blasting station.

Description of the Accident

At 0700 hours on the day of the accident, three VCR Stope blasting loaders, were assigned to the task of preparing the stope area, for blasting. Work progressed normally until approximately 2:00 p.m., at which time the loading was completed, the stope area was cleaned and the two zero delay detonators were placed parallel to the trunk line and secured with black electrical tape. Two operators proceeded to tie the detonators to the blasting line, while the other operator went to the 4400 level to "danger off" the scram area.

While testing the lead lines, they found a break in one line, repaired it and rechecked the circuit. They then decided to hook up one lead to the blasting machine, so that when the other stopes in the area were blasted, they would not have to spend so much time in the smoke hooking up leads. (The VCR Stope was to be blasted after all persons were cleared from the area.) They attached one lead wire to the machine, and as an operator was scraping the other lead wire with a knife, the shot went off. Both men thought that someone else had blasted, as they had not hooked the other lead wire to the machine. They were not aware that their round went off until they saw the smoke coming out of their blasting area.

An investigation of the VCR Stope area revealed that the distance from the location of the blasting machine to where the two electric zero detonators were connected to the leads was approximately 220 feet. The presence of a mucking machine and a string of eight rocker cars made the task of hooking the lead wires to the blasting machine somewhat difficult because of the close quarters. The clearance between the rib and the mucker was measured and found to be thirty inches. The lead wires, that the miners had repaired earlier, had not been insulated by taping; however, the bare spot in the one wire was separated from the surrounding rock, power cables and/or pipe by a distance of one foot. A 440 volt switch box was located at the intersection, but the lead wires were hung at least three feet above the box and were properly installed on wooden insulators all the way to the blast area.

Current leakage and resistance checks were made of the area by a company electrician and an MSHA inspector. Results showed no voltage leakage, and the resistance was within acceptable limits.

The blasting machine had been checked in the company electrical shop on June 1, 1980, and sent to the mine on June 16, 1980. It was returned to the electical shop again on June 24, 1980 (the day after the accident), by an MSHA inspector and tested again in his presence. It was

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found at this time that there was some current leakage. It was thought at this time that this might have caused the premature blast by hooking up one lead wire and touching the case with the other; however, after setting up a test with the same blasting machine, using approximately the same length of lead wire connected to two zero detonators, the detonators could not be initiated in any way except by connecting the leads to the machine and shooting in the normal manner. The blasting machine was then sent to Denver Technical Support Center for further tests. A sample of the detonating cord was also sent to Denver Technical Support Center for tests.

An investigation was also made to see if it were conceivable that another blast in the vicinity of the VCR Stope, or that nearby drilling might have caused the premature blast; this proved not to be the case.

In an effort to explore every conceivable cause, the lead miner voluntarily submitted to a polygraph test. Results of the test showed that the two miners in the VCR Stope did not hook both leads to the blasting machine, did not knowingly shoot the round, and that their statement that, when the blast occured, they thought some other crew had shot another round, was true.

Cause of the Accident

Although the direct cause of the accident could not be determined, the investigation group reached the following unanimous conclusion:

Had the two miners not removed the safety shunt from the lead wires 30 minutes prior to blasting time and inserted one lead wire into the blasting machine and started scraping the other lead wire in the close proximity of the blasting machine, this blasting accident would not have occurred.

3. Incident Due to Lightning[3]

An incident occurred due to lightning in an underground lead and zinc mine on June 21, 1978. At the time of the incident, operations were proceeding normally and stray currents were not measured after the incident. Lightning activity has caused problems in other areas of the mine; therefore, this incident was attributed to lightning.

The mining method used here was open stopes with random-pillars. Diesel-powered mobile equipment was used in some areas while electric and pneumatic-powered equipment was utilized in the remaining areas of the mine. Lead-zinc ore was transported from stope chutes to the shaft pockets with railmounted diesel-powered trains and was crushed underground before it was hoisted to the surface for processing in the mill.

The area of the stope involved was located above the tenth level, and access was provided through a heading off the tenth level. An electric slusher was installed near the ore pass from the eleventh level which intersected the stope at the end of the access heading.

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The 14-hole round was loaded with ANFO and primed with electric detonators which had 8-foot copper leg wires. A single series circuit arrangement was used. The circuit and main firing line were checked with a blasting galvanometer and found to be satisfactory. The other end of the firing line was shunted and located 6 feet from the blasting machine.

At about 1:55 p.m., after preparing the round for firing, two operators proceeded to work in the lower section of the stope which was in a safe area. At about 2:08 p.m., they heard an explosion. They then proceeded to the shaft telephone and notified the mine office.

On the day of the investigation a volt-ohm meter was used to check for stray currents. All pipes, electric conductors, slusher, slusher cable and the ground in the area were checked. No stray currents were observed.

The hoist-man reported that a severe electrical storm approached from the east and that lightning struck in the vicinity of the mine at the time that the blast reportedly occurred. An electrician also reported an unexplained arcing of electricity had occurred on the seventh level at a time which coincided with the lightning and the blast.

Cause of the Premature Ignition

Since stray currents were not observed in the stope area on the day of the investigation and since the lightning strike and the arcing on the seventh level coincided with the blast, it is presumed that the cause of the ignition was electric current entering the blasting circuit from the lightning during the storm.

4. Incident Due to RF or Induced Currents[4]

One of the accidents reviewed occurred in an area where high voltage power lines and microwave antennas were present. Since the actual cause of the incident was not determined, this mode of initiation is of special interest.

A premature initiation of a charged hole occurred on May 16, 1978 at about 7:30 a.m. The mine superintendent notified the field office the following day at 1:00 p.m., and the investigation began on May 18, 1978.

Three vertical holes had been drilled in the gray shale overburden on 20-foot centers. The area had been benched prior to drilling, and the holes were 25 feet deep and 6 inches in diameter. The drilling was done with a Chicago Pneumatic diesel-powered drill mounted on the bed of a 190 International truck, and electrical power was not available in the immediate vicinity.

Two of the three boreholes were charged in preparation for blasting. Each charge consisted of about 75 pounds of nitro-carbo-nitrate,

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with Hercules dynamite primer and a 12 millisecond electric detonator with 40-foot leg wires. A certified foreman who was in charge of the 3-man crew had been assisting the shooter in charging the holes. The other two employees were in the process of tramming the drill equipment from the bench.

The entire crew was about 100 feet away from the first charged hole when it detonated. The battery for blasting had not been brought to the blasting site at the time of the detonation. None of the men were injured. Minor damage resulted was sustained by a car passing by on a nearby highway. Flyrock penetrated the roof of a home about 250 feet north of the blast area, but there were no injuries.

The cause of the accident was assumed to be either static or stray electric current. A 7,200 volt a.c., single phase distribution power circuit installed on 40-foot high poles was about 150 feet from the blast area. A microwave transmission tower was about 1 mile south of the area. It was operated at 48.66 k Hz, but was reported to have been idle at the time of the blast. The leg wire shunts of similar detonators were examined. They were made of aluminum and were a little thicker than a chewing gum wrapper. The leg wires may not have been properly shunted at the time of the blast. It could not be determined whether CB or short-wave radios were being operated in the vicinity at that time.

5.   Accident With a Bulk ANFO Truck[5]

At the time of the incident the truck was being operated improperly while located over filled shot holes. The accident is of interest since the truck could have been the source of an electrostatic discharge.

On Thursday, June 22, 1978, the day shift crew reported to work at the dragline pit, at approximately 7:00 a.m. The blast pattern being used was a seven row drilling pattern with 9 inch diameter boreholes varying in depth from approximtely 35 feet at the pit to approximately 60 to 65 feet at the highwall side. Most of the blast holes had been drilled on the lower bench of the overburden and a drill operator proceeded to complete the drilling operation. The blast holes on the upper benches had been drilled previously.

The procedure used to charge holes was as follows: First, two "wet bags" were dropped in the holes. These "wet bags" were E-2 crushed ANFO prills, manufactured by Monsanto Chemical Company, and packed in bags 6 in. x 40 in. long. These were followed by a Sierra Chemical Company's cast primer containing an Atlas Rockmaster® delay electric blasting cap with 80 foot leg wires. Enough "wet bags" were then added until the hole was "dried." If more than four "wet bags" were needed to dry a hole, the hole was primed again. After the holes were primed, the truck containing the bulk ANFO, (Monsanto's HX-30 ANFO having a density of 0.85 g/cc) was positioned to complete filling the holes. The truck (a D.M. 600 Mack) equipped with a Hydra-Hauler for bulk ANFO for transportation to the blasting site, had a capacity of approximately 12 tons and was equipped with four

separate bins to transport bulk explosive. Apparently 400 pounds of ANFO was in the forward bin at the time of the accident; the other three bins were empty.

Usually the bulk material was poured out of the vertical auger on the rear of the truck. This auger was able to swing to each side of the truck so that the truck would not be required to pass over the holes to fill them. On the day of the accident, the vertical auger was not in working condition so the ANFO had to be discharged from a door at the end of the horizontal auger underneath the truck. This meant that the truck would have to straddle charged holes in order for the ANFO to be poured into the next hole. The blasting crew finished charging two rows of holes on the upper bench and moved to the lower bench to begin charging the three rows of 45 feet deep blast holes on this level. In charging the holes on the lower bench, the truck was driven forward over the third row of holes. The crew stemmed all the holes they had previously charged as they progressed down the row. When the blasting crew finished charging the third row of holes, the truck was moved to begin on the middle row of holes. The middle row of holes was dry, so all that was needed was an electric blasting cap and primer before the bulk explosive was added. Rather than driving the truck forward over the middle row, the crew backed the truck over this row. Thus the truck was directly over a completely charged hole while filling the next one. The blasting crew finished charging the first hole in the middle row but did not stem it. They then backed the truck to begin filling the second hole. The explosion occurred during loading of the second hole. It occurred in the charge hole directly under the truck, blowing a 15-inch diameter hole through the bottom of the truck in the area of the third bin from the rear of the truck. In addition, the explosion blew out the back window of the cab, blew off all four bin doors and injured all four men.

D.   Incident/Accident Summary

The total compilation consisted of 29 accidents or incidents in the U.S. coal and metal/nonmetal mining industry for the five year period 1976-1980. Six of these were attributed to lightning - too small a data base for statistically significant conclusions.

The records were reviewed and the information was classified into seven categories. These are presented in Table I-1 and include the following information:

        o Mine Information
            o Area
            o Mine Type
            o Mining Methods
            o Accident Date
        o Information Pertaining to the Accident
            o Primary Cause
            o Accident Summary
            o Injuries
            o Explosives and Equipment
            o Other Factors

These data were then examined for frequency of occurrence for mine type, mining method, substance mined, district, cause, phase of operation, explosive type, cap and booster type, and weather conditions. Metal and nonmetal data were compiled separately from coal data. These results are summarized in Table I-2. The data are compiled only on a single factor basis. The data base was too small for a review on a multifactor basis (e.g., "electric detonator in uranium mines" or "lightning in high wall operations"). The correlation study did point out the following:

o Electric Blasting Caps

Electric blasting caps were found to be involved in most cases. However, two cases involved nonelectric cap and fuse indicating that the elimination of electric detonators would not eliminate the problem.

o Substance Mined

The review did not indicate a significantly higher frequency for conductive ore mines as opposed to nonconductive ones. Ten occurrences were in uranium, gold and zinc mines as opposed to seven in limestone, talc, granite, and sandstone mines.

o Cause of Accidents

Lightning was the major cause with six occurrences; five were attributed to human error. All these could have been avoided. Three were attributed to stray current and three others were cases where the details of the occurrence were too sketchy to assess causes.

TABLE I-2

EVALUATION OF MINE ACCIDENT
FREQUENCY AS A FUNCTION OF VARIOUS PARAMETERS

Metal and Nonmetal Operations

Mine Type	Mine Method	Substance Mined	District	Cause	Phase of Operation	Explosive Type	CPP and Booster Type	Weather Condition
Underground 11	Not Available 5	Uranium 5	Rocky Mt. 8	Lightning 6	Explosive Loading 5	None 7	Electric Detonator 8	Not Available 8
		Limestone 4	Northeast 4	Human Error 5	Transportation 4	ANFO 4	Electric Detonator and Booster 7	Lightning 6
Surface 6	Stope 5	Gold 3	Southwest 2	Stray Current 3	Firing System Hookup 2	Dynamite or Pellite 4	Nonelectric Cap & Fuze 2	Rain 1
Metal 10	Bench 2	Zinc/Lead 2	North Central 1	Unknown 3	After Hookup 2	Slurry 2		Undefined 2
	VCR 2				Uncontrolled Explosives 2			
Nonmetal 7	High Wall 1	Talc 1	Southeast 1		Primer Assembly 1			
	Drift 1	Granite 1	South Central 1		Disposal 1			
	Room and Pillar 1	Sandstone 1						

Coal Mine Operations

Mine Type	Mine Method	Substance Mined	District	Cause	Phase of Operation	Explosive Type	CPP and Booster Type	Weather Condition
Surface 8	Strip 7	Coal 12	No. 6 3	Unknown 4	Explosive 4	Undefined 5	Electric Detonator 8	Undefined 5
Underground 4	Stope 3		No. 1 2	Lightning 3	Loading 3	Permissible 2	Undefined 2	Not Available 4
	Drift 1		No. 2 2	Human Error 3	Firing System Hookup 4	None Present 2	Electric 1	Lightning 3
	Not Available 1		No. 7 2	Stray Current or ESD 2	Picking Rocks 1	ANFO 2	Electric Detonator and Dynamite 1	
			No. 4 1		After Shot 1	Gel 1		
			No. 8 1		Undefined 2			
			No. 10 1					

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II. VISITS TO MINE SITES

An important phase of the evaluation of the magnitude and extent of electrostatic hazards was to observe actual practices in the mines. A total of sixteen mines representing a cross section of the mining industry were visited. The schedule of visits is included as Table II-1. The areas of concern to the analyst were (1) utilities at the mines which may produce stray currents, (2) loading techniques that may generate electrostatic charges, (3) personnel who may accumulate static charges or who, through improper actions, may inadvertently permit electrical currents to flow through the bridgewire circuits, (4) control and transportation of explosive materials, (5) blasting techniques, and (6) environmental factors.

TABLE II-1. SCHEDULE OF MINE VISITS

Type Mine	Location
1. Salt - Underground	Louisiana
2. Zinc - Underground	Tennessee
3. Limestone - Mine/Quarry	Tennessee
4. Coal - Underground	Kentucky
5. Uranium - Underground	Utah
6. Copper - Open Pit	Arizona
7. Tungsten - Underground	Nevada
8. Silver/Copper	Montana
9. Silver/Copper	Idaho
10. Trona - Underground	Wyoming
11. Lead - Underground	Missouri
12. Limestone - Underground	Kansas
13. Limestone - Underground	Kansas
14. Gypsum - Underground	Kansas
15. Coal - Open Pit (2 properties)	Pennsylvania
16. Coal - Underground	West Virginia

A. Mine Utilities

Electrically powered equipment is used in both underground and surface mining operations. The mines observed were supplied with electrical service from the local utilities ranging from 3 Phase 4160 VAC to 60 cycles single phase 120 VAC service. In general, this service terminates at substations on the mine property. From this point the service is installed and maintained by the mining company who uses the National Electrical Code and CFR 30 for guidance. A variety of electrical service installations were observed during the mine visits. A series of photographs (Figures II-1 through II-8) is included to illustrate the working environment as it is related to electrical service. Figure II-1 shows a typical substation where the utility company's responsibility ends. Figure II-2 shows a typical installation of a master breaker/fuse panel for a mine. Figure II-3 illustrates the transfer of utilities to working areas via a tunnel. Figure II-4

Figure II-1. Surface Power Substation, Termination of Power Company Service, Beginning of Mine Electrical System
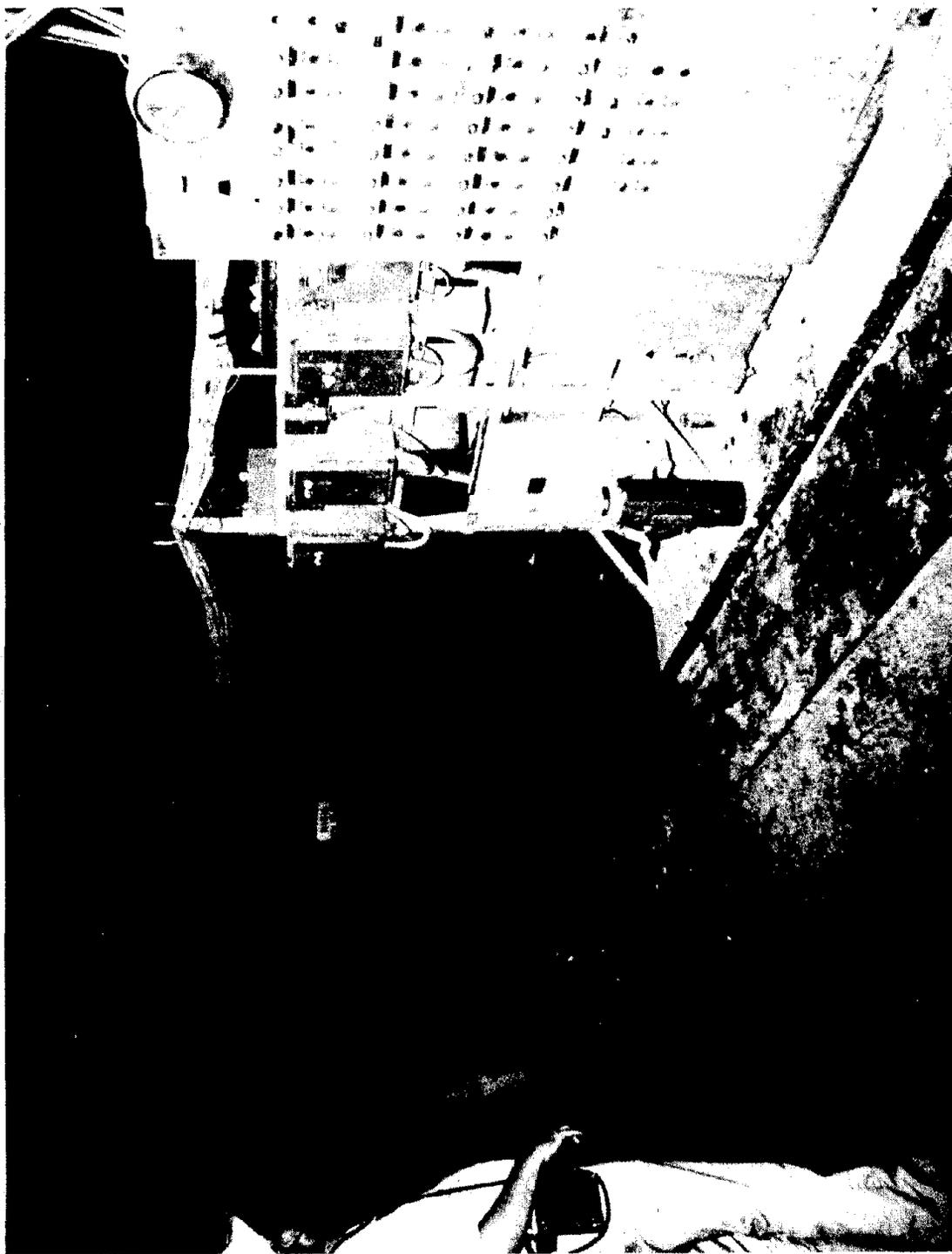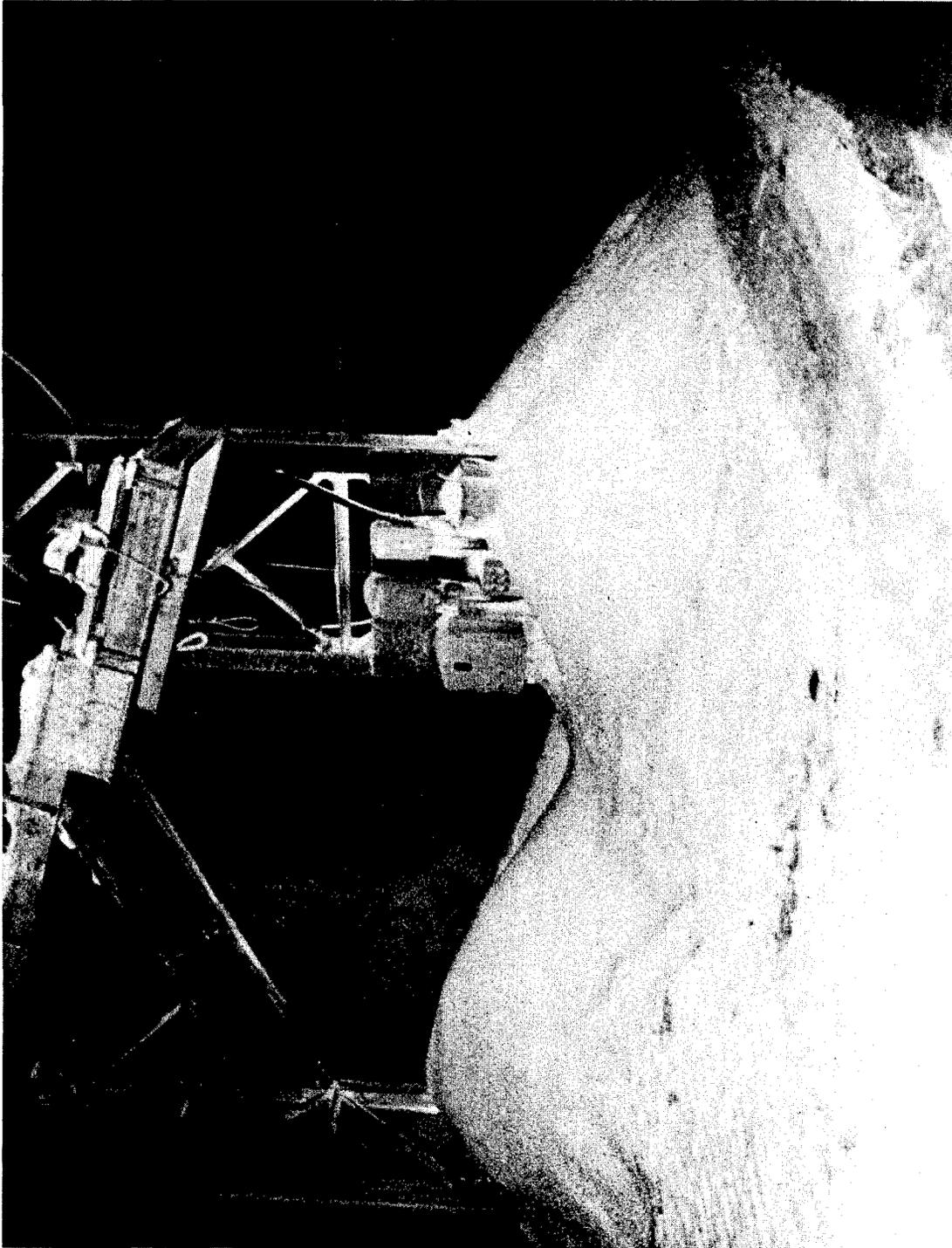
Figure II-2. Typical Power Control Panel for Mine Utilities

Figure II-3. Typical Service Tunnel Carrying Utilities into Mine
(Air, Water, Electricity, Communications)

NOTE: Pile of muck that has fallen from overhead belt is potential path for stray current.

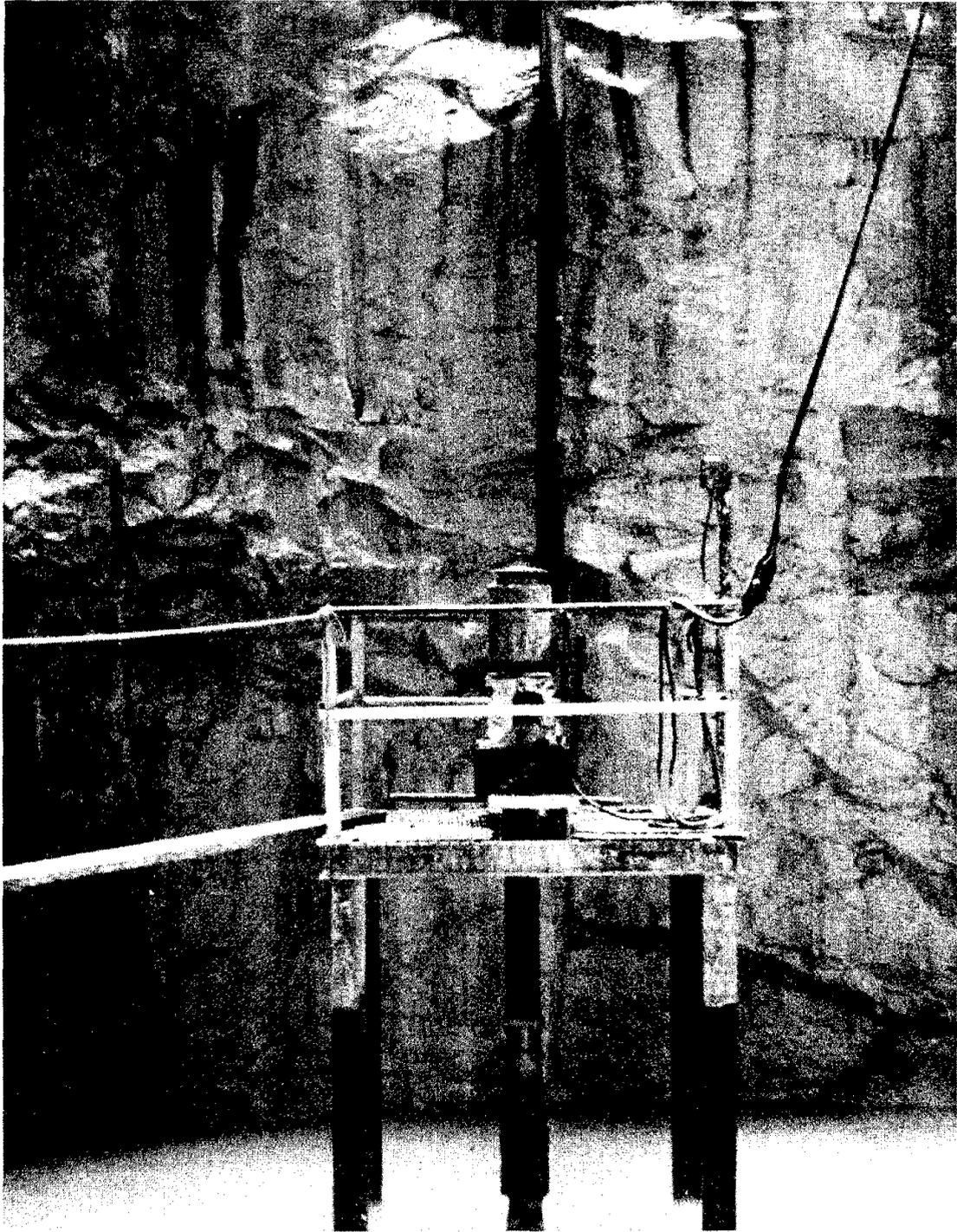Figure II-4. Local Control Box for Conveyor Sections

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Figure II-5. A 440V Water Pump in Quarry Operation (Note: This is a potential source of electricity that has direct contact with quarry ground water)

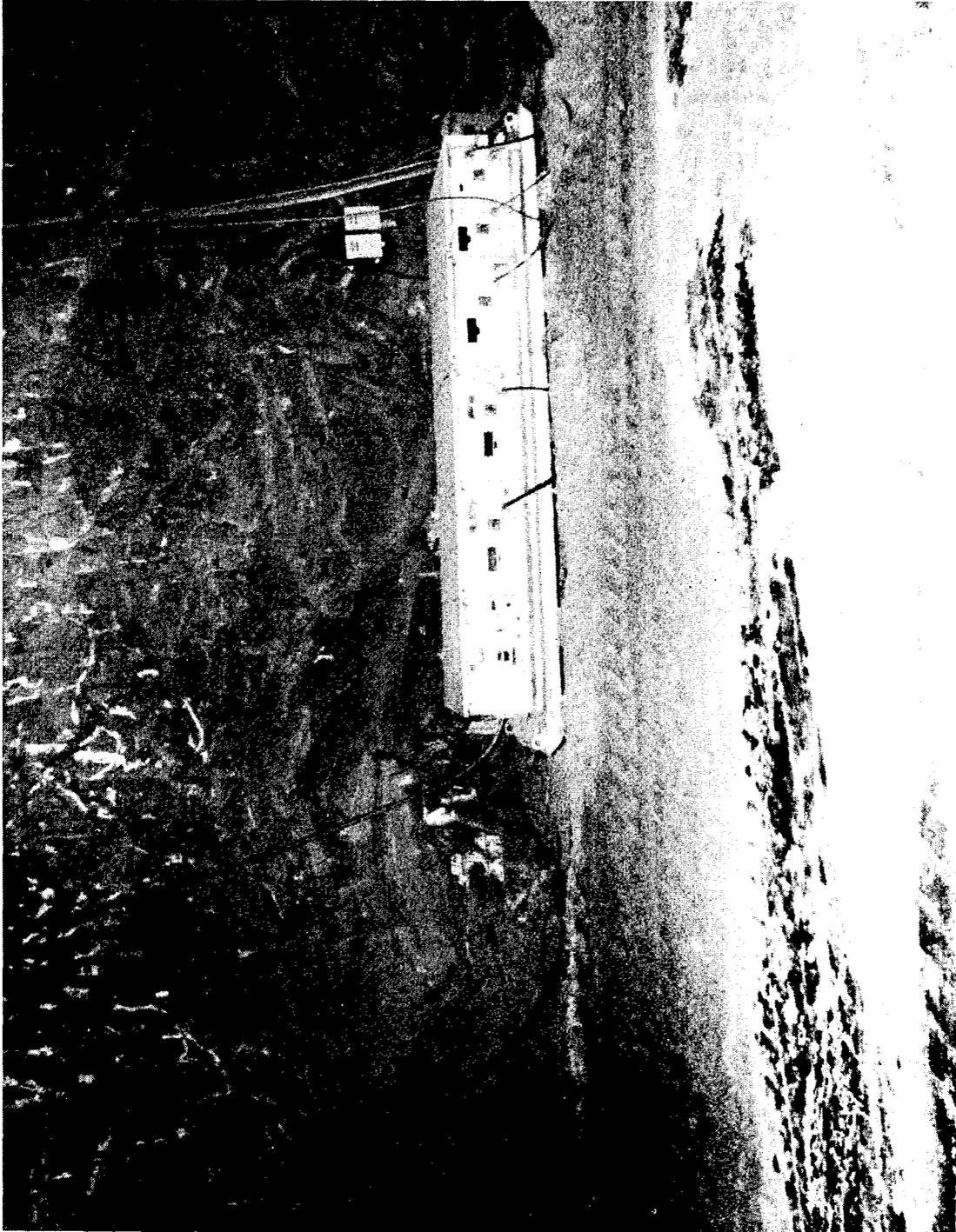Figure II-6. Underground Power Distribution Station

Figure II-7. Portable Power Transformers and Cables (Note: These are moved near electrical crushing, mucking, and hauling equipment in the mine)
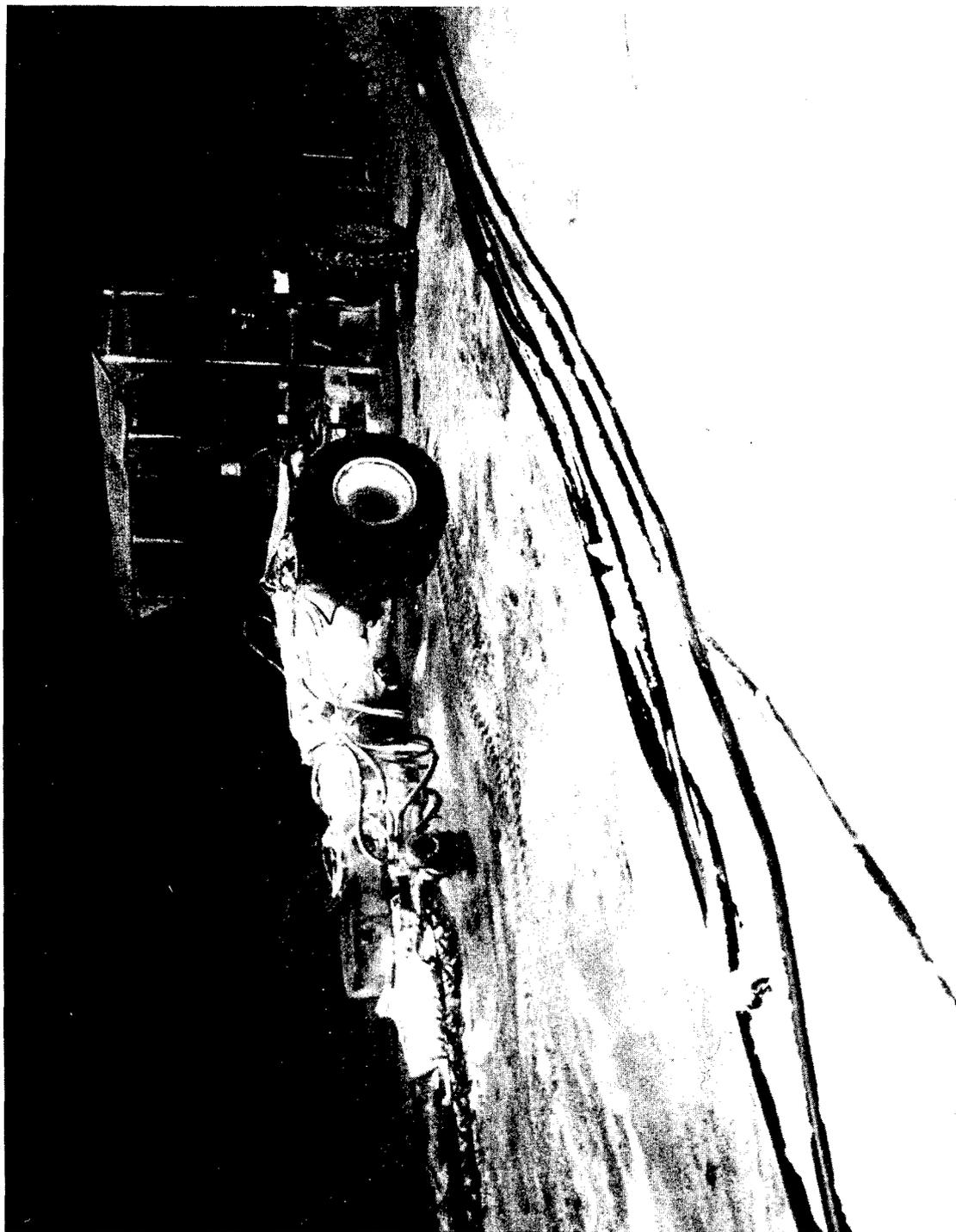
Figure II-8. Diesel/Electric Cutting Machine (Note the trailing power cables in the foreground)

shows an installation for a conveyor unit. Though they are generally located a considerable distance from the blasting site, they nevertheless serve as a potential source of stray currents. The remaining photographs (Figures II-5 through II-8) show a typical installation for electrical service (1) from a distribution station, (2) to a portable transformer, (3) to a pump, and (4) to a cutting machine. Based on these observations, it was not difficult to conceive of damage to the electrical distribution system from rock falls and hence stray current problems.

1. Grounding Techniques

Grounding techniques used in mines were not standard. Various grounding systems that were observed include the following:

a. The best grounding system observed utilized an extensive surface grounding grid at the transformers outside the mine. Another ground was brought into the mine and was distributed to all of the substations. In addition, all of these stations were connected to the earth ground using 6 to 10 ft long copper grounding rods which were surrounded by a 6 in. diameter layer of $CaCl_2$ part way below ground level. All water and air lines as well as rails were grounded to the rods.

b. Another grounding system observed, consisted of grounding the transformer on the surface but not carrying the ground down to the substations. Each substation has a ground rod driven into the floor. All of the machinery was grounded by a cable back to the substation. Pipes were attached to roof bolts and also grounded at the substation.

c. The third grounding system observed had a grounded transformer on the surface. Each substation was grounded to a copper rod driven into the mine floor. The piping was not attached to the ground system but was attached to roof bolts or wall hangers.

2. Ground Fault Interrupters

Ground Fault Interrupter (GFI) systems were observed in use at some mines. The primary purpose of the use of a GFI, in most instances, was to protect personnel from electrical shock. They may also serve as a means for reducing to a safe level stray current leakage, from electrically operated equipment to detonators and other blasting materials. Among the mines utilizing GFI systems, most had no regularly scheduled inspections of the systems to observe the current trip levels. In one mine visited, they experienced unwanted actuations of the GFI systems about once a week indicating current leakage problems. At this time, the mine is involved in non-electric blasting but if the current leakage problem can be solved, the mine

can have the option of blasting with electric detonators safely. The observation that GFI systems were actuated so frequently in mines making use of them was significant; it indicated that current leakage problems need to be addressed.

3. Cables

Many miners fail to recognize potential hazards associated with electrical cables in mines. Cables were observed to have been strung along the floor where they might be run over by machinery or be damaged by scale dropping from the ribs or the back. Unguarded light bulbs in work areas or temporary wiring constitute a source of stray current. Blasters sometimes string firing lines far away from other utilities but electricians later install power lines that intersect the firing line or they tie cables to the same roof bolt.

4. Firing Circuits

During the mine visits, the investigators had the opportunity to review some of the standard techniques utilized in the layout of blasting circuits. The problem areas noted were (1) firing lines contacting machinery, (2) untaped splices inserted into the unloaded bore holes during hookup, (3) variations in wire types, and (4) buildup of electrostatic charges on operators.

The miners interviewed during the visits agreed that firing wires should be composed of solid copper. However, most of them used stranded copper duplex cable because of its lower cost and availability. The type of wire used for firing lines varied even more. It was observed that they ranged from commercial firing lines obtained from explosive distributors to bell cord or reject telephone wire. This constitutes a wide variation in both circuit resistance encountered and capability to withstand abrasion.

These deviations can be detected when the circuit continuity is checked. The common practice observed was that if the needle of the circuit checker moved, the circuit was suitable for blasting. Very few miners observed, calculated the expected circuit resistance and compared it to the measured resistance.

The cables on mucking or hauling machines could be damaged during normal operations. It was observed that machinery often was parked just around the corner from the face and in preparation for blasting, a portable firing line was dropped over the machinery and its cables. Since these machines are a potential source of electrical energy, the location close to the face while loading and blasting is of concern.

B. Explosives Handling and Storage

In many mines observed, the underground magazines contained a sufficient supply of explosives and detonators for several days of blasting. The

method of transporting these materials to the face was quite varied (train, front-end loader, pickup truck, golf cart, ANFO loading truck). In one mine the blaster put stick powder in a conductive plastic bag which was placed in a wooden box and dragged with a rope to the various faces to be loaded. While it was apparent that the magazine storage area was being properly organized and managed, safety precautions were relaxed when transporting blasting materials for blasting or disposal. Figure II-9 illustrates a case where blasting materials were being transported properly – here a well-marked, electrically operated cart having a separate box for detonators and sufficient blasting materials for only one blasting round was being used. A poor storage practice is shown in Figure II-10 where damaged explosives, that were removed on about a weekly basis, were stored with blasting materials ready for use. Figure II-11 shows a violation of regulations; stick powder, primed with detonators, was being transported to the face.

Poor handling practices can and have resulted in incidents due to contact with electrical or electrostatic stimuli. The regulations in this area are sufficient if enforced.

C.   Pneumatic Loading of ANFO

Pneumatic loading of ANFO presents a potentially hazardous situation since electrostatic charge generation, from the movement of the ANFO in the loading hose, is a well-known phenomenon. The concern is primarily associated with its use with electric detonators but may apply to other blasting materials or personnel as well.

1.   Equipment

It was observed that there exists much diversity in the pneumatic delivery equipment used. Some of the pots (containers) used were simply containers manufactured for other purposes. In one case a container, initially intended for sand blasting, was used; it could be loaded with ~50 lbs of ANFO. Others used were truck mounted units having a capacity of 250 lb. Figure II-12 shows a one-man loading operation with the operator guiding the hose with one hand while he operated a full throat ball valve with the other hand. In this case, it is more difficult to control the pressure and hence control the loading rate or density. In two man operations observed, a three way valve was operated by one man while the other one did the loading; the loading rate and density were easily controlled here. Figure II-13 shows a larger capacity "prill pot" mounted on a truck with the control valves mounted on the pot. One man controlled the pressure at the truck, and the other one did the loading; he had the capability of loading from the mine floor or from a hydraulic boom. Figure II-14 shows a typical 2 man operation.

In all cases observed, the air was compressed at the utilities area of the mine or on the surface. Some moisture had been removed from the air by virtue of the compressing process.

Figure II-9. Electric Explosives Cart for Transporting Explosives to Blast Site

Figure II-10. Storage of Explosives Underground (Note the debris waiting for pickup and disposal)
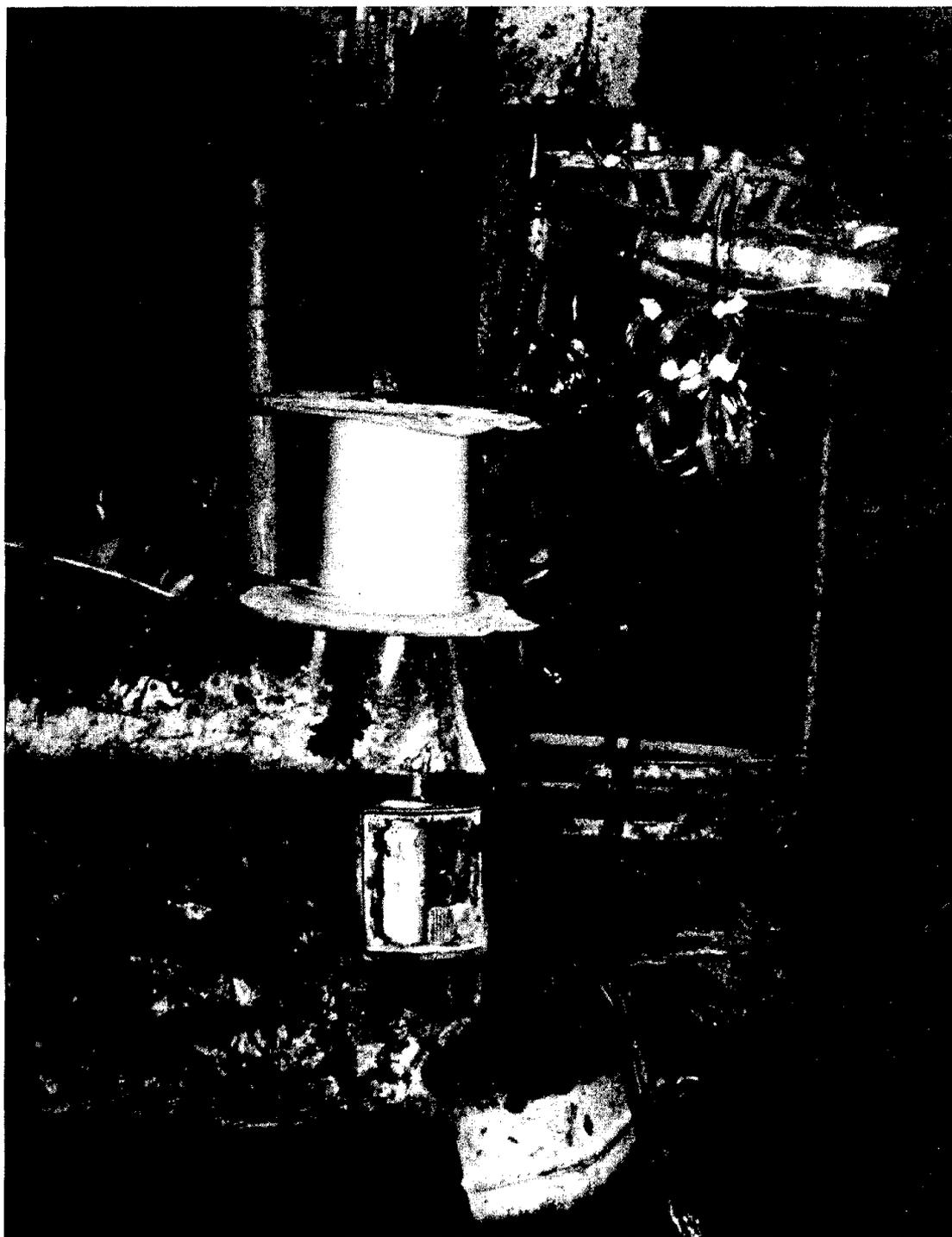
Figure II-11. Stick Powder Prepared with Hercudet Caps

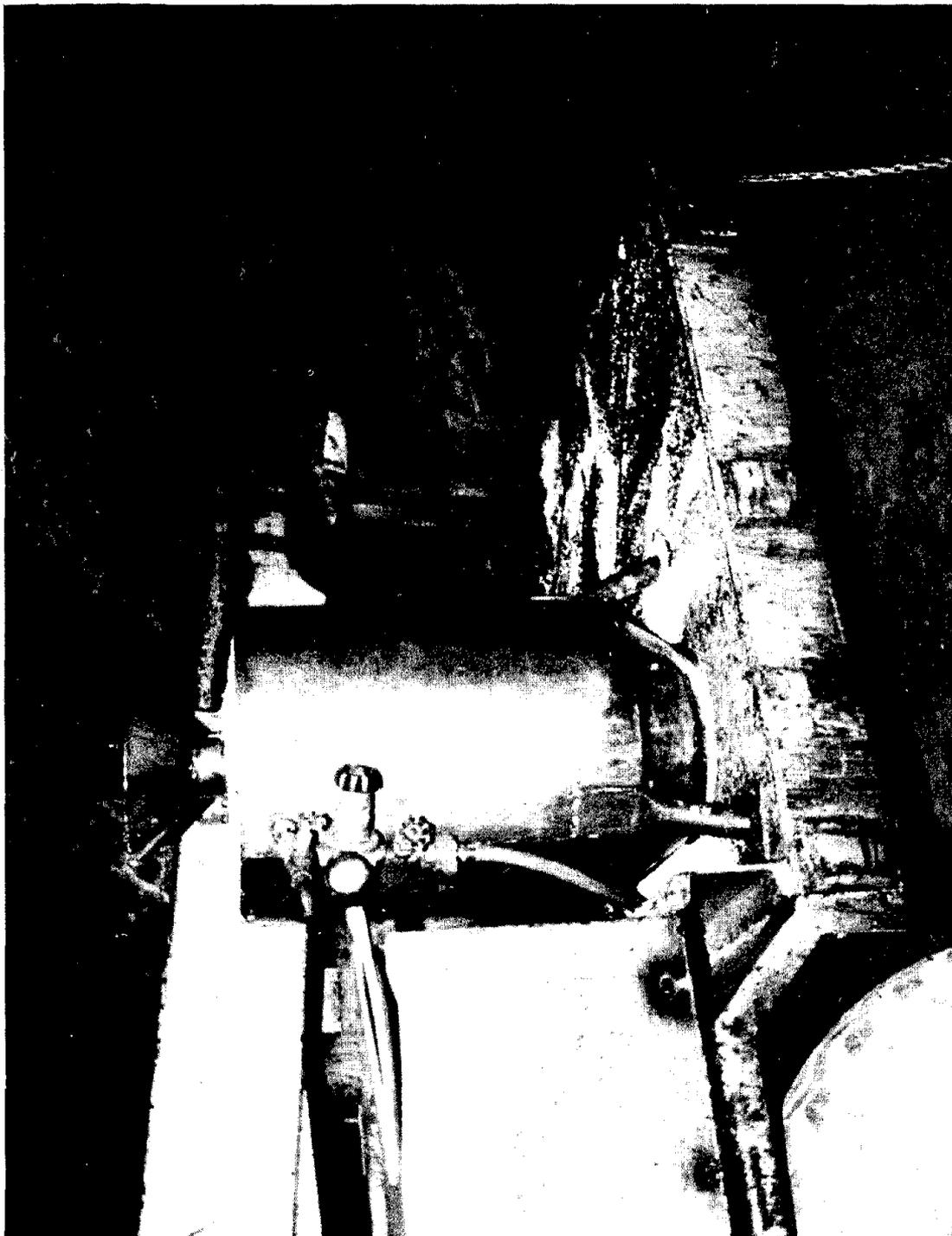Figure II-12. One Man ANFO Loading Technique

Figure II-13. ANFO Prill Pot Mounted on truck (Note the grounding chain attached to the truck bed)

Figure II-14. Miner Loading Bore Hole with ANFO in Typical Two Man Operation

These details, while not entirely related to electrostatic hazards, have been cited to illustrate some of the diversity in equipment and loading techniques observed, and the foregoing is not intended as a complete survey of the subject.

2. Semi-conducting Loading Hose

The use of semi-conducting hoses for loading ANFO is mandatory in the Federal Regulations (30 CFR 57.6-195). In all observations made on the various mine visits, semi-conducting hose was being used. However, the observers were told, by miners, of alleged cases where personnel in other mines experienced electrostatic shocks from hoses that were nonconductive. No incidents of ignition were reported.

Electrostatics is of particular concern in this operation since one of the common observed loading methods is to load over the cap and lead wires. Typically, an electric detonator with a small booster is placed in the end of the ANFO loading hose. The cap wires are untangled and slid along with the hose to the back of the hole. The air valve is opened pushing the ANFO into the hole as the hose is backed out toward the collar. The alleged electrostatic shocks experienced by miners could also be a source of premature explosion in the hole.

3. Grounding

Five different methods for grounding ANFO loading equipment were observed; these are given below:

a. A grounding cable from the pot to a roof bolt.

b. A grounding cable from the pot or truck to a stake pushed into a damp muck pile.

c. A chain attached to the pot or truck with the other end (sometimes with a plate attached) dragging on the mine floor.

d. A chain and clip arrangement with the clip attached to the pot or valve with the other end inserted into the borehole.

e. A chain and clip arrangement with one end clipped to the pot or truck and the other end clipped to a water pipe or messenger cable.

While several of the other grounding methods are not deemed satisfactory, the method under (e) was in direct violation of Federal Regulation 30 CFR 57.6-194 which prohibits grounding to water lines, air lines, rails or to permanent electrical grounding systems.

D. Bulk Loading

Several surface mine operations were visited with particular emphasis on observing procedures for bulk loading of ANFO or slurry. Plastic liners were used in some of these operations. Figures II-15 and II-16 show primed holes with and without liners. Federal Regulation (30 CFR 57.6-198) prohibits the use of plastic hole liners only for pneumatic loading of blasting agents into holes containing electric detonators. None of these operations used pneumatic loading techniques since the holes were vertical down holes; however, nonelectric initiation systems were being used in the two cases where liners were used. In both cases (a coal stripping operation and a quarry operation) the ANFO was loaded from bulk ANFO trucks fitted with an auger. In the coal operation, the Hercudet·initiating system was used while an arrangement of detonating cord and boosters was used as the initiator in the quarry operation. In another quarry operation, utilizing electric detonators, no plastic liners were used and the ANFO was poured from 50-lb bags into the hole by hand. These three cases are quite typical of bulk loading operations.

E. Lightning Detection

Lightning is a major concern in electric blasting. It was observed that relatively few miners were aware that lightning storms can present hazards even when at large distances from the blasting site. Weather reports or personal observations constituted the lightning warning system used at most mines visited. Criticism of lightning detectors ranged from false alarms to the inability of miners to hear the audible warning alarms over the noise of machinery. At most mines, blasting activities are halted if there is evidence of an electrical storm in the vicinity and in some underground gassy mines visited, personnel are evacuated out of the mine when electrical storms approach.

One mine visited had previously experienced a premature blast from a· lightning strike. This mine was quite wet and close to the surface. The grounding grid was connected to the skip tower as well as to the water and air pipes. It was reported to the observer that they believed that ground currents from the strike were picked up by the grid and conducted into the mine, causing the premature blast.
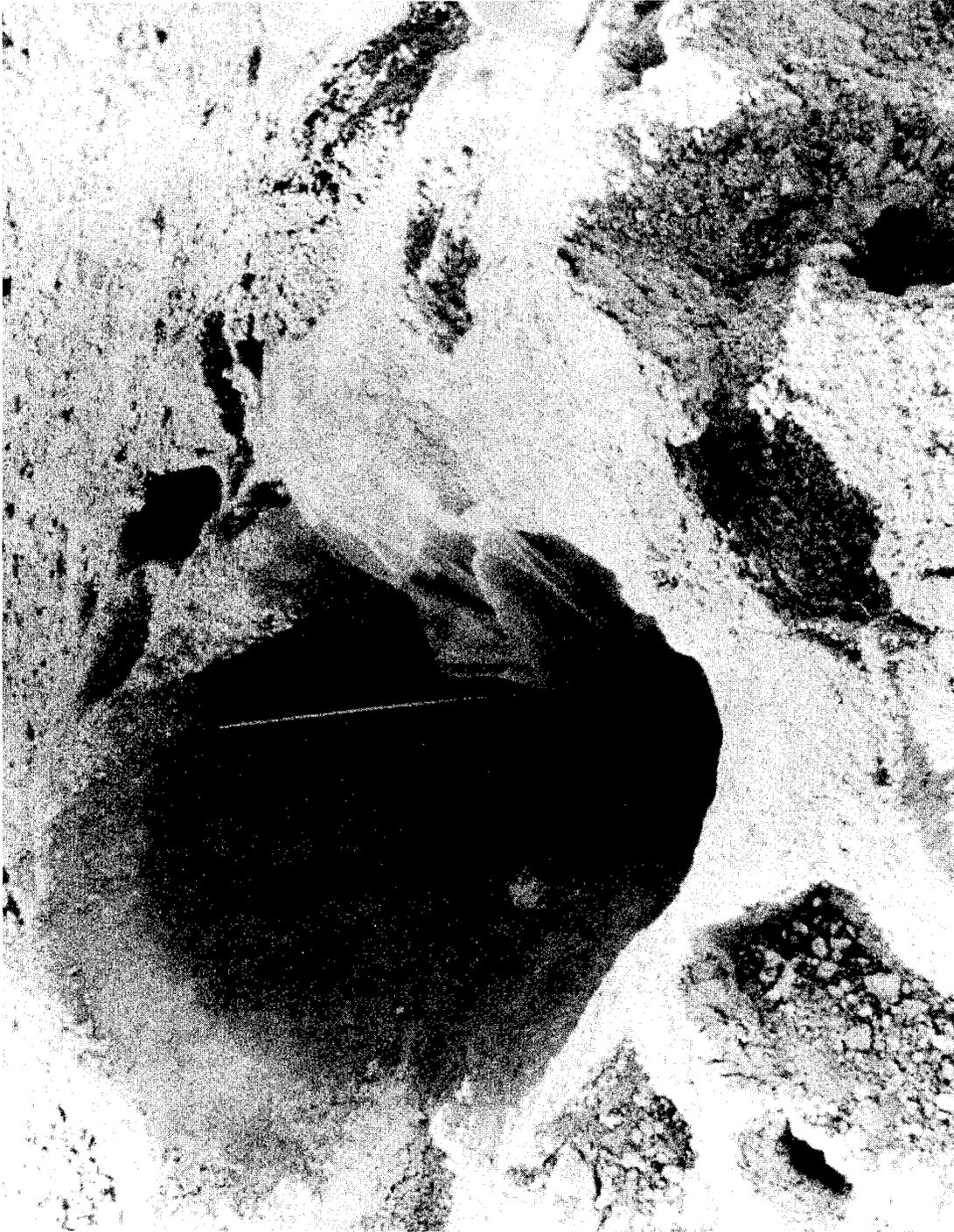
Figure II-15. Primed Borehole with Plastic Liner Prepared to Receive ANFO from Bulk Delivery Truck

Figure II-16. Primed Borehole without Liner in Process of Being Filled from Overhead Auger Delivery Truck

III. SENSITIVITY OF TYPICAL DETONATORS

Electric detonators are designed to be activated by the application of electrical current to the device. Typically, the current heats a bridge wire which causes an ignition charge to react which in turn causes a primary charge to react which causes the main charge (base charge) to detonate. If the cap contains a delay element, it is situated between the ignition and primary charges and it introduces a time delay between application of the current and detonation of the base charge. Variations in the typical arrangement of active elements cited may be encountered.

Electric caps are subject to initiation by electrical or electrostatic energy during use if sufficient stimulus is applied across the bridge wire or between the bridge and the case. These two modes are of primary interest where R.F., stray current, and electrostatic stimuli are present in mining operations. The test data available to define sensitivity to electrical stimuli are characterized as follows:

1. Maximum Nonfiring Current (MNFC)

    In this test, individual caps are subjected to a continuous current for 30 seconds. The maximum current at which these caps fail to fire is known as the maximum nonfiring current.

2. Minimum Firing Current (MFC)

    The minimum current that will ignite an individual cap within 0.5 second after application of the current is known as the minimum firing current.

3. Critical Firing Current (CFC)

    The current that will detonate 30 caps of the same type, when connected in series, is known as the critical firing current.

4. Mixed Critical Firing Current (MCFC)

    The current that will detonate 30 caps of mixed types (instantaneous and delay caps) when connected in series, is known as the mixed critical firing current.

5. Electrostatic Sensitivity - Maximum Nonfiring Voltage

    The maximum voltage at which a cap will not detonate when subjected to a discharge from various rated capacitors when this discharge is directed from shunt to shell or through the bridge wire is known as the maximum nonfiring voltage.

Commercial electric detonators are designed to have a minimum firing current of more than 0.25 amps. This lower limit establishes the guideline for determining safe limits on stray currents to prevent accidental detonation of explosives.

Several sources were reviewed to determine the firing current for presently available caps. The values found range from 0.20 to 0.54 amps. The data are summarized in Tables III-1 and III-2:

TABLE III-1. COMPARISON OF ELECTRICAL CHARACTERISTICS
OF ELECTRICAL DETONATORS

Manufacturer	Minimum Firing Current[a] (amps DC)	Max Non Firing Current[b] (amps DC)	Voltage Breakdown[c] KV
Manufacturer A			
Type 1	0.35	0.20	1.22
Type 2	0.30	0.20	1.385
Manufacturer B			
Type 1	0.45	0.35	3.935
Type 2	0.40	0.30	2.025
Manufacturer C			
Type 1	0.65	0.50	2.310
Type 2	0.60	0.45	1.830

[a] 5 consecutive caps shall be ignited within 0.5 seconds after application of the specific current.
[b] 10 consecutive caps shall not fire after application of the specified current for 30 seconds.
[c] The cap shall not exhibit current leakage in excess of one milliamp when tested shunt to shell

TABLE III-2. MAXIMUM NO-FIRE CURRENTS FOR VARIOUS DETONATORS
OBTAINED FROM THE BUREAU OF MINES[6]

Detonator*	TIL** Current, Amp
Manufacturer No. 4	
Type 1	.31
Type 2	.26
Type 3	.54
Type 4	.48
Type 5	.53
Type 6	.48
Type 7	.30
Manufacturer No. 2	
Type 1	.34
Type 2	.32
Type 3	.33
Type 4	.33
Type 5	.46
Manufacturer No. 5	
Type 1	.25
Type 2	.24
Type 3	.26
Type 4	.25
Type 5	.25
Manufacturer No. 6 (Foreign)	
Type 1	.35
Type 2	.35

_____

*All detonators utilized a copper alloy shell unless other-
wise indicated; current applied for 10 sec.
**Highest test level at which 10 non-initiations were ob-
served; test interval was 10 ma.

A. Electrostatic Sensitivity

The sensitivity of electric detonators to short electrical pulses was
recently investigated by Southwest Research Institute. It was found that
blasting caps from different manufacturers and even different types of caps
from the same manuafacturer exhibited markedly different responses to pulsed
electrical excitation. The range of values was from 2.5 to 10 milli-
joules.[20]

1. Electrostatic Sensitivity

The Bureau of Mines performed tests to assess the electrostatic sensitivity of a variety of commercial detonators; the electrical parameters of their circuit were a 10kV potential, variable capacitance and zero ohmic resistance. Results of these tests are presented in Table III-3 and they show that the threshold initiation level (TIL) values ranged from 16 to 350 mj in tests where the discharge was applied to the leg wires or pin-pin (P-P) mode. In tests where the discharge was applied between the shunted leg wires and the detonator's metal case (designated as the pin-case (P-C) mode), the TIL values observed ranged from 36 mj to greater than 12 j. Thus, overall, commercial detonators appear to be less sensitive to initiation from "static" discharges in the P-C mode. Note the values of less than 6 mj that were obtained for the first results in the table; this was for a military designation detonator that contained no anti-static protection devices whereas all the commercial detonators did have anti-static devices or features of one type or another (both P-P and P-C protection). These results suggest that the various anti-static features are effective.

The Bureau of Mines also performed another test on all the detonators listed in Table III-3. This was a test to observe their susceptability to initiation from electrostatic discharges from human beings. The test parameters in these tests were a 20 kV potential, variable capacitance and 5000 ohms resistance. These parameters were adopted as a result of a study by Amicone.* The capacitor energy here is 100 mj and while this exact value is possibly subject to dispute, it is viewed by Bureau of Mines personnel as more or less a worst case situation since it is believed that one is more likely to encounter much lower energies (perhaps 15-50 mj) stored on humans.

Results of these tests showed that none of the commercial detonators could be initiated in this "worst case" test; however, the military item did so.

The Bureau of Mines also performed electrostatic sensitivity tests on nonelectric detonators designed for use with safety fuse. In these tests they inserted a needle shaped pin electrode into the fuse well to within 1.6 mm (1/16 in) of the ignition charge. The electrode was insulated from the metal shell of the detonator and the detonator case was grounded; these tests were similar to pin-case tests performed with electric detonators except the potential used was 5kV. Results of these tests are presented in Table III-4; the TIL values for three manufacturers' detonators ranged from less than 4 to 14 mj. Thus the nonelectric detonators appear to be significantly more sensitive to electrostatic discharges than their electric counterparts. Though there apparently exists limited published information on the effectiveness of anti-static devices in detonators, the results from the

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*Amicone, R. G., C. T. Davy and J. B. Campbell, "Electrostatic Hazard to Electroexplosive Devices From Personnel-Borne Charges," Franklin Institute Monograph 65-1, The Franklin Institute Research Laboratories, Philadelphia, Pa., Feb. 1965, 26 pp.

TABLE III-3. RESULTS OF ELECTROSTATIC SENSITIVITY TRIALS
FOR VARIOUS ELECTRIC BLASTING CAPS[1,2]

Detonator Designation[c]	Test[a,b] Mode	TIL Energy mj
1 Manufacturer No. 1		
(Military EB Cap)	P-P[d]	16
(Military EB Cap)	P-C[e]	16
1 Manufacturer No. 2		
No. 6 Instant	P-P	250
No. 6 Instant	P-C	4000
2 Manufacturer No. 2		
Delay--100 ms	P-P	112
Delay--100 ms	P-C	500
3 Manufacturer No. 2		
Delay--250 ms	P-P	56
Delay--250 ms	P-C	4000
4 Manufacturer No. 2		
No. 8 Instant	P-P	56
No. 8 Instant	P-C	4000
5 Manufacturer No. 2		
Instant, steel shell	P-P	56
Instant, steel shell	P-C	4000
1 Manufacturer No. 3		
No. 6 Instant	P-P	16
No. 6 Instant	P-C	4000
2 Manufacturer No. 3		
No. 6 Delay No. 1	P-P	20
No. 6 Delay No. 1	P-C	12500

TABLE III-3. (CONTINUED)

Detonator Designation[c]	Test[a,b] Mode	TIL Energy mj
1 Manufacturer No. 4		
Instant No. 6	P-P	56
Instant No. 6	P-C	112
		4000
		(repeat)
2 Manufacturer No. 4		
Instant No. 8	P-P	56
Instant No. 8	P-C	1000
3 Manufacturer No. 4		
Delay--25 ms	P-P	36
Delay--25 ms	P-C	112
4 Manufacturer No. 4		
Delay--25 ms with steel shell	P-P	56
Delay--25 ms with steel shell	P-C	112
5 Manufacturer No. 4		
Delay--175 ms	P-P	36
Delay--175 ms	P-C	56
6 Manufacturer No. 4		
Delay--25 ms with steel shell	P-P	36
Delay--25 ms with steel shell	P-C	36
7 Manufacturer No. 4		
Delay--325 ms	P-P	36
Delay--325 ms	P-C	500
8 Manufacturer No. 4		
Delay--25 ms with steel shell	P-P	36
Delay--25 ms with steel shell	P-C	112
9 Manufacturer No. 4		
Delay--1000 ms	P-P	36
Delay--1000 ms	P-C	112

TABLE III-3. (CONTINUED)

Detonator Designation[c]	Test[a,b] Mode	TIL Energy mj
10 Manufacturer No. 4		
Delay--25 ms with steel shell	P-P	36
Delay--25 ms with steel shell	P-C	500
11 Manufacturer No. 4		
Delay--500 ms, Al shell	P-P	36
Delay--500 ms, Al shell	P-C	112
1 Manufacturer No. 5		
Instant	P-P	16
Instant	P-C	4000
2 Manufacturer No. 5		
Delay--50 ms	P-P	56
Delay--50 ms	P-C	4000
3 Manufacturer No. 5		
Delay--2.9 sec	P-P	36
Delay--2.9 sec	P-C	12500

[a]Test parameters: constant potential - 10 kV; variable capacitance; zero series resistance.

[b]All EBC's utilized copper alloy outer shells unless otherwise indicated. Numbers in parenthesis after detonator designation indicate BuMines' identification number.

[c]Numbers in parenthesis after detonator designation indicate BuMines identification number.

[d]P-P (pin-pin) - indicates that the discharge was applied across the two separated leg wires, i.e., as in the normal initiating mode.

[e]P-C (pin-case) - indicates that the discharge was applied with the high potential lead on the shunted leg wires and with the ground lead on the outer metal shell (case) of the detonator.

NOTE: All leg wires were shortened to a length of 7.6 cm (3 in.) for all tests.

electric detonator with no apparent anti-static features and the results from the nonelectric detonators suggest that the anti-static devices in commercial electric detonators provide at least some protection against initiation from electrostatic stimuli.

An effective method for reducing the electrostatic and stray current hazard to electric detonators is to keep them shunted until the final hook-up in the blasting cycle. Manufacturers invariably have their detonators shunted, by one means or another, when they are distributed. Hercules has measured the shunt resistances of various companies' detonators and found that there exists a variation in resistance among detonators from the same manufacturer as well as differences in average resistance values among different companies' detonators. Typical results of these measurements are shown in Table III-5 for various companies' detonators together with data for a case where bare lead wires were simply twisted together. The shunt resistances observed for commercial detonators range overall from 0.017 to 4.0 ohms. Some of the resistance values observed approached or exceeded the legwire plus bridgewire resistance and needless to say, in these cases, one-half or more of the current from an inadvertent electrostatic discharge or stray current would flow through the bridgewire. It can be demonstrated that some of these "shunted" detonators could be initiated with a 1.5 volt flashlight battery. These results are given simply to emphasize that good shunting methods should be used and good quality control of a given shunting method maintained for maximum safety.

TABLE III-4. RESULTS OF ELECTROSTATIC SENSITIVITY TRIALS
FOR SEVERAL NONELECTRIC, FUSE BLASTING CAPS[7]

Detonator[a]	Test[b] Mode	TIL Energy, mj
Manufacturer No. 2 No. 6 Strength (C-1644)[c]	Point electrode	14
Manufacturer No. 4 No. 6 Strength (C-1423)	Point electrode	4
Manufacturer No. 5 No. 6 Strength (C-1726)	Point electrode	14

[a]All detonators utilized copper alloy outer shells unless otherwise noted.

[b]Test parameters: constant potential - 5 kV; variable capacitance; zero series resistance.

[c]Numbers in parenthesis after detonator designation indicate Bureau of Mines identification number.

TABLE III-5. RESISTANCE OF TYPICAL SHUNTS

	Average Resistance[a] (Ohms)	Maximum	Minimum	Range
Mfg. No. 5 Tape Plus Foil	0.027	0.047	0.020	0.027
Mfg. No. 2 Coiled Wire (Spring)	0.38	0.75	0.05	0.7
Mfg. No. 2 Clip	0.019	0.022	0.017	0.005
Mfg. No. 4 Rolled Aluminum	2.5	4.7	0.7	4.0
Twisted Wire[b]	0.024	0.027	0.021	0.006

[a]The resistance was measured from one lead wire entering the shunt to the second wire leaving the shunt.

[b]In this case, two bare leads were twisted together and the resistance measured.

IV. ENGINEERING ANALYSIS

Information obtained from the accident/incident survey as well as that obtained from mine visitations was instrumental in establishing a basis for making assessments on potential hazards from electrostatic sources, stray current sources and lightning. Before specific recommendations can be offered, potential hazards must be examined in the light of sources of electrical stimuli and sensitivity of the blasting materials or devices involved. This, combined with results of other experimental and theoretical studies, can form a basis on which to make assessments. This section is divided into seven elements, six of which represent particular operations or electrical sources that were associated with potential hazards while the remaining element constitutes a review of methods for detection and minimization of hazards.

A.   Pneumatic Loading of ANFO in Underground Blast Holes

ANFO is a blasting agent containing ammonium nitrate prills with 5-6% No. 2 fuel oil. The mixture is free flowing and as such can be pneumatically loaded into boreholes. Pneumatic loaders are of two basic types — eductors (utilizing the venturi principle) and pressure pot loaders where the ANFO is contained in the pot under constant pressure.

In 1968, the Bureau of Mines studied the subject of pneumatic loading[8] and offered specific recommendations for minimizing electrostatic hazards. These recommendations are repeated below.

RECOMMENDED PRACTICES AND PROCEDURES FOR MINIMIZING
ELECTROSTATIC HAZARD

In addition to all normal safety practices in the handling of electric blasting caps or electric bridgewire detonators, it is strongly recommended that the operator:

1.   Test to assure continuity of the electric detonator legwire bridgewire-legwire circuit prior to insertion of the cap into charge or borehole. This testing should be accomplished with an electric blasting cap galvanometer suitable for use with the type of cap being employed in the blasting operation.

2.   Keep the legwires shunted but not otherwise connected to ground during loading of the blasting agent into the borehole. This will minimize the possibility of extraneous stray currents through the legwire to shell circuit. Detonators with plastic shells, and with the insulation on the lead wires in good condition, may have the shunted legwires grounded if desired.

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3. Assure that the borehole is discharged prior to hooking up the detonator leads. This assurance is provided by burden resistivities of less than about 10^7 ohm cm, reduction of the charge delivered to the borehole and discharge of the borehole during loading through control of resistivity of the ANFO and use of a semi-conductive loading tube, and dampening of the shot hole walls and use of an additional return path from the borehole.

4. Assure that the operator is discharged prior to handling the detonator leads. The simplest way of assuring that the operator is discharged is to have him discharge himself immediately prior to handling the leads.

5. Assure that the blasting cap leads are never carried to the ANFO loader. This will further minimize possible hazards from stray currents and hazards from such things as charged loader cases.

6. Assure that a semi-conductive loading tubing is utilized.

Adherence to these recommendations (1, 2, and 3) should remove any electrostatic hazards associated with the bottom-primed hole. Collar priming remains preferable, since it minimizes the probability of mechanical damage to the cap. Collar priming also minimizes any hazard from stray currents, from the power distribution system or from lightning, by minimizing the "exposure time" of the initiator. In collar priming, assurances that the borehole is discharged can be obtained as simply as with the operator: a grounding wire can be attached to two metal plates one of which is positioned near the operator's feet and the other brought into contact with the loaded column of ANFO prior to insertion of the priming charge.

Pneumatic loading of ANFO was observed during the mine visitations. It was found that collar blasting was not being used routinely and that the recommendation to use semiconductive hose was the only practice routinely followed.

Problem Area

The potential hazard is the generation and accumulation of static charges during pneumatic loading.

As a result of tests performed by Prugh and Rucker,[9] at 23 locations in 18 mines as well as seven above-ground locations, a correlation between current carried by blasting agent particles with absolute humidity and loading rate was obtained. Their equation for the relationship had the following form:

$$I = \frac{600f}{(H_E)^3} \text{ microamperes}$$

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where f and H_E were the loading rate (lb/min) and absolute humidity (grains of moisture/lb of dry air) respectively.

Using a previously published value[10] for the loading rate of an APM "powder monkey" of 25-90 lb/min and an absolute humidity value of 10 grains/lb (typical value expected for the air under pressure in this system), the current would be:

$$I = (600)(90)/10^3 = 54 \ \mu\text{amps} \quad \text{or} \quad 5.4 \times 10^{-5} \ \text{coul/sec}$$

As is well known, current values of 50 ma or less are considered acceptable current levels when taking blasting circuits measurements with blasters galvanometers or ohmmeters, hence, in this particular case the safety factor is about 1000. The loading rate would have to be increased by about a factor of 1000 or the humidity value decreased by a factor of 10 to realize the 50 ma current level. In fact, no conceivable attainable combinations of loading rate and humidity values will cause the current to reach the 50 ma level. Prugh and Rucker also found that the amount of current generated by moving ANFO prills is relatively independent of whether or not the hose was conductive or not. Thus, since modern commercial detonators have earlier been shown to require a current of about 250 ma for initiation, the hazard of initiating electric blasting caps by currents generated in this method appears to be negligible.

While little hazard exists from these currents, it should be emphasized that a potential hazard may exist if these moving charges are permitted to accumulate in sufficient quantity and potential on the loading equipment, accessory equipment, the borehole or possibly the operator. In Table III-3 the Bureau of Mines[7] cited threshold initiation values for the electrostatic initiation of commercial detonators. The lowest value measured was 16 mj in the legwire to legwire mode and 36 mj when applied between the shunted legwires and the outer metal case of the detonators. These values are in substantially good agreement with Prugh and Rucker's corresponding values of 10 mj and 30 mj.

To reduce the hazard from electrostatic charge buildup on the equipment, borehole, etc., the generated charges should be dissipated continuously so that the accumulated energies are at all times well below that which would initiate detonators. To gain a perspective on the relaxation time i.e., (the time required for a charge to dissipate by leakage, which is equal to $\epsilon \epsilon_o / \gamma$ or RC), we will consider a typical case, that would apply if the rock strata were relatively conductive or if semi-conductive hose were used.

The relaxation time may be calculated using the expression

$$t_r = \epsilon \epsilon_o / \gamma$$

where: t_r is the relaxation time
      ϵ is the dielectric constant = 2.6
      ϵ_o is the permittivity in a vacuum = 8.85×10^{-14}
      γ is the conductivity = 4×10^{-14}

The main factor resisting the transfer of charge is the low conductivity of ANFO. The conductivity is variable depending upon the make-up of the ANFO, any contaminates or minor constituents, and the moisture present. The conductivity of several samples of ANFO was determined: (1) for a relatively dry, uncontaminated mixture of AN prills and fuel oil, $\gamma = 4 \times 10^{-14}$ mho/cm, (2) for a commercial ANFO sample, $\gamma = 2 \times 10^{-12}$ mho/cm and for a special aluminized ANFO, $\gamma = 1.2 \times 10^{-8}$ mho/cm. The relaxation time, t_r, for these three samples would be (1) 5.75 sec, (2) 0.12 sec, and (3) 2×10^{-5} sec, respectively. The semi-conductive hose and/or the rock strata should have a conductivity greater than that of the ANFO (standard) to minimize the hazardous accumulation of static charges.

On the other hand, if the borehole is relatively nonconductive and if nonconductive loading hose is used then a hazardous buildup of static charges can occur as described below.

Evaluation of Stored Electrical Energy in a Pneumatically Loaded Hole Without Charge Dissipation

Example Loading Conditions

o Hole is 20 feet deep and 2 inches in diameter
o Loading rate (f) is 90 pounds of ANFO per minute
o ANFO density $\sim$ 50 pounds per cubic foot
o Bore hole electrical capacitance estimated to be 20 pf/ft[8]
o Humidity (H_E) is 10 grains/lb of dry air

The electrical charge can be estimated by considering the electrical current (I) produced by the operation in the equation $I = 600\ f/H_E^3$. For this example, $I = 600\ (90)/10^3 = 5.4 \times 10^{-5}$ coul/sec. The loading time can be determined by the relationship

$$t_1 = dV/R$$

where: t_1 is the loading time
         d is the loading density of ANFO = 0.029 lb/in.^{3}
         V is the volume of the hole   = 754 in.^{3}
         R is the loading rate   = 90 lb/min = 1.5 lb/sec

Therefore in the example, $t_1 = 14.6$ sec and the electrical charge ($I \times t_1$) is 7.9×10^{-4} coul.

The electrical energy can be estimated by using the equation for the energy of charged condensers,

$$E = \frac{Q^2}{2C}$$

where: Q = charge   = 7.9×10^{-4} coul
         C = capacitance of borehole = 20 ft borehole length x
             20 pf/ft = 400 pF

$$E = \frac{(7.9 \times 10^{-4})^2}{2(400 \times 10^{-12})} = 780 \text{ Joules}$$

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This calculation results in an energy that is believed to be a gross overestimation erring in the direction of safety. It is believed that a more realistic energy value would be about 0.02 Joule. This is based on Hurry's[11] observations of 10,000 volts potential which when used in the equation $E = 1/2 \, CV^2$ yields 0.02 Joules. Yet, this value must be considered to be a hazardous energy, especially to electric blasting caps.

B. ANFO and Slurry Bulk Loading in Surface Mines

Bulk loading of explosives at surface mine operations is becoming a standard technique. This is accomplished with bulk mix/load trucks and by operators pouring ANFO prills manually from plastic lined paper bags. The potential problems associated with these operations are somewhat different than those associated with underground pneumatic loaders. The principle differences are a result of (1) the large size of the bore holes (length and diameter) which increases the electrical capacitance, (2) the use of plastic liners which inhibits the relaxation of electrostatic charges, and (3) the potential sources of ignition due to the presence of a bulk delivery truck.

When the bulk explosive is a slurry, the material conductivity due to the water content decreases the electrostatic hazard. Some of the ingredients, however, can increase the sensitivity of spilled materials or abnormal mixes due to the presence of aluminum powder, gun propellant, etc.

The presence of an electrostatic hazard depends on how well the ground can dissipate the charge entering the hole and the sensitivity of the explosive material.

The evaluation of the energy was approached theoretically, utilizing the following empirical relationship,

$$E = \frac{I_o^2 \, t_r^2 \, (1 - e^{-t_1/t_r})^2}{50 \, \epsilon \epsilon_o \, L}$$

where: E = residual energy

I_o = input charge rate, coul/sec

t_r = relaxation time, sec

ϵ = dielectric constant of material under consideration

ϵ_o = permitivity in a vacuum in farads/cm

L = height of material in borehole (cm)

t_1 = load time, sec

The derivation of this relationship may be found in the Appendix.

Evaluation of Stored Electrical Energy in a Surface Mine Blasthole Loaded With Bulk ANFO

o Blasthole Loading Condition

Hole Size: Diameter –10 in.; Depth – 60 ft (filled to 30 ft)
or 914.4 cm

Loading rate: 500 lb/min

ANFO properties:

Conductivity $= 4 \times 10^{-14}$ mho/cm
Density $= 0.029$ lb/in.^{3}
Electrostatic charging rate $= I_o = 3 \times 10^{-4}$ coul/sec

Loading time $= 817$ lb/500 lb/min $= 98$ sec

o Energy Estimate for Hole in Conductive Rock Strata

Relaxation time $= t_r = \dfrac{(2.6)(8.85 \times 10^{-14})}{4 \times 10^{-14}} = 5.75$ sec

$$\text{Residual Energy} = \frac{I_o^2 \, t_r^2 \, (1 - e^{-t_1/t_r})^2}{50 \, \epsilon \epsilon_o \, L}$$

$$= \frac{(3 \times 10^{-4})^2 \, (5.75)^2 \, (1 - e^{-98/5.75})^2}{50 \, (2.6)(8.85 \times 10^{-14}) \, (914.4)}$$

$$E = 283 \text{ joules}$$

o Energy Estimate for Hole With a Nonconductive Plastic Liner

Relaxation time (based on conductivity of liner of 1×10^{-16})
$= 37$ minutes or 2200 sec

$$\text{Residual Energy} = \frac{(3 \times 10^{-4})^2 \, (2200)^2 \, (1-e^{-98/2200})^2}{50 \, (2.6) \, (8.85 \times 10^{-14}) \, (914.4)}$$

$$= 7.8 \times 10^4 \text{ Joules}$$

Nonconductive bore hole liners are often used for wet hole conditions. In this situation, the liner increases the relaxation time to ~ 37 minutes based on a typical liner conductivity of 1×10^{-16} mho/cm. The residual energy is higher than situations where there is no liner; however, in both cases there can be very high residual energies involved because of the non-conductive nature of ANFO. Since the time constant for charge dissipation through the plastic liner exceeds the time constant for charge redistribution in the explosives, the charge will diffuse to the walls forming an equipotential surface. The potential of this surface can be large and not

reach the maximum value until some time after the filling operation. If the bore hole is treated as a simple capacitor with the liner as the dielectric and the explosive as one plate and the ground as the other surface, a large potential can result. The voltage on the capacitor will be dependent on the breakdown voltage of the plastic liner. For plastic films, the dielectric strength can vary from 500 (polyethylene) to 10,000 volts/mil (polypropylene). Thus for the polyethylene liner 7 mils thick used in quarrying operations visited during this study, the breakdown voltage would be 3500 volts.

Potential Hazards

The residual energy that would be hazardous to operations with electroexplosive devices would be conditions that permit accumulation and discharge of energies in the range of 14 mJ which is a typical for the nonelectric detonators. Based on the energy relationship utilized, this could occur when the bore hole conductivity is less than 5.8×10^{-12} mho/cm.

For the case of nonconductive hole liners, the use of electric detonators is a potential problem and they are currently not utilized in conjunction with liners. The electrostatic sensitivity of explosive items other than electric detonators range from 0.0022 J for lead azide to 1.26 for non-NG dynamite. This indicates that initiation of these items under certain bore hole loading conditions, is also of concern. The analysis is inconclusive, but does indicate the need for additional study and possible regulations.

C. Mine Operator as a Source of Electrostatic Initiation Energy

Blasting operations in both surface and underground mining are labor intensive. The operators are interacting with the various blasting components from the time they are removed from the storage area until they are detonated. Typical situations where the operator can inadvertently apply electrostatic energy to the blasting components are illustrated in Figure IV-1.

An analysis to classify how important the operator is as an initiation source in a mine requires: (1) consideration of the amount of energy that a man can release in a spark discharge, and (2) a comparison of this energy to the sensitivity of blasting components in a system.

1. Operators as a Source of Stored Electrostatic Energy

A circuit commonly used to simulate an electrostatically charged human is a 500 pf capacitor charged to 20 KV potential discharged through a 5000 ohm series resistor into the detonator.[7] Another source quotes a value of 400 pf discharging through 100,000 ohms.[13] This equates to an energy of 100 millijoules. A third source treats the available energy as 15 mJ obtained from a human of 150 to 300 pf at 10 K volts. From the literature it can be concluded that the source of electrostatic energy from a human can be between 15 and 100 millijoules. The environment in a mine and the miner's dress can affect this energy.

A typical miner will not be dressed to reduce static accumulation via conductive shoes, etc., as are the operators producing the blasting components in manufacturing facilities. The miner will most likly be wearing

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PICKING UP EXPLOSIVES
AT MAGAZINE

CAP WIRES SLIDE THROUGH
HAND WHILE DROPPING
EXPLOSIVES INTO HOLE

CONTACTING CAP WIRES
WHILE LOADING ANFO

CONTACTING CAP WIRES
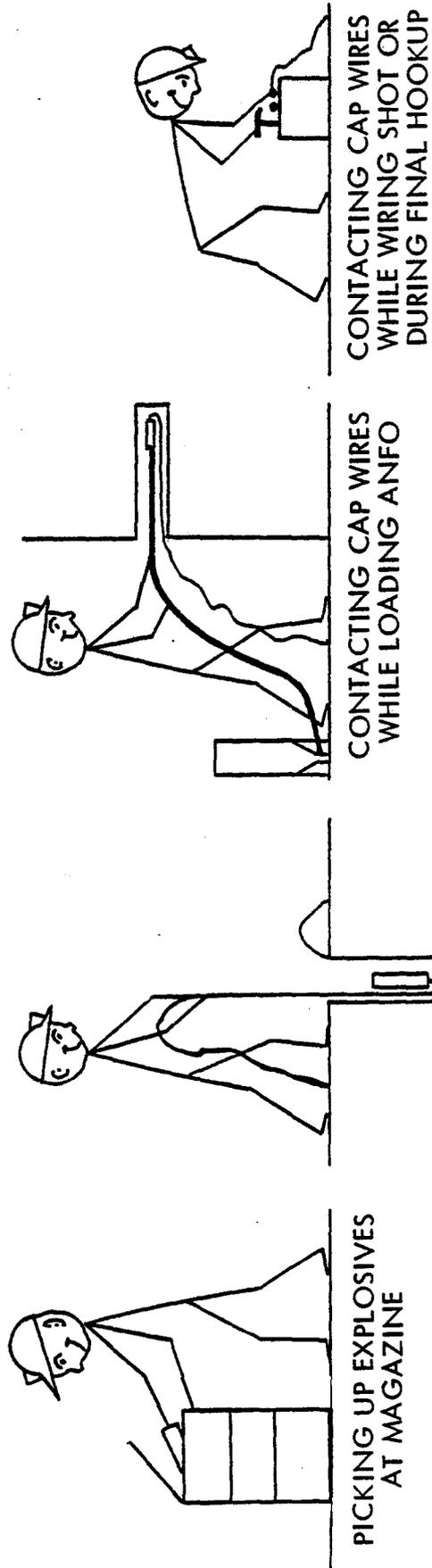WHILE WIRING SHOT OR
DURING FINAL HOOKUP

Figure IV-1. Mine Blasting Situations Where Operator is a Potential Source
of Initiation from Electrostatic Discharge

heavy work shoes or boots, coveralls, wet suits, etc. The mine environment can also vary widely in temperture and humidity conditions such as 80°F, 10% RH in the southwest to 45°F, 80% RH in the northeast. An evaluation of various parameters affecting the energy of charge accumulation on humans is treated in the ensuing paragraphs:

a. Capacitance

The electrical capacitance of the operator is formed primarily between the sole of his shoes and his feet with the shoe acting as a dielectric. One factor that influences the capacitance is the size of the shoe. The bottom surface of the foot and the ground constitute a parallel plate capacitor governed by the equation:

$$C = \frac{\epsilon \epsilon_o A}{d}$$

ϵ = Dielectric constant
ϵ_o = Permittivity in a vacuum
A = Plate Area
d = Plate Separation

A study conducted by the British Ministry of Aviation[14] showed, as expected, that shoe size increased the contact area and subsequently capacitance. It was determined that a person wearing a size 11 shoe had a capacitance ~ 25% greater than a person wearing a size 6 shoe. The capacitance values, Table IV-1, range from 160 pf to 200 pf.

TABLE IV-1. EFFECT OF SIZE OF FOOTWEAR ON CAPACITANCE
OF PERSON STANDING ON LINOLEUM FLOOR[14]

Size of Shoe	Capacitance, $\mu\mu$ F
6	160
7	175
8	180
9	190
10	200
11	200

The type of footwear utilized in the mine is not a controlled factor. The influence of the type of footwear and surface was also considered by The Ministry of Aviation[14]. For our purposes, the data in Table IV-2 for leather and rubber are most applicable and the capacitances observed are in the range cited above.

b. Available Energy

Electrostatic charge generation can occur by the rubbing action of different materials and occurs, for example, by the simple action of rising from a chair or by a miner rising after having been seated on

TABLE IV-2

ELECTRICAL CAPACITANCES ($\mu\mu$F) OF TWO SUBJECTS[14]
WEARING DIFFERENT TYPES OF FOOTWEAR ON
VARIOUS INSULATING AND CONDUCTIVE FLOORS

Type of Flooring	Subject No. 1 Wearing Shoes, Size 8, of:				Subject No. 2 Wearing Shoes, Size 9, of:		
	Leather	Rubber	Antistatic Rubber	Conductive Rubber	Leather	Rubber	Antistatic Rubber
Linoleum	140–200	150	155	160	155–180	170	165
Linoleum Near Earthed Cupboard	–	160	170	200	225	–	200
Dry Asphalt	–	150	170	160	–	150	180
Concrete	–	140	260	290	–	205	310
Antistatic Rubber	400	200	420	405	340	155	425
Conductive Rubber	250–750	160–300	700–900	1200–1450	225	190–290	700
Copper Sheet	400–640	210	1340	1060–1400	215–640	255	1110
Galvanized Iron Grid	160	150	850	400–720	150–1500	170	880

equipment. This takes place for a variety of natural fibers, but is usually more pronounced for some synthetic fibers having high resistivities. Figure IV-2[17] shows the effects of surface resistivity on voltages and hence energy for various fabrics with energies ranging from the μj range up to about 12 mj. (The given body capacitance was held constant at 220 pf and the chair in this case was covered with a PVC coated cotton seat cover.) While the higher calculated energies ($\sim$ 12 mj) would appear to approach the maximum no fire energy level for detonators (previously shown to be about 16 mj in the pin-pin mode), it should be remembered that the 16 mj value was obtained in a test set-up containing no series discharge resistor.

The surface resistivity of these materials is also a function of the relative humidity with lower resistivities for higher humidities; this is shown in Figure IV-3a[15] where resistivities drop roughly four to five orders of magnitude for a relative humidity increase from 15 to 80%. The figure also shows maximum attainable voltages decreasing several orders of magnitude over the same range of humidity increase.

Another typical way a human accumulates electrostatic charges is by walking. Table IV-3[14] shows typical capacitances, potentials, leakage resistances and electrostatic energies for insulated and conductive footwear on insulated and conductive flooring. The stored energy values observed for the various combinations range from 3×10^{-9} to 10^{-4} joules.

c. Body Resistance

Resistance of the human body can vary depending on contact area and pressure. This in turn will alter the characteristics of the discharge. Typical discharge resistance values for different hand and finger pressures as well as needle point contact, are shown in Table IV-4.[9] The lowest values were obtained for hand area (20 cm^2) contacts, and they ranged from 340 to 590 ohms for five different subjects. Relatively high resistances up to 6.7×10^5 ohms were observed in needle point contacts.

Table IV-5[16] shows the insulation resistance, potentials and charge relaxation times for different sole materials worn by humans. Conductive rubber soles provide an insulation resistance of 10^5 to 10^6 ohms; whereas, ordinary rubber soles have a resistance of 10^{13} ohms and the corresponding RC decay times are about .05 and 200 sec while the corresponding energies would be 1.6×10^{-8} J and 1.6×10^{-2} J. These data indicate that a miner wearing ordinary rubber boots could retain electrical charges for several hours providing he did not have hand, finger or other body contact with ground.

Table IV-6[14] shows fingertip resistances at different voltage levels for several subjects; the measurements were obtained with current flowing continuously. It is to be noted that body resistance decreased significantly as the potential was increased from 12 to 60 volts - for one subject (No. 3) it decreased to about 6% its initial value. Thus, human body resistance, for a given set of circumstances, is difficult to define.
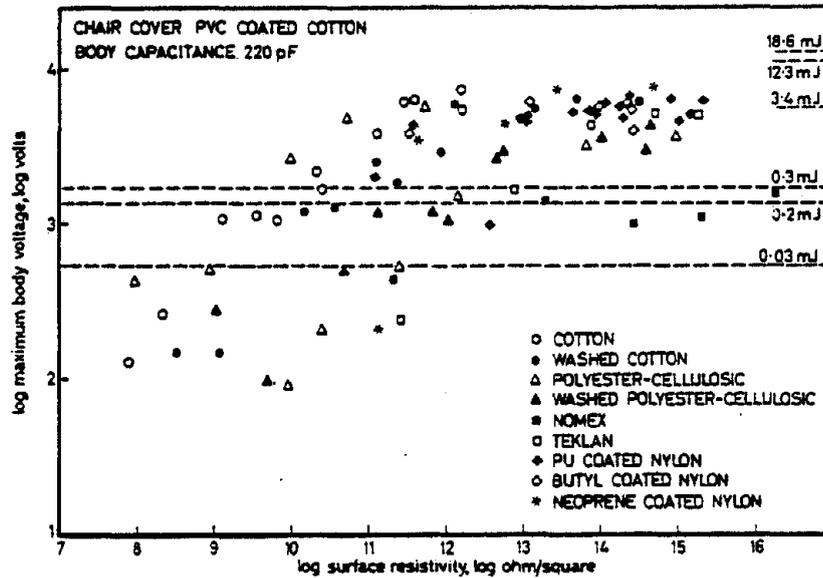
82

Figure IV-2. Log Maximum Body Voltage vs. Log Surface Resistivity.
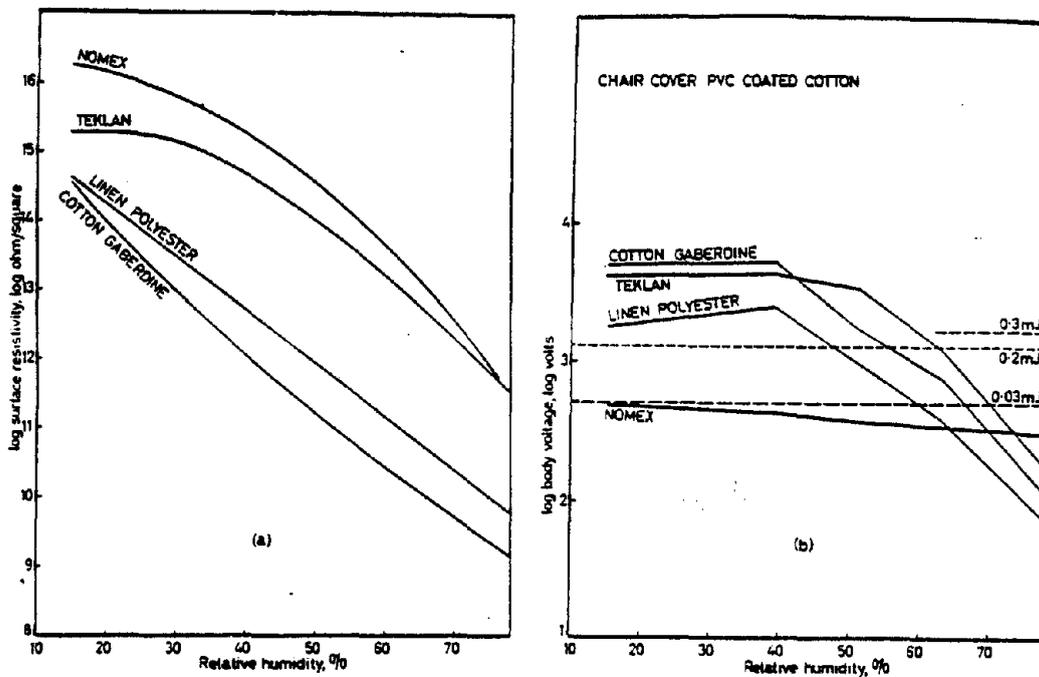PVC-Coated Cotton Seat Cover. Body Capacitance 220pF.[17]

Figure IV-3. (a) Log Surface Resistivity vs. Relative Humidity, and (b) Log Body
Voltage vs. Relative Humidity, Using a PVC-Coated Cotton Seat Cover[15]

TABLE IV-3. CAPACITANCES, POTENTIALS, AND ENERGIES DUE TO FOOT[14]
MOVEMENTS OF PERSONS IN INSULATING AND CONDUCTIVE
FOOTWEAR ON INSULATING AND CONDUCTIVE FLOORING

Footwear	Floor	Capacitance, $C, \mu\mu F$	Potential, V, Volts on Foot Sliding	Leakage Resistance, R, Megohms	Energy, $1/2\ CV^3$, ergs
Composition	Linoleum	150	1,000	1,000	750
Composition	Conductive Rubber	200	1,000	1,000	1,000
A/S Rubber	A/S Rubber	425	100	500 to 1,000	21
Conductive Rubber	Linoleum	160	200	1,000	32
Conductive Rubber	A/S Rubber	405	60	1,000	7
Conductive Rubber	Dry Concrete	290	20	6 to 12	0.6
Conductive Rubber	Conductive Rubber	1,450	2 to 10	1	0.03 to 0.7

TABLE IV-4. DYNAMIC RESISTANCES AT THE HAND AND FINGERTIPS OF
VARIOUS SUBJECTS, IN OHMS[14]

Subject No.	Hand, Area 20 cm^2	Fingertip, Area 0.8 cm^2	Needle Point Moderate Contact	Needle Point Light Contact
1	430	1900	4.5×10^4	6.7×10^5
2	550	1400	7.4×10^4	6.5×10^5
3	590	1500	6.5×10^4	2.2×10^5
4	340	4400	2.1×10^4	4.4×10^5
5	490	1900	5.2×10^4	0.8×10^5

TABLE IV-5. RESISTANCE, DECAY TIME AND POTENTIAL FOR
VARIOUS SHOE MATERIALS[14]

Sole	Resistance (ohms)	Decay Time (sec)	Potential Volts
Conductive Rubber	1×10^5	-	1
Conductive Rubber	1×10^6	0.05	8
Leather	1.5×10^7	0.1	60
Leather	2×10^8	0.17	200
PVC	3×10^9	0.7	2500
Rubber	1×10^{13}	200	4000 to 8000

TABLE IV-6. FINGERTIP RESISTANCES OF A NUMBER OF SUBJECTS AT
DIFFERENT[14] APPLIED POTENTIALS
(CONTINUOUS-CURRENT METHOD)

Subject No.	Resistance (X 1000 Ω) Measured With Applied Potential				
	12 V	24 V	36 V	48 V	60 V
1	240	225	180	96	60
2	400	340	–	320	210
3	1200	400	–	150	68
4	400	270	–	160	–
5	150	80	45	–	–
6	150	27	–	–	–
7	300	240	90	48	–

2. Hazards of Electrostatic Discharges From Humans

At this point, we wish to comment briefly on possible hazards to detonators and blasting materials with regard to static discharges from humans. First, these discharges were characterized by Tucker[17] and reported in the annuals of the New York Academy of Sciences. Two curves, that are reproduced in Figures IV-4 and -5, were presented and they show current as a function of time for a discharge off a bare finger and off a metal tool held by a subject. Though peak currents are quite high, they concluded that body voltages of at least 71 Kv would be required to initiate PETN. This explosive is most often the explosive used in detonating cords or used as a base charge in detonators and is used as a constituent in some booster materials. Unpublished data from Bureau of Mines tests as well as results of Hercules tests show that PETN is more sensitive to initiation from electrostatic sparks than NG sensitized explosives and certainly more sensitive than blasting agents such as ANFO. Thus, static discharges from humans do not present a significant hazard insofar as most explosives and blasting agents are concerned.

The initiation of electric detonators is, very possibly a different matter. For sake of demonstration, if the static charge energy on a human is 15 mj, and using the static sensitivity data in Table III-3 which show TIL values as low as 16 mj, then the probability of initiation would be 2×10^{-2} which is unacceptably high.

D. Radio Frequency and Induced Current Hazards

The use of blasting agents for mining purposes has a long history with a typical consumption of 2.5 billion pounds in one year (1975)[18]. Through the end of World War II, improvements were made in blasting safety by replacing treacherous primaries with more stable materials such as the replacement of lead azide with secondary explosives as used in EBW (explosing bridgewire) systems. However most commercial detonators still utilize a primary explosive.

Parallel with this effort there has been a trend toward increasing energy thresholds in the initiator systems. Since the 1940's, a 0.25 amp no-fire level was common for standard blasting caps. This is still the industry standard in 1982.

The military continued to develop less sensitive caps due to the increased intensity of radar fields resulting in first a 1 amp no-fire requirement and later the present 1 amp, 1 watt no-fire requirement. The open wire pair and single wire, ground return circuits were replaced with low impedance, twisted, shielded pairs isolated from ground which further reduces the susceptibility to inadvertent firing.

The sensitivity to electrostatics has not been ignored either. The design criterion is generally that they must pass a 25 KV discharge from a 500 pf capacitor through 5000 ohm resistance in both the pin-to-pin and pin-to-case mode or 0.3125 J.
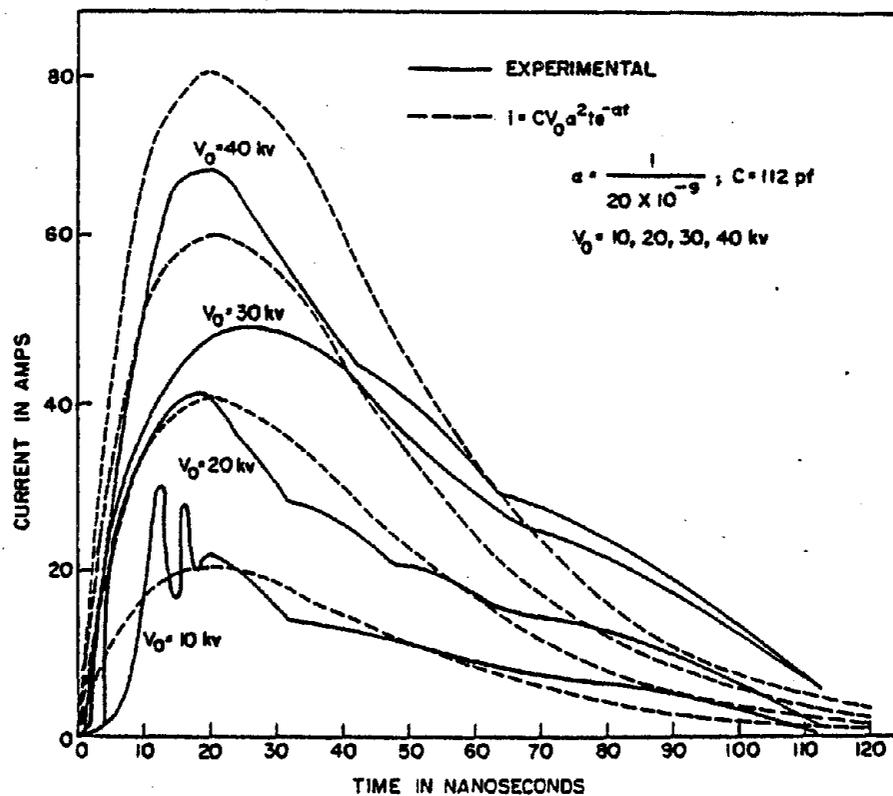
Figure IV-4. Typical Spark Currents Observed for Initial Body Voltages of 10, 20, 30, and 40 kv; Sparks Drawn from the Subject's Finger[17]
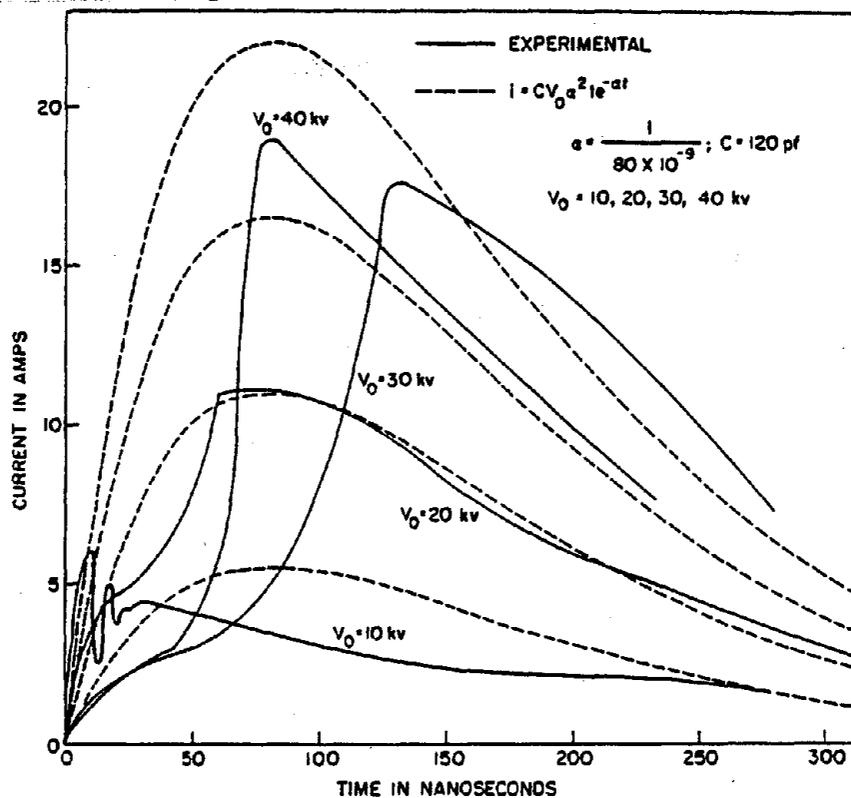

Figure IV-5. Typical Spark Current Observed for Initial Body Voltages of 10, 20, 30, and 40 kv; Sparks Drawn from a Tool Held in the Subject's Hand[17]

The electric detonators presently used in mines do not have this reduced current sensitivity. They are normally 0.25 amp no-fire with a sensitivity of 4 mJ to 500 mJ being observed. The presence of electromagnetic radiation in the radio frequency range has increased over the years. The safety guide for use of electric blasting caps in this environment is supplied by the Institute of Makers of Explosives (IME). This criterion is based on a cap sensitivity of 40 mw no-fire level.

According to IME Publication No. 20, the few cases of caps being fired accidentally by Rf pickup on the wires would not have occurred if the table of distances had been followed. No information has been developed in this study to require a change in these guidelines. However, surveillance of Rf source at mine sites should be a continuing effort as new Rf producing devices are developed.

The secondary effect of induced currents on operating equipment is not covered by these guidelines and should be addressed in future publications.

E.   Lightning

The following is a summary of an evaluation performed by Southwest Research Institute (SRI)[20]. This study of lightning warning technology indicates that four alternatives may be pursued in implementing an effective electrical storm safety program. These are: (1) to encourage the use of common sense safety practices, (2) to use one of the lightning warning systems tested, (3) to develop a system which meets the unique needs of the mining industry, and (4) to develop a device independent safety specification in terms of electric field strengths, number of lightning events, etc. In the following discussion these alternatives are considered.

1.   Common Sense Safety Practices

One approach in reducing exposure to lightning hazards is to explore a knowledge of the local climatology. At Kennedy Space Center, for example, peak summertime thunderstorm activity occurs around 1400-1500 local time. To ensure minimum downtime and reduce the possibility of accidental predetonation, blasting schedules can be adjusted accordingly.

Also a broadcast receiver can be tuned to a frequency which is not used by a transmitting station, and one can listen for static crashes to estimate if lightning is occurring in the area. This information coupled with visual observations of cloud formations has been used extensively to forecast potentially hazardous lightning conditions.

A major concern in the common sense approach is the lack of exact knowledge about the location of lightning events and the consequent use of subjective judgement. While the common sense approach is extremely appealing from a cost consideration, the incurred risk may be substantial. As a tradeoff among the various safety alternatives, this approach represents minimum cost and maximum risk.

2. Lightning Warning System Evaluation

An alternative approach to preventing lightning induced predetonation is the use of one of the devices tested. SRI stated that an effective automatic lighting warning system (LWS) has the following advantages over relying on weather observations:

 a. On overcast days warning is provided when storm system buildup cannot be seen from the ground.

 b. Operations can continue until evacuation is initiated based on storms detected within a predetermined hazardous range.

 c. Operations can resume when lightning passes beyond the hazardous range.

 d. Warning is provided when the entire crew is engaged in operations.

An automatic lightning warning system which provides the above capability is required to ensure maximum safety in blasting operations. Of the devices evaluated in the SRI study, three evidenced unacceptable performance. The flash counter proved to be unacceptable because of its noise susceptibility and it required continual recalibration of the alarm threshold. The corona point sensor also evidenced a high degree of noise susceptibility and a vulnerability to direct lightning strikes. The false alarm rate observed in the performance of the azimuth/range locator was determined to be intolerable.

Acceptable techniques among the devices tested were the field mill, radioactive probe and the triangulation locator. The field mill device (which measures electric field strength, flash rate and ambient background noise level) exhibited failure-to-alarm rates which were of concern; however, as a cost tradeoff, this device appeared to be cost effective as an off-the-shelf single point sensor in comparison to the triangulation locator. Closely comparable in performance to the field mill was the radioactive probe which measures potential gradient and changes in the gradient. This system also had a relatively high failure-to-alarm rate and exhibited probe saturation effects which may require a development effort to correct. The overall best performance was obtained from the triangulation system; however, this system is ineffective against overhead lightning hazard buildup since it reports only cloud-to-ground lightning flashes.

Based upon the results of this evaluation study, the system presently available commercially which technicaly best satisfies mining safety requirements is the triangulation location system.

Table IV-7 compares the performance of each LWS under conditions of frontal, convective, and orographic storms. The units generally provide 30 $\pm$ 10 minute warning time prior to storm location at 10 nautical mile range with the exception of the azimuth/range locator and the corona point device. The azimuth/range locator tends to warn one hour or more in advance

TABLE IV-7 - LIGHTNING WARNING SYSTEM COMPARISON BY WARNING PARAMETER

Warning Parameter	Radioactive Probe			Field Mill Device			Corona Point Device			Flash Counter			Triangulator			Az/Range Locator		
	Front*	Conv.*	Mount*	Front	Conv.	Mount	Front	Conv.	Mount	Front	Conv.	Mount	Front	Conv.	Mount	Front	Conv.	Mount
Mean Warning Time in Minutes	33	28	21	40	27	50	-20	-15	-20	35	32	-4	21	21	20	121	39	101
False Alarm Incidence Percentage	9	0	0	18	0	0	27	0	0	0	0	0	0	0	0	73	0	82
Failure to Alarm Incidence Percentage	9	10	27	9	15	20	55	80	80	9	55	64	9	6	9	0	5	0
Incidence of Valid Alarm Percentage	91	100	100	82	100	100	73	100	100	100	100	100	100	100	100	27	100	18
Mean Time to Clear After Hazard Passed in Minutes	15	-8	33	22	9	67	-19	22	40	44	-18	23	5	-4	0	109	17	117

*Front - Frontal
Conv. - KSC
Mount - Mountain

NOTE: The instruments evaluated were intended to be representative of their respective classes; however, specific findings apply only to the units tested.

of 10 nmi range. This is apparently due to the variability of received signal strength and the assumed mean amplitude at the given ranges. As a result, range is underestimated with apparent 10 nmi range locations indicated one hour before the storm reaches this radius.

The results of the LWS evaluation are for specific equipment items and are not necessarily representative of all similar devices. Each manufacturer's device should be evaluated on its own merits.

F.   Review of Methods for Electrical Hazard Minimization

   1.   Ground Fault Systems

Stray current from mine electrical equipment is of concern in mine operations from the standpoint of both the exposure of operators to electrical shock hazard and premature activation of electric detonators. The survey identified mine accidents where stray current from electrical equipment was a probable cause for the accident. Ground fault devices are designed to provide additional protection against line-to-ground shock hazard.

During the mine visits, it was found that not all mines have ground fault systems. Of those that did, each had experienced trips because of ground fault incidents. Some mines had trips of the systems weekly while others have had one a year. There is no doubt with this history of occurrences that the ground fault system is needed in mines to provide practical protection against electrical accidents. If this type of system is not present, miners are exposed to potential hazards when working around any electrical equipment as well as the hazards of stray current running in the grounding system of the mine during blasting operations.

A study by Biegelmeier entitled, "The Electrical Impedance of the Human Body at 50 Hz Alternating Current"[21] discusses the potential current through a human in contact with an electrical source and methods of ground fault protection. One surprising result was the immediate onset of high current levels at voltages just over 100 V for contact surface areas such as those present when electrical accidents occur while handling tools. It is apparent from the data that the body impedance drops sharply as the voltage is increased. Thus, even when dry, the skin forms only relatively slight protection against fatal electrical accidents. Furthermore, the moistness of the hand, if it has been wetted with normal tap water or rain water, does not play a decisive role. Only with exception- ally conductive solutions does the body impedance drop significantly.

Biegelmeier demonstrated that ground fault interrupters could be properly adjusted and provide complete protection for a human being who was in a situation where he could not let go of the source that bound him to it. His results showed that, at about 200 volts potential, body currents exceeding 100 ma could be observed for contact surface areas prevailing when gripping metal tools. In fact with contact surface areas of only 1 cm^2, currents exceeding 30 ma were observed. Under similar conditions, the hazard is lessened if the current is entirely through the shoes to ground. However in planning preventative measures, worst case parameters should be considered.

A standard tripping level is not specified in the National Electrical Code for personnel protection; however, typical values are 0.03, 0.3, 0.5, and 1 amp according to Biegelmeier. Based on his studies, a value of 0.1 amp as a standard, would provide maximum protection.

The maximum no-fire level for electric detonators is 250 ma; therefore, to provide stray current protection for these devices, a 0.1 amp standard is also advisable.

2. Semi-Conductive Hose for Pneumatic Loading of ANFO

The safety aspects of the use of semi-conducting hose in pneumatic loading operations were amply demonstrated by duPont[9] and by Gates Rubber Company.[11] Both studies demonstrated that, properly used, these hoses reduced electrostatic hazards significantly.

One concern is that, while new hose may have a given resistance, its resistance may change over a period of usage or time. Several samples of new and used hoses were obtained during the mine visits. The time element for the used hoses was unknown. Results of resistance measurements on these hoses are given in Table IV-8. The resistivity of the six samples of new hoses varied from 1.93×10^6 to 3.95×10^6 ohms/cm. For the two cases (samples 3 and 4) where measurements from new and used hoses were available, the resistivity of the used hoses increased by 9% and 4%, respectively. Thus, used hoses can lose some of their semi-conductive properties presenting another factor that should be taken into consideration when surveying the electrostatic hazard in pneumatic loading operations.

TABLE IV-8. COMPARISON OF RESISTIVITY OF VARIOUS SEMICONDUCTIVE HOSE SAMPLES OBTAINED DURING MINE VISITS

Sample	Length in.	Hose Size in (ID)	Resistvity, ohm/cm	
			New	Used
1	6	5/8	2.36×10^6	—
2	6	1	1.93×10^6	—
3	3	3/4	3.95×10^6	4.32×10^6
4	6	3/4	2.00×10^6	2.09×10^6
5	6	3/4	—	2.06×10^6
6	6	3/4	2.06×10^4	—
7	6	3/4	1.98×10^3	—

NOTE: The resistivity was calculated by dividing the applied test voltage (2800 V) by the current measured on a hose sample between two metal clamps. This result was then divided by the length of the hose between the clamps.

3. Stray Current Monitors

Stray current monitoring devices have been developed for mine use. Typical devices are (1) the VAO blasting meter by Bitronics, (2) duPont Blasters Multimeter Model 101, and (3) GM 100 Ground Fault Monitor by Austin Powder Company. Comments concerning these meters are as follows:

a. VAO (Volt Amperes Ohms) Blasting Meter

This is a multi-purpose meter covering the range of values applicable to mining operations as follows:

Ohms   0, 8, 16, 32, 64, 112, 192, INF

Volts 0.5, 1.0, 1.5, 2.0, 2.5

Amps   0, 0.5 0.10, 0.15, 0.20, 0.25, 0.30

                                 red zone

This device is not for continuous monitoring.

The instrumentation booklet for the device refers to pin jacks marked A-V and V. No such markings were found on the meter available to Hercules. Also Page 6 of the instructions indicates that in the case of conductive ores, various metal conductors such as rails, pipes, and ducts should be connected to the ore body by means of heavy copper wire attached to low resistance grounds driven into the ore body. It appears that this practice could introduce electrical currents.

b. DuPont Blasters Multimeter Model 101

This meter is manufactured for duPont by Simpson Electric Company for use in mine applications. Like the VAO blasting meter, it does not offer continuous monitoring or alarm capability.

c. Austin Model GM 100 Ground Fault Monitor

This meter is furnished with a built-in alarm system that is activated when the input current approximates 50 milliamperes DC and/or 50 milliampers RMS AC. A meter was not available for this study, therefore additional comments are not provided.

4. Lightning Warning Devices

The event that is the highest contributor to mine accidents, as to electrical/electrostatic causes, is lightning. Warning devices were evaluated by SWI in another study[20] and will not be discussed in detail. Several types of devices are available, according to the referenced report,

such as radioactive probes, field mill devices, Corona point devices, flash counters, triangulation locations and azimuth/range locators. The objective of these devices is to provide early warning of lightning so that the mine operators can safely stop blasting operations and remove personnel to a safe area until the potential danger passes.

Each of the devices evaluated by SRI exhibited advantages and disadvantages which are discussed in the report. The data presented can be useful for evaluation of particular operations.

5. Nonelectric Blasting Systems

   a. NONEL System

   The NONEL system, manfactured by Nitro Nobel,[22] is a closed system with each drill hole served by a separate NONEL unit. It is delivered complete to the blasting site and consists of a moisture proof assembly of a suitable length of NONEL tube, a detonator and a transmitter cap with connecting block.

   The safety features of the system, as reported by the company, are as follows:

   (1) Immune to electricity – the immunity applies to all types of electricity – stray currents from fixed installation and safety-lamp batteries, earth potentials in the immediate proximity of power lines, current leakage in conductive ore bodies and static electricity. The immunity of NONEL to electricity is absolute and does not need to be measured or checked.

   (2) Immune to thundery weather – charging can take place without interruption throughout the working day even in climates where recurrent stoppages due to threats from nearby atmospheric disturbances otherwise have to be taken into account.

   (3) Immune to radio and radar – the risk of inadvertent initiation via radio waves is generally considered to be extremely slight, but it is nevertheless one of those hazards which are difficult to foresee and thus difficult to protect against. NONEL is totally unaffected by radio and consequently it is also completely immune to marine radar and radio signals from low-flying aircraft.

   This, in turn, means that radio techniques can now begin to be adapted for underground use and that the benefits of communication radio can be enjoyed also on blasting sites.

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(4) Immune to jolts and blows - tests have shown that the NONEL tube does not react even to very hard blows. This ensures great safety in transport and handling, but particularly in the course of actual charging.

(5) Fireproof - the plastic hose itself burns, but the explosive cannot be made to detonate by fire, not even by an open flame introduced right into the tube.

(6) Safe delay - with detonating cord, there has always been a certain risk that blast stone, gas, etc., would tear off the cord before all holes had been reached by impulses.

With NONEL, the impulse goes straight into the hole, where the delay element in the detonator takes over the time keeping while the round fires over.

When it comes to the influence of the delay element on dispersion and thus on fragmentation and ground vibrations, the NONEL system does not differ from the electric detonator, since the detonators themselves are of nearly identical construction.

The system appears to correct many of the problems associated with EB caps in areas where stray currents, RF energy or lightning exists. However, electrostatic incidents were reported by customers using the system as initially fielded. The material inside the system was semi-conductive and would conduct a charge to the cap and discharge to the case through the primer material. The design of the cap was changed to incorporate a static bleed hole in the tubing to allow the charge to dissipate at a preferred spark gap point.

The cap at the end of each line is a modified electric detonator. The modification involves removing the bridgewire and does not effect the explosive material in the cup. Therefore, the sensitivity of the NONEL detonator remains unchanged from the original electric detonator which is governed by the primary explosive involved.

This system is reported to withstand 60,000 volts applied to the NONEL line using the static bleed hole without an arcover to the cap case. These results indicate that the NONEL system should be immune to electrostatic discharges in this voltage range, however, no test data were available to Hercules so that comment could be made on claim that it is safe to use in lightning storms or "thundery weather."

b. Hercudet® System

System Description – Hercudet is a nonelectric delay blasting cap system for the initiation of commercial explosives. The principle of operation is to fill a firing circuit (a circuit consisting of the Hercudet caps and interconnecting tubing) with a detonable gas mixture. The caps are then initiated by igniting the gas mixture from a firing chamber. Since the caps are nonelectric, and the system is inert during installation of the firing circuit, the Hercudet system eliminates hazards generally associated with the use of electrical blasting caps.

The Hercudet system is composed of three primary subsystems: (1) a Bottle Box Assembly, (2) a Blasting Machine Assembly, and (3) the Firing Circuit. A simplified diagram of the system is shown in Figure IV-6. The Bottle Box Assembly's function is to supply components of the gas mixture and nitrogen for purging and pilot valve operation. The function of the Blasting Machine is to meter the gas mixture and pump it through the firing circuit, and to fire the mixture. The Firing Circuit consists of the blasting caps, and the interconnecting tubing and fittings for piping the gas mixture to each cap. In the paragraphs that follow, each of these subsystems will be discussed in greater detail.

The Bottle Box contains the supply for the three gases (fuel; oxidizer – O_2; and inert – N_2) which are used in the blasting machine. As each of the three gas supplies is similar, only one will be described.

Gas is supplied from a pressure cylinder. A fill line with check valve is provided for recharging of the gas cylinders. The cylinder can be valved off by a shut-off valve on the cylinder. Integral to this valve is a safety relief which provides protection in the event of overpressurization (e.g., high temperature in a fire). Gas supply remaining is monitored via the pressure gauge. A primary gas regulator reduces the pressure to around 175 psig for delivery to the Blasting Machine in a flexible hose with quick-disconnect.

System Operation – The following system operation describes how a firing circuit is completed and tested, how the circuit is charged and checked, and finally fired. It is based on observation of a quarry shot.

(1) Individual caps are lowered in holes and holes loaded with explosive and stemming.

(2) Individual caps are circuit tested with the hand pump tester. Any "bad" caps are marked for replacement.

(3) Connections are made from cap-to-cap to form the circuit for the several cap-lines.
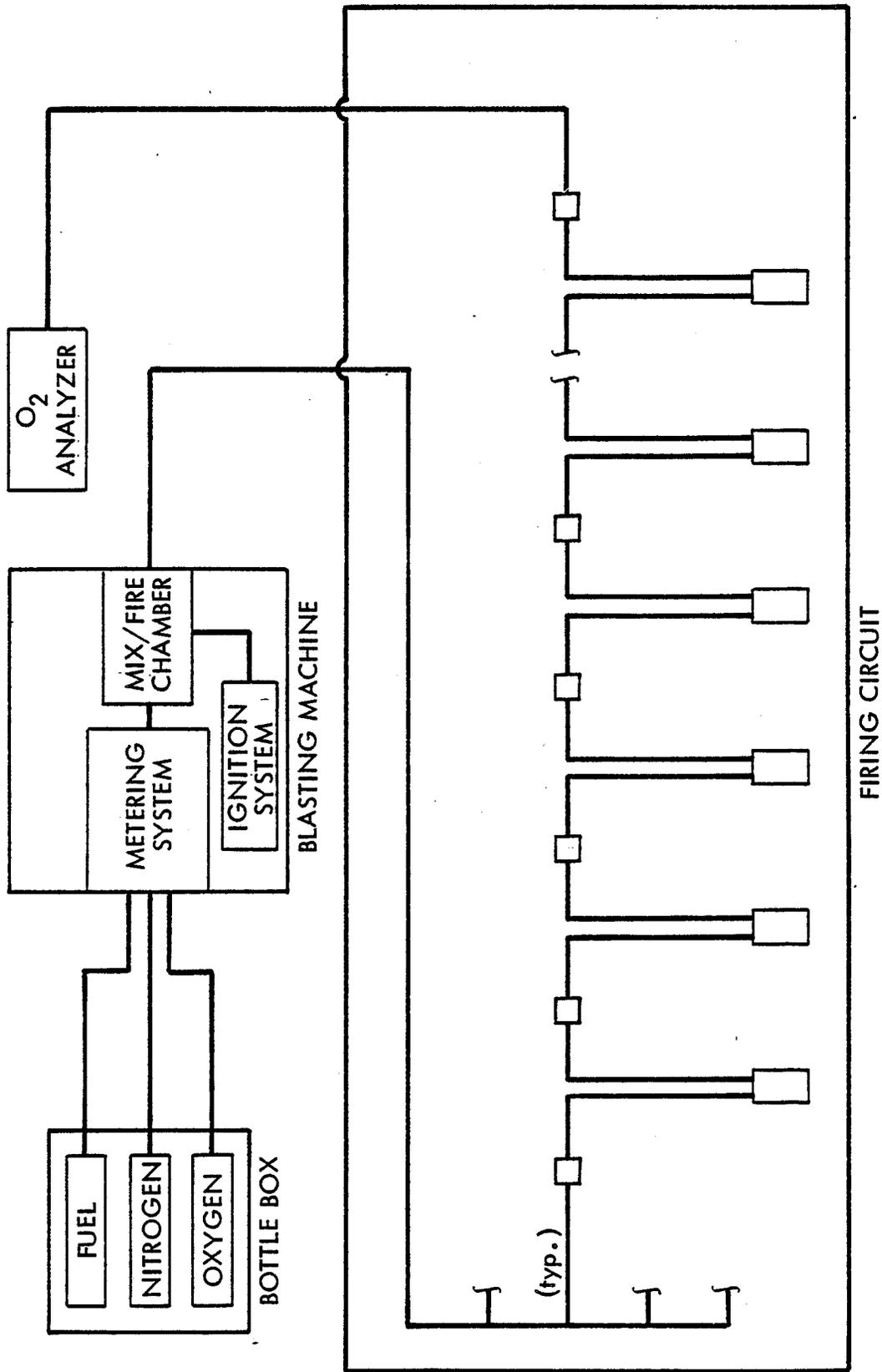
(4) Each cap-line is circuit tested.

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Figure IV-6. Hercudet System Diagram

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(5) The cap-lines are connected to the trunk line and the trunk line and tail lines run from the blast area to the firing point.

(6) The bottle box is connected to the blasting machine and the blasting machine is connected to the trunk line.

(7) The blasting machine is actuated to charge the firing circuit until all tail lines indicate approximately 50% O_2 on the O_2 analyzer.

(8) The shot is fired.

Safety Aspects of the Design - The Hercudet nonelectric delay blasting system offers a method for initiating commercial explosives. The blasting cap is nonelectric, with plastic tubing replacing wires. A gaseous mixture is introduced to a firing network of small plastic tubing after all preparations for firing are complete. Until the mixture is introduced, the network is inert except for the cap itself. After charging, the gas mixture is ignited from a specially designed blasting machine, and the reaction travels through the firing circuit at a rate of 8,000 ft/sec to initiate the caps.

Some of the safety features are:

0 Cannot be initiated by radio frequency energy, stray current, or static electricity.

0 The only nonelectric initiation system with circuit test capability.

0 The gas reaction within the Hercudet tubing cannot initiate or dead-press main explosives charge.

Analysis of the System - When the system design was reviewed in this study, several items of concern were as follows:

During the mine visits, an abnormal use of a Hercudet system was observed. In this case, the operators installed the system normally. However, in the process of evaluating the vapor concentration in the tubing, they were found to be checking the concentration at the blast hole rather than at the gas charging station. This situation placed the operators back at the bore hole where inadvertent ignition could have produced a serious accident.

The caps are designed to be initiated by the stimuli from a vapor reaction. Except for the mode of initiation, Hercudet caps are very similar to conventional nonelectric blasting caps. Since incidents (see Table I-1, Page 12) have been reported with initiation of conventionally fused nonelectric caps, it is important to remember that the Hercudet system is only as immune to electrostatic hazards as the caps attached to the end of the tubing.

6. Electrostatic Discharge Protection for Electric Detonators

One concern with electro explosive devices is inadvertent activation due to electrostatic discharge between the input lead wires and the metallic case or closure. To protect against this mode of firing, many techniques have been proposed. A review of the resistive techniques was provided by Rosenthal and Leopold.[23] The following section is from their discussion of protection by resistive padding.

Protection by resistive padding – recognizing that an equivalent series resistance is always available, protection can be accomplished by installing a resistance (R_p) across the cap. The resistance can be linear and can be externally applied or built (intrinsic) into the hardware of the system. The lead wires or pins can be painted with a conducting coating so as to make connection to the outside case. This continuous coating over the back end of the device is the padding resistor. This procedure has been successfully employed for hermetically sealed devices. Another variation of this technique based on intrinsic resistive padding employs a header made of conducting plastic. The input leads are molded into this plastic, and when the outside metallic case is crimped over the header, the composite assembly is a nearly perfectly shielded enclosure. The idea that the cap becomes a near equipotential shield precludes any internal electric field formation. The imperfectness of the shield is a consequence of requiring two ports to feed the bridgewire energy.

Another successful esoteric approach employs a nonlinear resistor material such as silicon carbide or zinc oxide as the dissipative material. These ceramic-like materials can be employed as resistor assemblies or can be integrally made a part of the header system. Because these materials exhibit small or negligible conductivity at normal voltages, they do not interfere with the regular firing modes. However, they conduct readily at the higher voltages and serve as the equivalent of a breakdown diode (i.e., Zener), with higher joule ratings.

Resistance padding protects by two means – simple voltage division limits the voltage that can appear across the cap. For example, if the R_p built into the device is 200 ohms/pin to case, then for a 5000 ohms resistor (R_s) only 4 percent of the available voltage will ever be transferred to the internal device gap. It is important to note that the typical bridgewire header is less than 10 ohms (most often in the range of 1-5 ohms) so that the pin to pin shunting resistance does divert a small fraction of the firing energy. This penalty cannot be avoided but should be considered negligible as a practical consideration. The reduced voltage can insure that there will never be an internal discharge. There is a secondary protection mechanism offered by resistive padding. For a given spark gap, there is a time of formation. By insuring a rapid discharge, the breakdown can be cheated and prevented from forming.

These or similar devices are available in blasting caps used in mines and appear to be beneficial.

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7. DuPont Pneumatic Loading Calculator

DuPont has devised a slide rule type instrument for quick field evaluations of electrostatic hazards associated with pneumatic loading. In essence, the calculator can provide the operators with the maximum loading rate that is compatible with a previously measured ground return resistance. Other parameters considered are the absolute humidity and legwire lengths of the detonators. The first step is to measure and record the resistance between a metal rod, inserted all the way into a borehole that is loaded with ANFO, and the end of the loading hose. Another preliminary measurement taken is the relative humidity (RH) and temperature (T). The absolute humidity (AH) is then located from RH and T scales on the slide rule. At this point the maximum loading rate to be used and length of legwires (indication of the depth of borehole and hence its capacitance) are used in the slide rule calculation to find a maximum allowable resistance for these conditions. Then so long as the slide rule resistance is greater than the measured resistance, safe loading can proceed.

The slide rule appears to be a useful tool. While the use of semi-conductive hose generally solves most electrostatic problems in loading operations, the slide rule method can forewarn an operator if semi-conductive hose resistances (and rock strata resistances) have increased with time or blasting site. This, together with other precautions such as are outlined by Litchfield[8] (e.g., making sure that the blaster grounds himself before handling the detonator wires, keeping the detonator leads shunted but not grounded to guard against stray currents, etc.) should reduce the probability of accidental initiation of electric blasting caps to acceptable levels.

SUMMARY, CONCLUSIONS AND RECOMMENDATIONS

SUMMARY

The objective of this study was to make a critical evaluation of the magnitude and extent of electrostatic hazards and to assess the adequacy of techniques, materials, and equipment for detecting and minimizing these hazards.

The evaluation of the magnitude and extent of the hazard was accomplished by the accident survey. The key word for the survey was "explosives" or "explosions." A total of 893 incidents or accidents were identified for the period 1979-1980. The review was extended to include the period between 1976 and 1978 which yielded another 49 reports for a total of 942 incidents/accidents.

The review of the data separated the incidents into 572 coal mine incidents, many of which involved minor methane ignitions of no consequence, and many that were nonapplicable. The final result was 29 accidents or incidents in the U. S. coal and metal/nonmetal mining industry for the five year period from 1976 to 1980 as the reported magnitude of the problem.

This level of incidents seems small compared to the man hours involved in the mining industry and the quantity of explosives used for a five year period. The purpose and scope of the Occupational Safety and Health Act of 1970, however, states that every employer covered under the Act must furnish to his employees a place of employment which is free from recognized hazards that have caused or are likely to cause, death or serious physical harm to his employees. The 29 incidents cited resulted in six fatalities and 31 injuries.

The second criteria that is normally applied is that the work environment shall not reduce the life expectancy of the individual. This cannot be applied within the scope of this report since the life expectancy of a miner must include all hazards and only electrostatics was addressed in this study.

The types of incidents that have occurred can be categorized as follows:

1. Incidents Attributed to Lightning

Six incidents were attributed to lighting. In some of the cases abnormal situations were present but in several others, all known preventative procedures were adhered to.

2. Incidents Due to Stray Currents

Six incidents fall into this category. However, three of these are attributed to human error since the current was applied to the detonators due to abnormal, unacceptable actions or practices.

Several of the incidents were readily attributed to faults in mine electrical utilities.

3. Incidents Where the Cause Is Unknown

Several incidents that were reviewed must remain in the unknown cause category. A review of the incident reports at this time did not afford a better resolution of the cause (other than pure speculation) without additional facts.

CONCLUSIONS

The assessment of the adequacy of techniques, materials and equipment for detecting and minimizing these hazards was accomplished by on-site visits to mining operations, review of technical reports and review of materials and devices. The following are specific areas that were addressed.

1. Lightning

Lightning was found to be the greatest cause of mine incidents related to the subject of this report. Breakdown of the atmospheric dielectric starts at a field strength of about 3×10^6 V/meter at ground level. At an average field of about 10^5 V/meter streamers and spark discharges may develop over large distances; many kilometers in the case of lightning.

The most obvious hazards are caused by the direct lightning strikes. The secondary effects are branching discharges and junction steamers, induced currents in conductors and, consequently, the stimulation of blasting materials by currents.

The detonators presently utilized in the U. S. are not immune to this stimuli and the intent of the safety regulations are clear in that safety is obtained by withdrawing from the area. It states "when electric detonators are used, charging shall be suspended and men withdrawn to a safe location upon the approach of an electrical storm."

The key to increased safety is how to get the warning to the operators in time.

In a study by Southwest Research Institute, six devices for early warning were reviewed including: (a) Flash-Counter Warning System, (b) Corona Point Warning System, (c) Radioactive Probe System, (d) Field Mill Warning System, (e) Azimuth/Range Warning System, (f) Triangulation Location System, (g) Modified Lightning Flash Detector, and (h) an Optical Lightning Detector.

Of the six devices, the Azimuth/Range Locator and Corona Point Device showed generally unacceptable performance as lightning warning systems. The one hour or more warning and clear times plus the high false alarm rate are the disqualifying features of the Azimuth/Range Device. The late warning time and high probability of failure to alarm are the primary disqualifying features of the Corona Point Device.

The remaining four units are:

(a) The Triangulation Systems which showed overall best performance of all the systems tested in all performance categories.

(b) The Flash Counter which suffered from a high probability of failure to alarm.

(c) The Radioactive Probe using static field sensing and measurement of field changes.

(d) The Field Mill device using static field plus flash counting and radio noise criteria for alarm indication.

2. Electric Detonators

Electric detonators were involved in 15 of the incidents recorded for the period 1976-1980. The basic mining electric detonator is a one (1) ohm bridge wire with a (0.25 amp no fire current) specification. This is the technology of the 1940's. Other advances have been included to reduce static sensitivity by use of various devices. The development of military caps has continued, spurred by the presence of radar fields where they are used. This led to the specification of a 1 amp, 1 watt no fire requirement which can be met.

The European blasting industry has also made advances in the design of detonators designated as HU which require 500 to 1000 times the energy of our commercial detonators.

It is quite clear from results of this study that the use or development of electric detonators that are less sensitive to current and electrostatic stimuli would go a long ways toward solving the hazard in question. However, because of foreseeable ramifications of such a step; e.g., the cost to the manufacturers and the fact that other extant accessories (blasting machines, etc.) may be incompatible with the new detonators, this may not be feasible from a pragmatic point of view. Although all aspects must be considered before implementation, this still should be seriously considered. The alternative recommendation is to create a greater awareness of the stray current and electrostatic hazard and encourage the mining industry to make greater use of existing guidelines for assessing these hazards.

3. Pneumatic Loading of ANFO

Major advances in the safety of this operation have been accomplished by the use of conductive loading hoses. This has been verified by studies in the field and laboratory. The airveying of ANFO over cap wires still seems to be introducing unnecessary exposure to miners if the collar priming technique is adequate.

4. Bulk Loading

The major concern in this area is the high residual energy developed by the flow of ANFO down boreholes. For boreholes in conductive rock strata, energies can typically range from ~ 0.1 Joule (ANFO with a conductivity of 2×10^{-12} mho/cm) to > 280 Joules (ANFO with a conductivity of 4×10^{-14} mho/cm). For boreholes with a nonconductive liner, the energies are higher, up to 8×10^4 Joules for ANFO with a conductivity of 4×10^{-14} mho/cm. The hazard is lessened by the standard practice of not using electric caps or cap and fuse in these holes. The voltage accumulation on the plastic could be significant; however, insufficient data are available at this time to consider the energy as a possible initiation source for explosive. It was noted in the mine visits that electrostatic charges are generated on the nonconductive delivery hose used on the bulk truck to such a degree that spark discharge to grounded operators is possible and even probable. This should be corrected.

5. Stray Currents

Stray currents can be a cause of incidents or accidents in mines. Several of the incidents reviewed were a direct result of stray current from utilities in the mines.

In view of the complexity of electical utilities and devices in mines and the apparent inadequacy of present detection methods, it seems reasonable that the use of ground fault systems should be made mandatory.

6. Miner Awareness of Hazards

It is the opinion of the investigators that miners are not fully informed as to the nature of explosive related hazards in mining operations or the importance of strict adherence to approved procedures as the best method of controlling these hazards. Special training courses should be promoted by the Bureau of Mines.

7. Accident/Incident Report Format

The format of Accident/Incident report was not sufficiently detailed to permit precise evaluation of causes. Often when reviewing and comparing reports, vital information about climate, explosives or procedures was omitted. It would aid future investigations of accidents and hazard evaluations of mining operations if a more detailed format could be set up for reporting accidents.

RECOMMENDATIONS

The following are specific recommendations that should be considered to improve mine safety. However, the economic impact of these recommendations should be evaluated before they are implemented.

1. Establish a More Formal Criteria for Evaluating Operator Safety

The incident survey conducted as a part of this work indicated that electrostatics was responsible for only a small portion of the safety problems in the mining industry. The present Code of Federal Regulations addresses specific known hazards in blasting and provides operating parameters for preventing accidents. These regulations or IME specifications estabish guidelines such as that require detonators to have a no fire current sensitivity of no less than 0.25 amps. This does not adequately define the exposure of an operator to injury in a mine. His exposure is a summation of the probability of death or injury due to all events associated with his work day. Each specific task, such as wiring a blasting round, is a function of (1) the probability of the cap meeting a specified sensitivity, (2) the probability of his performing the operation incorrectly due to limited training or enforcement, (3) the probability of a source of initiation being present, and (4) the frequency of performing a specific task. A methodology for determining this level of risk has not been established for the mining industry, but could be beneficial for evaluation of mine safety on an individual basis and setting priorities for research programs to identify and correct problem areas.

The U. S. Army has established very restrictive guidelines governing the design and operation of munition plants. While the highly formalized Risk Analysis procedures which are utilized by the Army to demonstrate compliance with these risk criteria may not be literally applicable and appropriate to the mining industy, the philosophy of establishing criteria for acceptable risk to a miner's life and limb - coupled with a systematic approach to the identification and correction of unacceptable hazards - definitely is applicable, appropriate and badly needed in the mining industry. The following excerpt from the Army Standard is included only as an example of how an effective risk criteria can be established.

This excerpt from the Military Standard establishes the hazard classification and the criteria for acceptble risk for designing and operating an explosives manufacturing operation or an ordnance loading facility.

Category Designation	Hazard Classification and Consequences
Category I	Catastrophic: Death, or severe personnel injury (permanent total disability), or system loss expected as consequence of failure mode occurrences. NOTE: A Category I hazard can be counteracted or controlled so that the system, including countermeasures, represents a Category II, III, or IV hazard. If the probability of occurrence of the failure mode for the system is unacceptably high, Category I hazards within the system will be controlled or counteracted to assure the system, as a whole, does not represent a Category I hazard.

Category Designation	Hazard Classification and Consequences

Category II — Critical: Personnel injury, which results in permanent partial disability, or critical system damage expected as a consequence of failure mode occurrence. NOTE: A Category II hazard can be counteracted or controlled so that the system, including countermeasure, represents a Category III or IV hazard. If the probability of failure mode occurrence for the system is unacceptably high, Category II Category II hazards within the System will be controlled or counteracted to assure the system, as a whole, does not represent a Category II hazard.

Category III — Marginal: Personnel injury, limited to temporary total disability, or noncritical system damage expected as a consequence of failure mode occurrence. NOTE: A Category III hazard can be counteracted or controlled so that the system, including countermeasures, represents a Category IV hazard. Countermeasures or controls will be effected within the constraints of cost, schedule, and system effectiveness.

Category IV — Negligible: No personnel injury, other than medical treatment injury (First Aid), or system damage expected as a consequence of failure mode occurrence. NOTE: Countermeasures or controls are unnecessary.

The design goals for each accident category in a project will be no greater than the following mean probability values:

Accident Category	Accidents per Facility-hr	Accidents per Man-hr
Iα	10^{-6}	–
Iβ	–	10^{-7}
IIα	10^{-5}	–
IIβ	–	10^{-6}*
IIIα	10^{-3}	–
IIIβ	–	10^{-6}*
IV	1	1

*NOTE: The sum of the probabilities of a Category IIβ or IIIβ accident occurring shall be 10^{-6} per manhour or lower.

NOTE: The severity of the consequence alone determines the category of hazard, irrespective of the effectiveness of control or the probability that the hazard will be transposed into an undesired event.

This risk methodology is desirable, but from a practical view-point, would require a considerable time to institute. A desirable interim step would be the utilization of a preliminary hazard analysis concept. For Government programs, this is utilized to (1) establish potential problems associated with a particular operation, (2) review the techniques used or available to introduce safety into the operations, and (3) identify areas of uncertainty or where studies must be conducted to establish safe operations. For mining operations, this would form the basis of safety review on an individual basis and would be formalized on a form unique to each mine and supplied by the mine.

2. Lightning Protection

If economically feasible, it is recommended that electric detonators be developed that are less sensitive to electrostaitc stimuli. While it is tempting to recommend development of electric detonators that are less sensitive to currents as well, it is believed that this may make them incompatible with existing blasting circuits and will require concurrent development in this area. Other measures or alternatives would be to encourage or perhaps require the exclusive use of nonelectric initiating systems in these areas and to adopt and deploy a network of one or more lightning sensor systems that are judged, by Government and industry experts, as having the most desirable features.

The following recommendations were made by Southwest Research who reviewed lightning warning systems in a separate study. Hercules' endorses these recommendations as being appropriate to and consistent with the findings of this study.

(1) Based on the terrain dependent predetonation hazard observed in this work, it is recommended that effective mine safety lightning warning system performance specifications be developed independent of sensor technology and based on actual explosive hazard. An extensive survey should be performed to determine incidence of simulated detonation versus terrain for the development of this specification.

(2) In the event sufficient interest exists among the mining industry users, a network of triangulation lightning location sensors should be deployed to provide lightning mapping data to concerned mining operations. The deployment and operational expenses of the network can be prorated through a moderate subscription rate of shared cost by the individual users. Initial outlay will be approximately $500,000 with monthly operating costs near $5,000 for a 20 station net in the eastern United States. The western United States may benefit from the existing Bureau of Land Management lightning location network.

(c) If the mining industry prefers single point sensors at each mining site, then an adaptive lightning warning system should be developed to reduce the failure to alarm rates of existing devices. It is anticipated that an adaptive system could be developed and marketed in production quantities at a cost comparable to existing moderately priced devices, say $5,000-$7,000 (1979 dollars).

3. Stray Currents

To minimize the hazards of stray currents, the following recommendations should be considered:

(a) Encourage or perhaps even require, the use of ground fault interruptors on all electrical service furnished to mining sites where explosives are employed.

(b) Institute a development program for a stray current monitor and establish exact criteria for use and calibration of the equipment.

4. Pneumatic Loading

(a) Establish exact method and standard for grounding during pneumatic loading. This is difficult since the conductivity of the mine can be a significant variable. It is the opinion of the writers that the following should be considered:

(1) Provide integral grounding between all components of the pneumatic loading system with the resistance not to exceed some small value such as 10 ohms.

(2) Insure that the operator becomes a part of the grounded system by requiring conductive gloves or no gloves be worn while loading.

(3) Provide a loading system ground to the mine by either a ground rod near the face or a ground in a breaker hole in the face.

(b) Establish criteria for determining the conductivity of pneumatic loading hose. Semi-conductivity is already defined in CFR 29, part 1910.109(2) as hose of not more than two megohms resistance over its entire length and of not less than 5000 ohms per foot. This could be included in the mining regulations. The measuring method should specify the method of probe attachment and applied voltage. The Army specification for floor conducitivity tests (Paragraph 7.7 of DARCOM-R-385-100) should be considered. This calls

calls for the use of a suitably calibrated ohmmeter which operates on a normal open-circuit output voltage of 500 volts D.C. and a short circuit current of 2.5 milliamperes.

(c) Institute an experimental program to determine the safety of airveying over cap wires.

5. Bulk Loading

Theoretical calculations indicate that the energy generated while loading a borehole exceeds the initiation level for explosive material. It is recommended that a study of Bulk Loading charge generation be undertaken to obtain field measurements for charge generation rates and hole capacitances to determine the electrostatic environment during bulk loading.

6. Cap and Explosive Quality

Cap and explosive quality standards should be established for blasting materials. The following are some of the items that should require certification:

(1) Cap nofire current.

(2) Cap ESD sensitivity.

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APPENDIX

I.   CALCULATION OF THE ENERGY (E) IN AN UNLINED BOREHOLE

$$\text{Vol} = \pi r^2 L$$

$$\int E \cdot dr = \int \rho \, d\,\text{Vol} \quad \text{Gauss' Law}$$

$$\epsilon \epsilon_o E_r \, 2 \pi rL = \rho \pi r^2 L$$

$$E_F = \frac{\rho a}{2 \epsilon \epsilon_o}$$

$$E = \frac{1}{2} \int \epsilon \epsilon_o E_F^2 \, dV$$

$$= \frac{1}{2} \frac{\epsilon \epsilon_o \rho^2}{4 \epsilon^2 \epsilon_o^2} \int_o^a r^2 \, dV$$

$$\int dV = \int 2 \pi L \, r^2 \, dr = \pi r^2 L$$

$$E = \frac{1}{2} \frac{\rho^2}{4 \epsilon \epsilon_o} \int_o^a r^2 \, 2 Lr \, dr$$

$$= \frac{\rho^2 \pi L a^4}{16 \epsilon \epsilon_o}$$

$$= \frac{\rho^2 L^2 \pi^2 a^4}{16 \epsilon \epsilon_o \pi L}$$

$$= \frac{q^2}{50 \epsilon \epsilon_o L}$$

E   = Energy
E_F = Electric field
ρ = Charge density
Vol = Volume of Cylinder
r = a = Radius of hole
L   = Height of material in hole
ϵ = Dielectric constant
ϵ_o = Permittivity

V   = Voltage on hole
q   = Charge on hole

$q = \rho \, \text{Vol} = \rho \pi r^2 L$

For the case of the ground being conductive, the charge will dissipate during loading. The dissipation factor for q was as follows:

$$q = I_o t_r (1 - e^{-t_L/t_r})$$

I_o = Charging current, coul/sec

t_r = Relaxation time, sec

t_L = Loading time, sec

Energy with unlined borehole

$$E = \frac{[I_o t_r (1 - e^{-t_L/t_r})]^2}{50 \, \epsilon \, \epsilon_o \, L}$$

II. CALCULATION OF ENERGY E IN AN UNGROUNDED BOREHOLE (NONCONDUCTIVE ROCK OR A PLASTIC BOREHOLE LINER)

$$E = \frac{q^2}{50 \, \epsilon \epsilon_o L}$$

q = Total charge on hole

ϵ = Dielectric constant of liner

ϵ_o = Permittivity

For the ungrounded hole, the total charge would be:

$$q = I_o t_L$$

$$= (3 \times 10^{-4} \text{ coul/sec}) (98 \text{ sec})$$

$$= 2.94 \times 10^{-2} \text{ coul}$$

I_o = Charging current, coul/sec

t_L = Loading time, sec

$$E = \frac{(2.94 \times 10^{-2})^2}{50(2.5) (8.9 \times 10^{-14}) \, 910 \text{ cm}}$$

$$= 8.54 \times 10^4 \text{ Joules}$$

Voltage on the hole:

$$V = \left(\frac{2E}{C}\right)^{\frac{1}{2}}$$

C = Electrical capacitance of hole

$$= \left[\frac{2 \, (8.54 \times 10^4 \text{ Joules})}{5.39 \times 10^{-2} \, \mu F}\right]$$

$$= 1.78 \times 10^6 \text{ volts}$$

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