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West Virginia University



Proceedings of

THE FOURTH WVU CONFERENCE ON COAL MINE ELECTROTECHNOLOGY

August 2-4, 1978

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PROCEEDINGS
OF THE
FOURTH WVU CONFERENCE
ON COAL MINE
ELECTROTECHNOLOGY

AUGUST 2-4, 1978

LAKEVIEW INN AND COUNTRY CLUB
NEAR
MORGANTOWN, WEST VIRGINIA

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WEST VIRGINIA UNIVERSITY
COLLEGE OF ENGINEERING
DEPARTMENT OF ELECTRICAL ENGINEERING
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AND
U.S. BUREAU OF MINES
IN COOPERATION WITH
DEPARTMENT OF ENERGY
AND
MINE SAFETY AND HEALTH ADMINISTRATION

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PREFACE

The Fourth WVU Cofnerence on Coal Mine Electrotechnology continued to show growth in attendance and participation. The program was expanded to accomodate parallel sessions on one day. The annual meeting of the IEEE Industrial Applications Society Mining Industry Committee was held as part of the meeting. The day preceding the conference was used for a workshop on microprocessors and several IAS committee meetings.

All of this activity demonstrates that this series of meetings is maturing into an established and important meeting for electrical engineers in the mining industry. Therefore, I believe that this document will prove to be a valuable reference for industry and academe.

I want to note special thanks to the authors without whose work this document would not be possible. I am also very grateful to the Session Chairmen and to the Steering Committee for their willingness to work on any job assigned. Special recognition is due John Murphy of the U.S. Bureau of Mines, William Schmidt of the Department of Energy, Michael Hall and Richard Reynolds of the Mine Safety and Health Administration, Al Newcome of the Consolidation Coal Corporation, and Jack Strange of the Mine Safety Appliances Company for their coopration in planning and executing the program.

Also, a very special note of gratitude is due Mrs. Marian Jones and my secretary, Miss Karen Ruckle, for their extra effort which made the conference and the proceedings possible.

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Conference Chairman

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OPERATION OF LIGHTING SYSTEMS ON COAL MINE D.C. POWER SYSTEMS

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Mine Electrical Systems Branch
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Pittsburgh Technical Support Center
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ABSTRACT

The 300 volts direct current (vdc) or 600 vdc power utilized in coal mines contains high-voltage transients and has poor voltage regulation. This can cause temporary blackouts or permanent damage to lighting power systems. Recent lighting applications are discussed including industry solutions to these problems. A current and voltage switching regulator is described including field application experience.

INTRODUCTION

The mining industry has encountered difficulties in the past when new equipment was required to be powered by direct-current line voltage. Since approximately 25 percent of all electrical face equipment, required to be illuminated, is powered by direct-current, a reliable power supply source is a necessity.

Although operation of lighting equipment on d.c. voltage would be easier with incandescent lights, the illumination requirements listed in the Code of Federal Regulations,(1)[1]/ could not be achieved using present incandescent sources because they only yield approximately 20 lumens per watt. Incandescent sources also have a relatively short life due to their filaments failing with the vibration experienced on most mining machines.

1/Numbers in parenthesis refer to a list of references shown at the end of this report.

Present day mercury lamps have efficiencies two or three times greater than general service incandescent lamps. Fluorescent lamps range from 45 to 80 lumens per watt and high pressure sodium discharge lamps can achieve an efficiency of 115 lumesn per watt. These gas discharge lamps however, must have a ballast to limit the current going through the lamps once the arc that provides the illumination is struck. The ballast functions as an equalizing force and stabilizes the current flow through the gas lamp.

These gas lamps are therefore, dependent upon a power supply or power conversion for proper operation on direct-current power. This circuitry must accept the mine trolley voltage and either invert it to an a.c. voltage or condition the supplied voltage to an acceptable level.

D.C. SYSTEM CONSTRAINTS

Direct current line voltage as supplied from rectifiers may be nominally in the regions of 250 to 350 or 550 to 650 vdc. Variations can be expected as low as 170 vdc for the lower range and as high as 750 vdc for the higher range.

Severe electrical transients in mine power systems are numerous. Line transients can be two or three times the normal voltage. The currents associated with these surges are generally limited to the surge voltage divided by the effective surge impedence at that point in the system. Surge current with a high rate of change can be a serious problem for solid state devices. These devices must be designed to have a high enough rating to meet these higher voltages.

Since the sources of transients are varied; prediction and measurement of transients is very difficult. It has been shown however, that bursts of energy can be as high as 50 joules. (2)

LIGHTING REQUIREMENTS

Fluorescent lights used at present on mining machines range from 15 to 65 watts per lamp. The voltage at the lamp depends on the type of lighting system, but is at least 100 volts steady state, with a starting pulse of 200 volts. The frequency may be 600 Hertz (Hz) or higher. Lamp efficiency is higher at high frequency and the ballast inductive components are smaller and lighter. The frequency supplied by Ocenco's[2/] inverter-ballast can be from 18,000 to 100,000 Hz.

2/Reference to specific brands or trade names is made for identification only and does not imply endorsement by MSHA.

Voltage regulation is obtained in this system by varying the frequency so that the voltage drop through a series element changes.

Mercury vapor lamps presently used are either 100 watts or 175 watts. The lamps may be operated on d.c. with a slightly decreased brilliance and total life. A pulse of about 200 volts or higher is required to start these lamps. The lamp is constructed as illustrated in Figure 1 which shows a third electrode providing a shortened starting gap to permit breakdown of the gap at a low voltage.

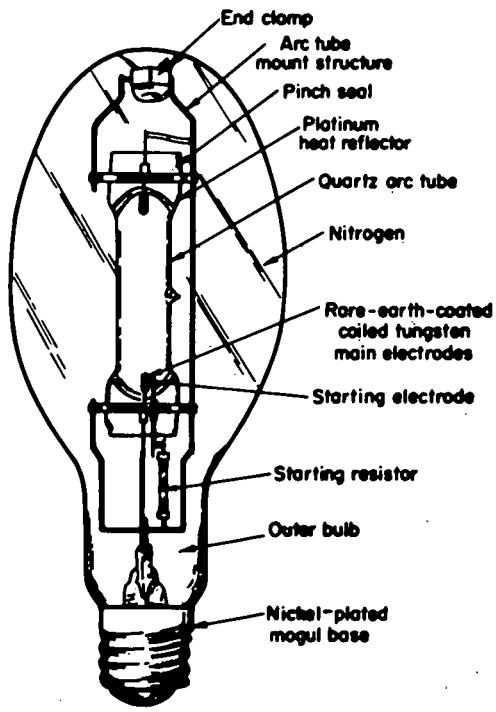


Figure 1

Figure 2 shows the voltage and current that must be supplied by the power supply ballast while the lamp is building up to its full brilliance. The voltage across the lamp is initially about 30 volts and gradually increases to a steady state value of about 130 volts in about 4 minutes. The current through the lamp starts at about 1.3 amperes and decreases to about 0.8 amperes. If the voltage drops from the operating level of 130 vdc to approximately 90 vdc the lamps will go out and stay out until the lamp can restrike. This can take as long as 4 to 5 minutes.

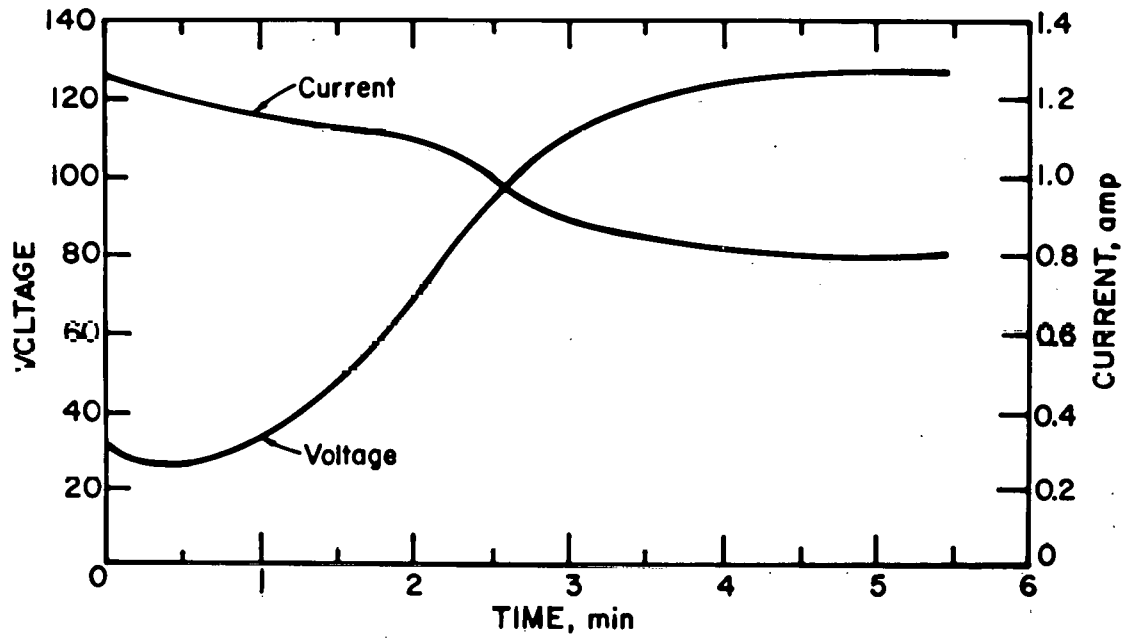


Figure 2 - Mercury Starting Curves

Another lamp which is utilized in d.c. lighting systems is the high pressure sodium lamp. This is an arc lamp utilizing mercury plus sodium. An impulse of about 1500 volts is required to start the arc. The power supply is generally a specialized inverter circuit providing the necessary high voltage kick required to start or restrike the lamp. Restrike time for this lamp is only about 30 seconds.

POWER SUPPLY CIRCUITS

The basic circuits used for powering the types of lamps considered have been either switching regulators, inverters, or series resistors.

A 240 volt, 500 watt ballast resistor placed in series with a 100 watt mercury headlight across the 300 volt line is shown in Figure 3.

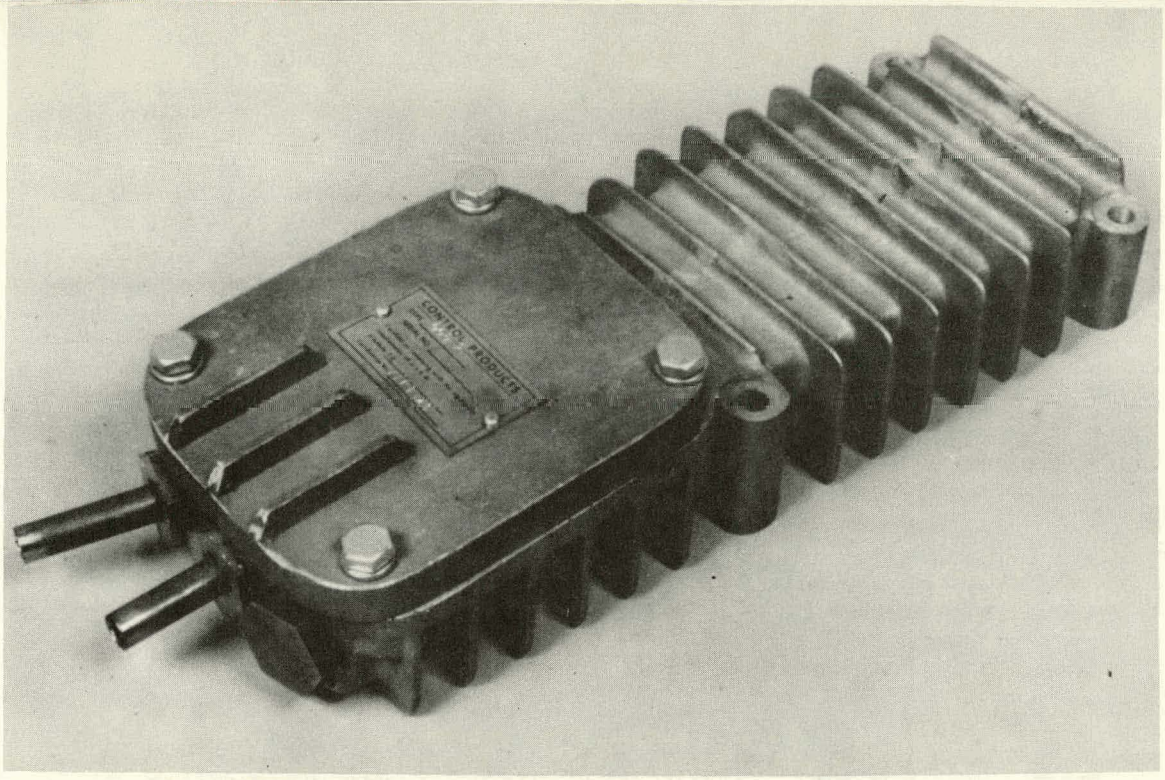


Figure 3.

Total resistance in series with the lamp is 220 ohms under operating conditions, resulting in a drop of about 175 volts when lamp is at steady state. The wattage dissipated would totally negate the efficiency gained by using a gas discharge lamp. Also there would be no voltage regulation. This is necessary since arc lamps operating voltages tend to change throughout lamp life.

The basic circuit for a switching regulator used as a lighting power supply is shown in Figure 4. The output voltage is a pulsating wave whose shape depends on the value of the input voltage relative to a nominal value.

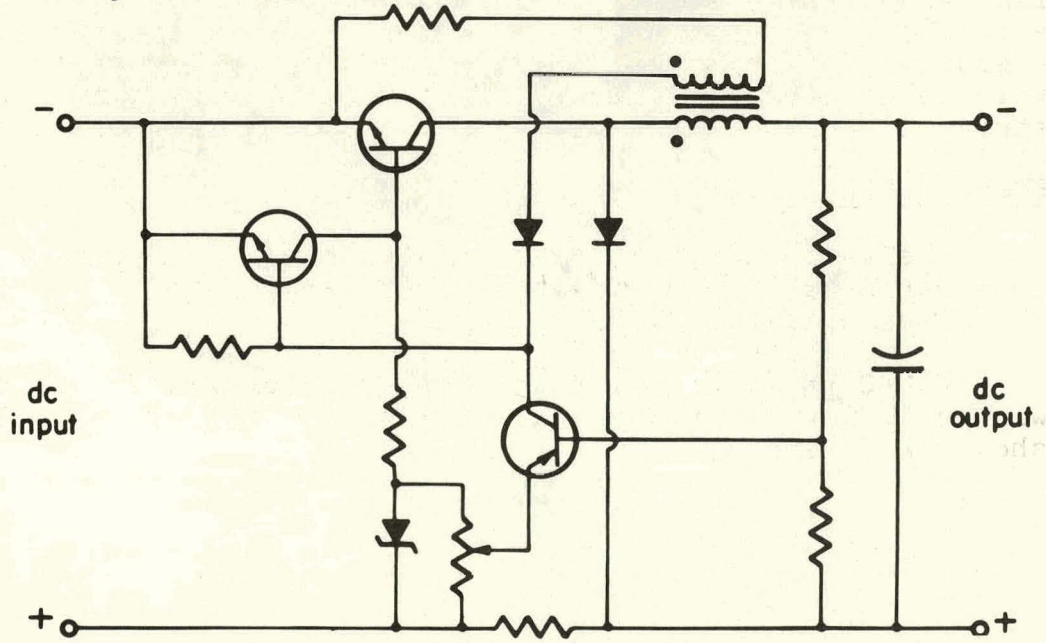


Figure 4

MSHA technical support personnel decided to improve the basic switching regulator circuit performance by combining it with a voltage oscillator. The ballast, Figure 5, can be designed to function with a nominal 300 or 600 volts. Its use is presently limited to mercury vapor lights, but it may be used wherever a constant average direct current is desired. For instance, this regulator can be used to supply a high output fluorescent lamp. The most important feature of this ballast is to supply an average constant current to the load impedance. (Figure 6) Gas discharge lamps require a constant current even with a varying input voltage. As a mercury lamp warms up, its resistance increases and a higher voltage is needed to provide the rate current. This ballast provides this constant current with the necesary voltage increase demanded by the increasing resistance.

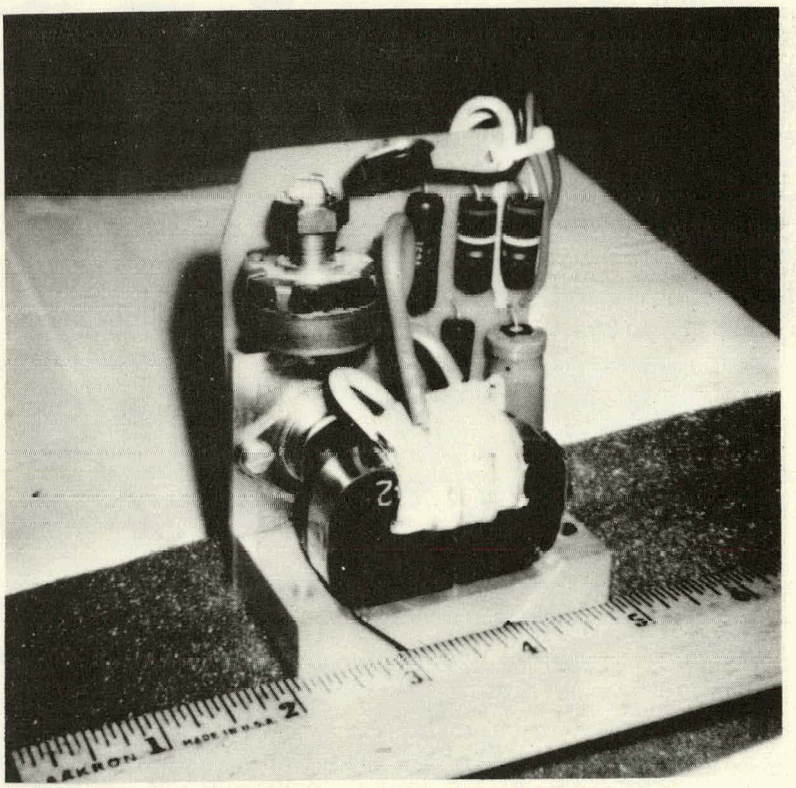


Figure 5. Keltech, Inc. Ballast

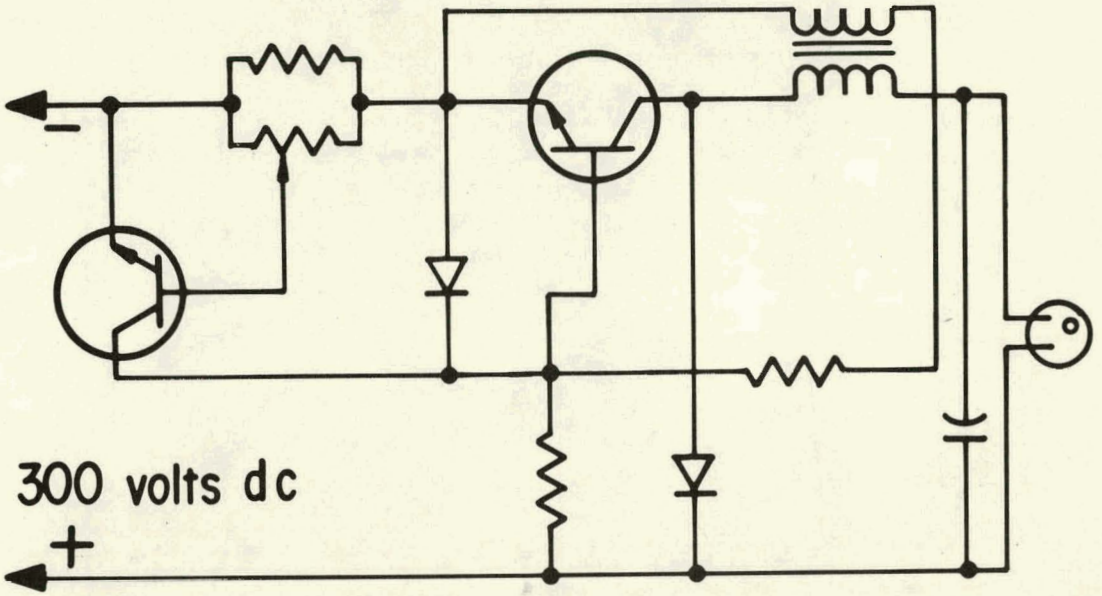


Figure 6. MSHA Switching Regulator Ballast

This power supply system is presently installed on a Galis 300-A roof bolter at a Valley Camp Coal Mine in Charleston, West Virginia. It has been functioning since March 1978. (Figures 7 and 8)

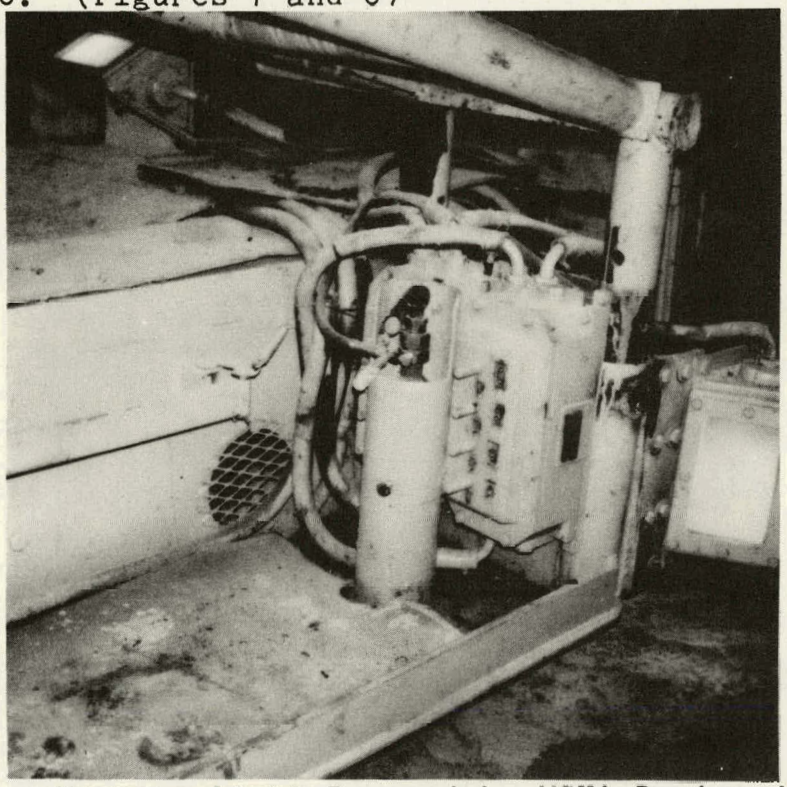


Figure 7. Mercury Light Powered by MSHA Designed Ballast

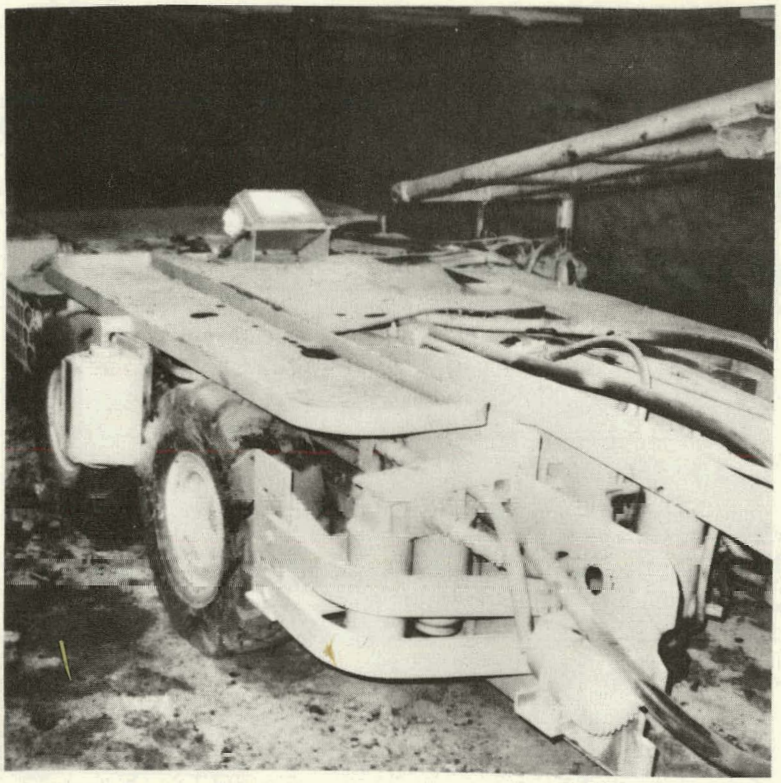


Figure 8. FMC Galis 300A Roof Bolter

This ballast configuration has also been installed near the rotary pump area at Montour #10 Mine of the Consolidation Coal Company. The system as shown in Figure 9 has been operating since July 6, 1976, with relatively little down time.

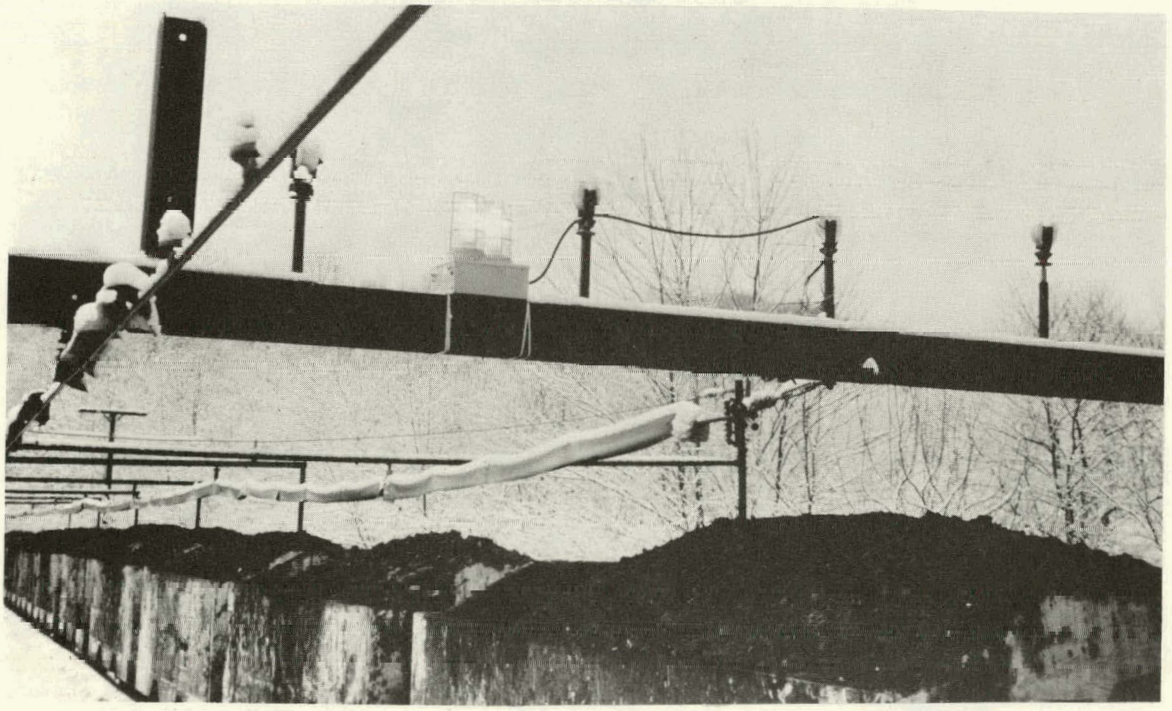


Figure 9.

A 600 vdc ballast and mercury vapor lights have been in operation since July 6, 1976, at Consolidation Coal Company, Westland Mine, with two failures to the present time. (Figure 10)



Figure 10.

Another device was designed to be utilized with Ocenco's Fluorescent Illumination System. Operating from a 300 vdc line requires a d.c. to d.c. converter to supply up to 10 amperes within the intrinsically safe power requirements, therefore the cables leading to the luminaires do not require explosion proof junction boxes and can utilize connectors for fixture replacement. (Figures 11 and 12)

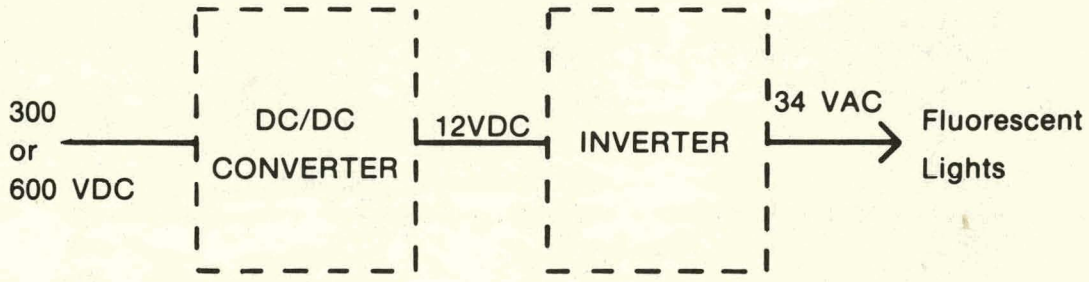


Figure 11.

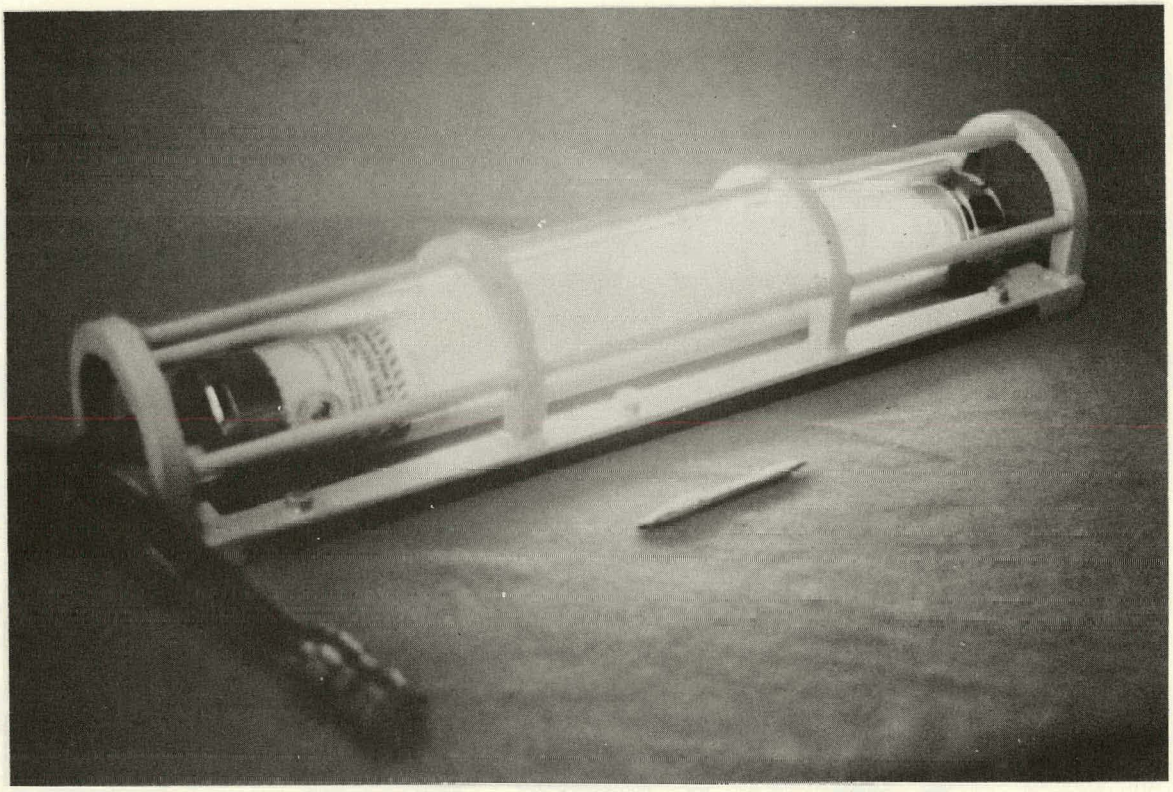


Figure 12.

This system is presently installed and operational at Canterbury Coal Company, in Avenmore, Pennsylvania. This 300 vdc system was installed October 1976, on a Galis 300A Roof Bolter under a Bureau of Mines contract. It has been operational continuously with two power supply failures and one transient suppression network failure. The lights are acceptable to mine personnel and mine engineers. This coal company has ordered more systems for other mining equipment.

A 600 vdc system in being tested on a trolley line at a mine in Pennsylvania under a Bureau of Mines contract. This system has been operational for two months with a minimum of failures.

Design of a 650 watt resonant feedback inverter for operation from a 300 volt trolley line was completed recently by Westinghouse Electric Corporation under a Bureau of Mines contract. This unit will supply full power for two 100 watt mercury lamps and six 75 watt VHO fluorescent lamps for an input voltage range of 230 to 350 vdc.

The inverter includes a resonant feedback circuit which converts the output of the d.c. power source to an alternating current output and regulates the inverter output voltage level by varying the frequency. This unit has not been field tested at this time.

McJunkin Corporation has an inverter and transient suppression network installed at several mines which use 300 vdc power. (Figure 13) They use both fluorescent and high pressure sodium lights for their machine mounted lighting syste. Several installations were installed during 1977. A 600 vdc lighting system is scheduled to be installed in July 1978.

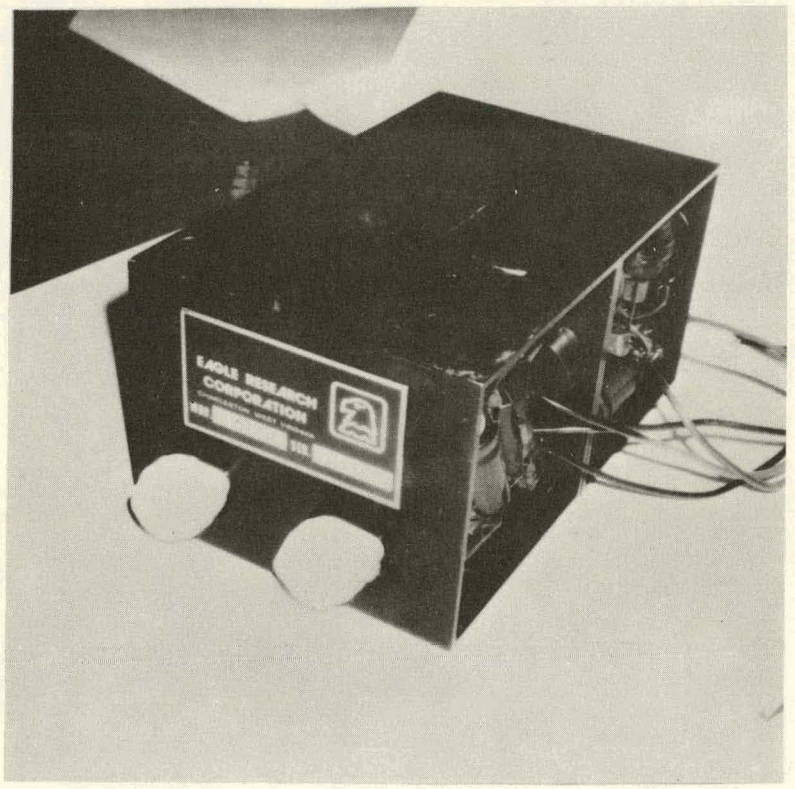


Figure 13.

Availabe information as to the status of development of other d.c. lighting systems is as follows:

Bacarach Co.

A 300 vdc power supply has been designed and bench checked for fluorescent light only.

Crouse Hinds Corp.

A 1 KVA current limiting inverter has been designed and submitted for certification. This system is designed for use with mercury vapor, high pressure sodium and fluorescent sources. This sytem has been designed for both 300 and 600 vdc nominal voltages.

Control Products

A 1 KVA inverter is being tested on a trolley line.

Keltech, Inc.

The MSHA designed ballast for 300 vdc mercury lamps is currently available from Keltech, Inc., Roanoke, Virginia.

REFERENCES

1/Code of Federal Regulations, Part 75 of Title 30, Sections 75.1719 through 75.1719-4.
2/Digital Computation of Transients and Safety Testing of Mine Electrical Power Systems - WVU (NTIS PB265074/AS)

ILLUMINATION CASE STUDY-40 CU. YD. STRIPPING SHOVEL

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ABSTRACT

This paper addresses itself to the problem and solution of illuminating the working area around a large (40 cu. yd.) stripping shovel. It is a case study in which the lighting fixture supplier employs a computer program to determine floodlight numbers, types, positioning and aiming in order to meet specified levels of illumination.

An operational description of the "CALL" (Computer Aided Lighting Layout) program used in this study is discussed as a lighting design tool. Computer program inputs, problem size limits, versatility, and final printout options are defined.

Design of the recommended lighting system is described and results of "on-site" photometric measurements are presented as evidence of the "CALL" program effectiveness.

INTRODUCTION

Four events have encouraged me to think a paper on this subject might generate interest among the attendees of this conference. These events are:

- Proposed Illumination Regulations for Surface Coal Mines.
- Establishment of the Mine Safety and Health Administration under the Labor Department.
- Increased interest in illumination in the mining industry.
- Computerized Illumination System Design.

"Proposed Mandatory Safety Standards for Surface Coal Mines and Surface Work Areas of Underground Coal Mines," was published in the January 13, 1977 Federal Register as proposed revisions and amendments to "Part 77, Subchapter 0, Chapter I, Title 30, Code of Federal

Regulations." Mine Enforcement and Safety Administration (MESA) was to be the enforcement agency under the Department of Interior. Included in these proposed safety standards are requirements for illumination in, on and around the various mining equipments used. A primary purpose of this paper is to show the relative ease of designing an illumination system to meet the proposed regulations.

MESA has been reorganized under the Department of Labor as the "Mine Safety and Health Administration" (MSHA). MSHA has broader responsibilities, whereby they now protect both the coal and non-coal miner under a single law. Experience gained in underground mine lighting should, with good reason, encourage MSHA to implement the surface mining illumination regulations with dispatch and confidence. In addition, it is anticipated that the proposed regulations will also apply to non-coal mining in the near future.

There is a growing trend in the mining industry, when procuring machine mounted lighting equipment, to specify that the equipment meet the proposed illumination regulations. This case study is one of those instances and points out not only an industry awareness of the proposed regulations, but a committment to comply even before promulgation.

Calculating illumination values is extremely tedious which makes it very attractive for computer programming. In this case study a computer program is the key that reduces the design function to nearly a routine task. In addition, the computer instills confidence that the illumination system will perform to its design criteria.

This paper will share a lighting design experience. More specifically, it involves the design of an illumination system for a large stripping shovel. In describing this design experience the following topics will be discussed:

- General Illumination Problem.
- Definition of Specific Illumination Problem.
- Operational Description of "Computer Aided Lighting Layout" (CALL) Program.
- System Design.
- System Performance.
- Conclusions.

GENERAL ILLUMINATION PROBLEM

This discussion is intended to review enough of the basic illumination theory to merely highlight the magnitude of the design calculations.

Two basic laws are involved in calculating illumination:

- Inverse square law.
- Cosine law.

The inverse square law states that illumination varies directly with the luminous intensity (candlepower) and inversely with the distance squared. The cosine law states that the illumination varies as the cosine of the angle between the light ray striking the surface and a normal to that surface. These laws are illustrated in Figure 1. The beam axis of a light source(S) is aimed at point A, with illumination at point P to be calculated. A straight line between the source and point P represents the distance (D) traveled by the light ray and forms angle α with a line perpendicular to the horizontal surface. Illumination at point P is computed as follows:

$$\text{footcandle (fc)} = \frac{\text{candlepower (CP)}}{D^2} \times \text{Cosine } \alpha$$

Candlepower is determined from published photometric data such as the typical plots shown in Figure 2. The beam angle (β), shown in Figure 1, is the angle between the beam axis and the light ray striking point P. In calculating the illumination at point P the candlepower of the source, at beam angle (β), must be extracted from a photometric curve. Although some luminaires have a circular beam pattern, most have distinctly different beam intensities in their horizontal and vertical planes as shown in the examples of Figure 2. Regardless of the type of plot, they all provide candlepower versus angular displacement from the light beam center.

Figure 3 is a three source problem where the illumination at any given point on the surface is the sum of the values contributed by each source. Under the proposed surface mining illumination regulations (which will be reviewed later) footcandle measurements are to be taken at least every ten feet. In designing a mining machine illumination system a dozen or more luminaires may contribute light to a given area defined by several hundred points. The large number of calculations required in the system design is the fact that makes the case for a computerized lighting system design.

DEFINITION OF SPECIFIC ILLUMINATION PROBLEM

The machine for which this lighting system was designed is a 40 cubic yard Marion 5561 electric power shovel. It was required that illumination be provided around the main frame, beneath the boom and on a work face of up to 85 feet of overburden.

The customer had the following well conceived ideas which were incorporated in the illumination requirements:

- A mix of H.I.D. and Incandescent to preclude a total blackout in the event of H.I.D. outage due to a momentary power interruption.
- Good color rendition.
- Illumination levels that would promote production.
- Illumination around the main frame and beneath the boom must exceed the proposed MSHA regulations.
- Heavy duty lighting equipment designed to withstand the rigors of the application.

As a base line for design the MSHA proposed regulations were used. Those proposed regulations; as they apply to shovels, draglines and wheel excavators, are summarized in Table I. Figure 4 depicts the shovel in a working place with the areas to be illuminated outlined.

To provide good color rendition metal halide was selected as the H.I.D. source and tungsten halogen as the incandescent type. To promote production it was determined that an average intensity of at least 10 footcandles of illumination should be provided around the machine and on the work face. This value would also satisfy the proposed regulations by a wide margin.

OPERATIONAL DESCRIPTION OF "COMPUTER AIDED LIGHTING LAYOUT" (CALL) PROGRAM

Computing the illumination at a given point at a given distance from a single source is not difficult. But, when several sources contribute light on a given point and when hundreds of points must be considered, the problem becomes an enormous mathematical exercise. To manage this problem, "CALL" was developed by Phoenix Products Company in late 1973 and put into operation in early 1974 as an illumination system design tool. What prompted that development was a U.S. Coast Guard regulation requiring specified illumination levels on tanker decks. Ship owners and ship builders were asking for lighting recommendations and "CALL" provided the answers. In as much as our lighting products are designed for specialized industrial applications, "CALL" was programmed to handle a variety of problems. Other examples of "CALL's" versatility include illumination system designs for ship engine rooms, nuclear power plant reactor buildings, dragline machinery rooms, dragline working pits, and a wide variety of shovel and crane applications.

A Burroughs 6700, contracted on a time sharing basis, is the computer used for the "CALL" program. This provides economical access to a high speed, large capacity machine.

Each lighting problem is defined by an X-Y coordinate system shown in Figure 5 and the following information fed into the computer:

- Photometric characteristics of the fixtures used.
- Height of each fixture above surface.
- X-Y location of each fixture.
- Azimuth aiming angle.
- Elevation aiming angle.
- Identification of the project by customer or job.
- Type of computer printout desired.

Three types of computer printouts can be selected. The most widely used is a graphic-grid printout. This prints a rectangular coordinate system with the Y-axis across the page and the X-axis down the length of the page. An asterisk (with total accumulated horizontal footcandles) is printed at the intersection of each ordinate and abscissa value selected in the print option.

The graphic-grid printout is selected to some convenient scale from one inch equal one foot up to one inch equal any whole number. Illumination points can be printed on several optional spacings. The most dense is on the corners and centers of one inch squares. Such a printout option, with a scale selection of one inch equivalent to ten feet, would provide "to scale" illumination point printouts on ten foot squares at corners and at the center over the entire area under study. This option is identical to the grid pattern required for photometric measuring in the proposed Federal regulations.

The second optional printout is a tabular-grid report. Total illumination values for each point in the grid is given in tabular form including the X-Y coordinate of each point and the contributing footcandles from each lamp.

An individual-point report is the third printout option. Points which do not fall nicely into a grid format can be selected individually and entered into the computer. The printout is like the tabular printout with total illumination at each selected point and each lamp's contribution to that total.

"CALL" can handle, in a single lighting problem one to ninety-nine fixtures each mounted at different heights, locations, elevation aiming angles and azimuth aiming angles. In addition, twenty different types of fixtures (Photometric curves) can be included in a single problem.

SYSTEM DESIGN

Typically, illumination system design begins with a layout of the problem. Figure 6 is representative of such a layout. On the layout, lighting fixtures are positioned in logical mounting locations. Based on knowledge of the equipment, experience and a few quick hand

calculations the number and types of fixtures as well as aiming angles are established for a first iteration "CALL" run. Depending on the results, aiming angles may be adjusted or fixtures may be substituted to achieve the desired illumination levels. Generally, no more than two iterations are required to reach the design goals.

Figure 7 is a computer printout listing the pertinent information for each fixture included in the particular computer run. Figure 8 is an explanation of the column headings shown on the fixture information printout of Figure 7.

Figure 6 is the final layout of the Marion 5561 machine showing fixture locations, and azimuth and elevation angles for each. Computer printout, Figure 9, is a plan view of the ground level illumination around the machine. For convenience in orientation to the X-Y coordinate system, the main frame is shown solid and the boom with dashed lines. The values on the printout are horizontal footcandles.

Figure 10 is a computer printout of vertical footcandles on the work face. In interpreting Figure 10, be aware that orientation is such that ground level (X=0) appears at the top of the printout and top of the excavation (X=85) is at the bottom.

Inspection of Figure 9 reveals that the proposed MSHA illumination levels have been exceeded, as intended, and uniformity is extremely good. Although the work face is not covered by the proposed illumination regulations, the design goals have been exceeded as shown by Figure 10.

SYSTEM PERFORMANCE

The stripping shovel used in this study is an older machine transported from a distant site, modernized and re-erected near Zanesville, Ohio. Because of the coal strike, and other delays, the machine did not begin operation until mid July 1978 at which time the site was visited to measure actual illumination levels. The object was to compare measured against computed footcandle values. During erection, the lighting fixtures had been installed and aimed for general illumination around the erection site. Unfortunately, the lights had not been aimed according to the "CALL" design criteria. To further compound the situation severe electrical problems were being encountered on the machine which completely curtailed operations. Because of the pressure on the maintenance people any suggestions to re-aim the lights would have been imprudent.

To provide data that would substantiate the validity of the "CALL" program it was decided to make photometric measurements around the machine with the lighting system as it was being operated and then simulate that situation on a separate "CALL" run.

Footcandle readings were taken at eighty seven measured points around the machine and on a vertical plane representing the work face.

A "CALL" program was then run with our best approximation of the lighting situation as it existed. Results of a comparison between 87 actual and computed footcandle values is summarized as follows:

29 points (33.3%) ---------- Zero variance
36 points (41%) ---------- within 10% variance
56 points (64%) ---------- within 20% variance
66 points (75%) ---------- within 30% variance

The large percentage of points that show a relatively small difference, between measured and computed footcandles, would seem to verify the accuracy of the "CALL" program and therefore substantiate its use as an illumination design tool. It also is noteworthy that where differences occurred, 45 percent were plus and 55 percent minus indicating that no bias was experienced.

Although the responsible mine personnel were very satisfied with both the quantity and quality of the light, approximately 20 percent of the applicable footcandle readings were below the proposed illumination regulation levels. This can without question be corrected when the fixtures are aimed for optimum effectiveness.

Photographs of the machine and some of the lighting installations are shown in Figure 11.

CONCLUSIONS

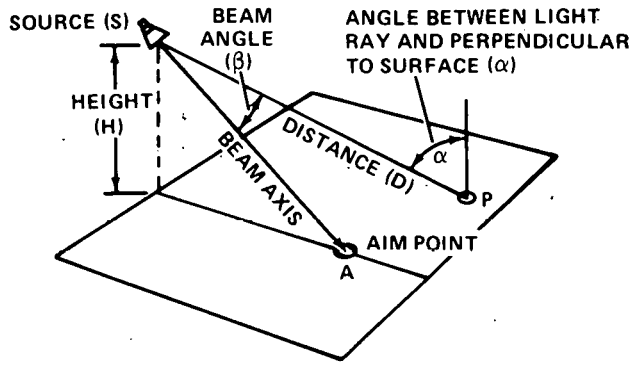
The primary conclusion drawn from this experience is that computerization of lighting problems, such as that performed by our "CALL" program, is a very valuable design tool for the lighting system designer. Because of the number of unique and special lighting situations experienced in the mining industry, computerized lighting design should be especially attractive. Engineers responsible for lighting would do well to avail themselves of lighting system design services offered by the lighting industry.

It also can be concluded that when aiming of the light fixtures has been optimized the proposed Federal illumination regulations, where applied to this machine, will be exceeded.

Finally, although a fairly arbitrary matter, the color rendition of the mix of Metal Halide and Quartz Halogen lamps provides a highly satisfactory quality of light for this application. In addition, the illumination levels are better than the operators have previously experienced in similar "seeing" tasks.

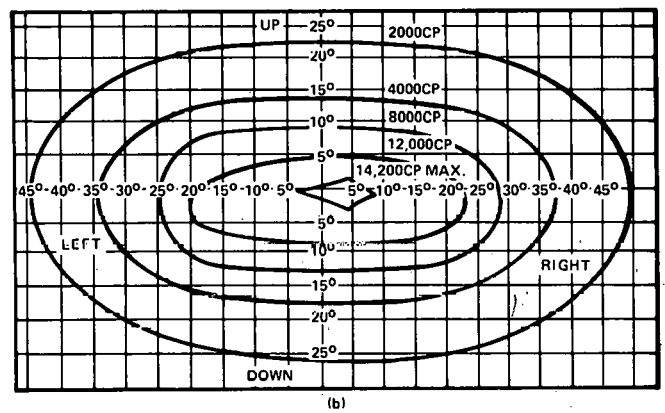
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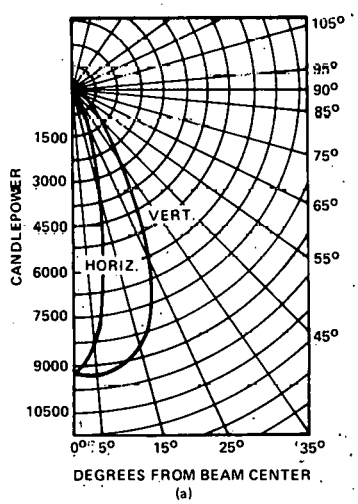
acknowledgement: Phoenix Products Company, Inc. sincerely appreciates the cooperation and assistance extended by Muskingum Mine Joint Venture at Zanesville, Ohio. Their support was vital to the preparation of this paper.

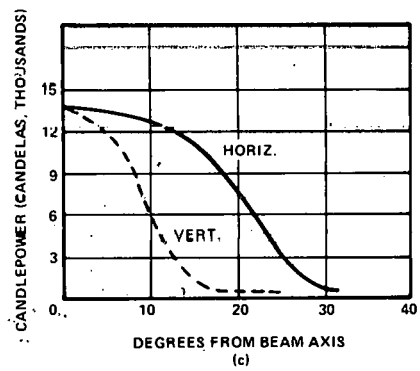


$$fc = \frac{\text{CANDLEPOWER}}{(\text{DISTANCE})^2} \times \text{COSINE } \alpha$$

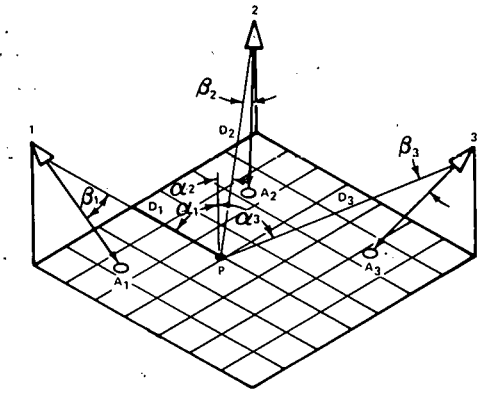
BASIC ILLUMINATION PROBLEM
FIGURE 1





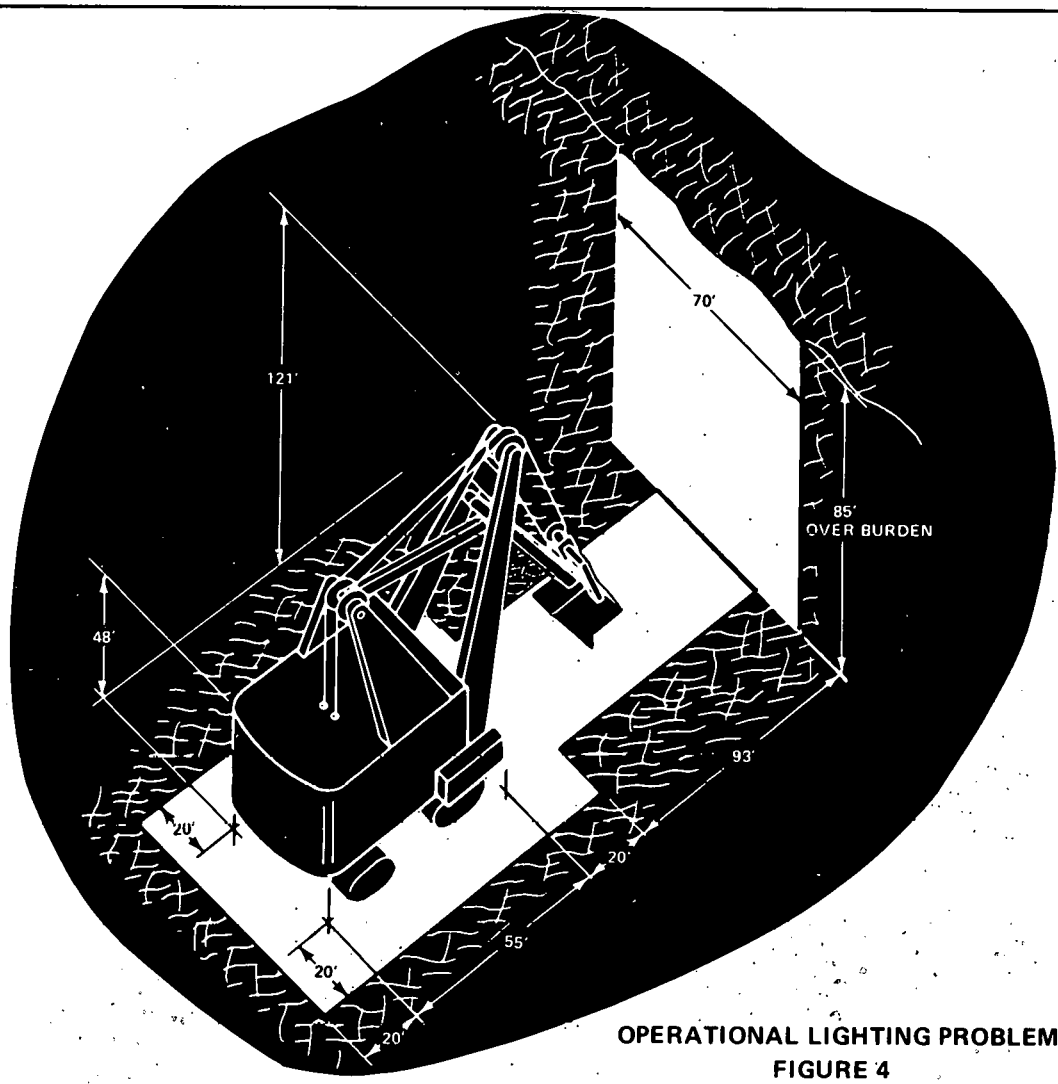


TYPICAL PHOTOMETRIC PLOTS
FIGURE 2

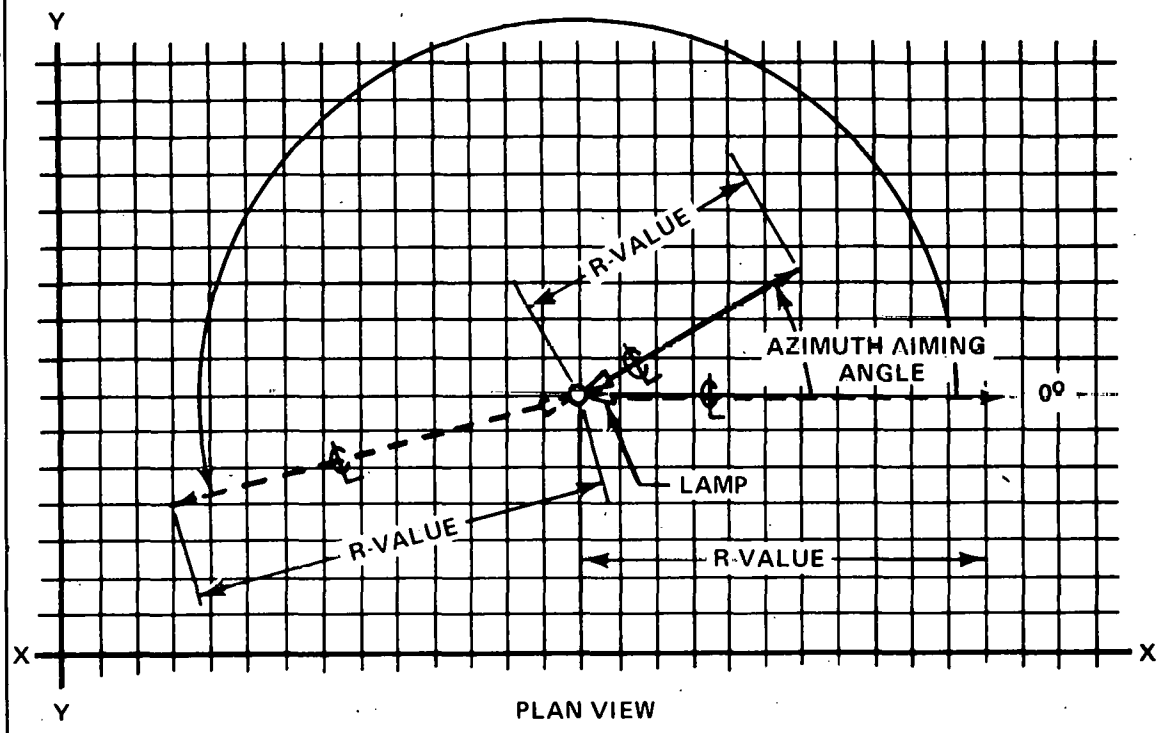


$$\text{TOTAL FOOTCANDLES AT POINT P} = \frac{CP_1}{(D_1)^2} \times \cos \alpha_1 + \frac{CP_2}{(D_2)^2} \times \cos \alpha_2 + \frac{CP_3}{(D_3)^2} \times \cos \alpha_3$$

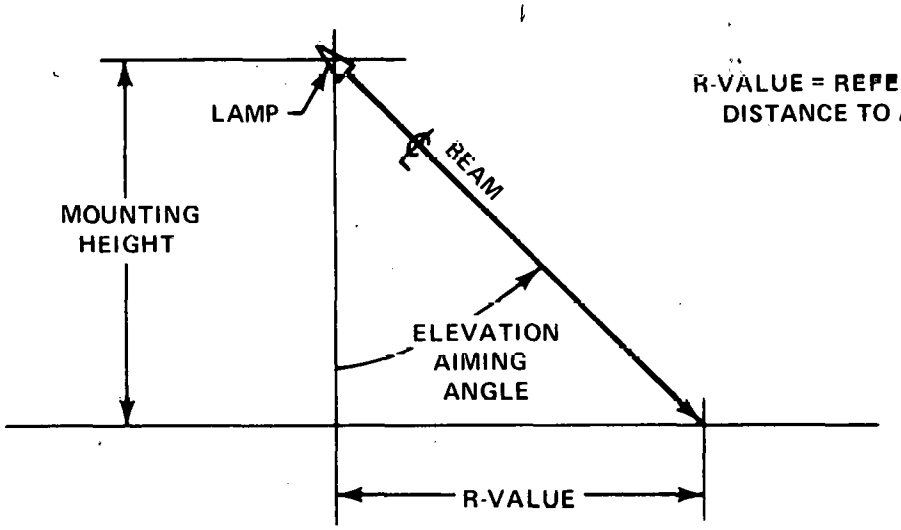
MULTI-SOURCE ILLUMINATION PROBLEM
FIGURE 3



OPERATIONAL LIGHTING PROBLEM
FIGURE 4



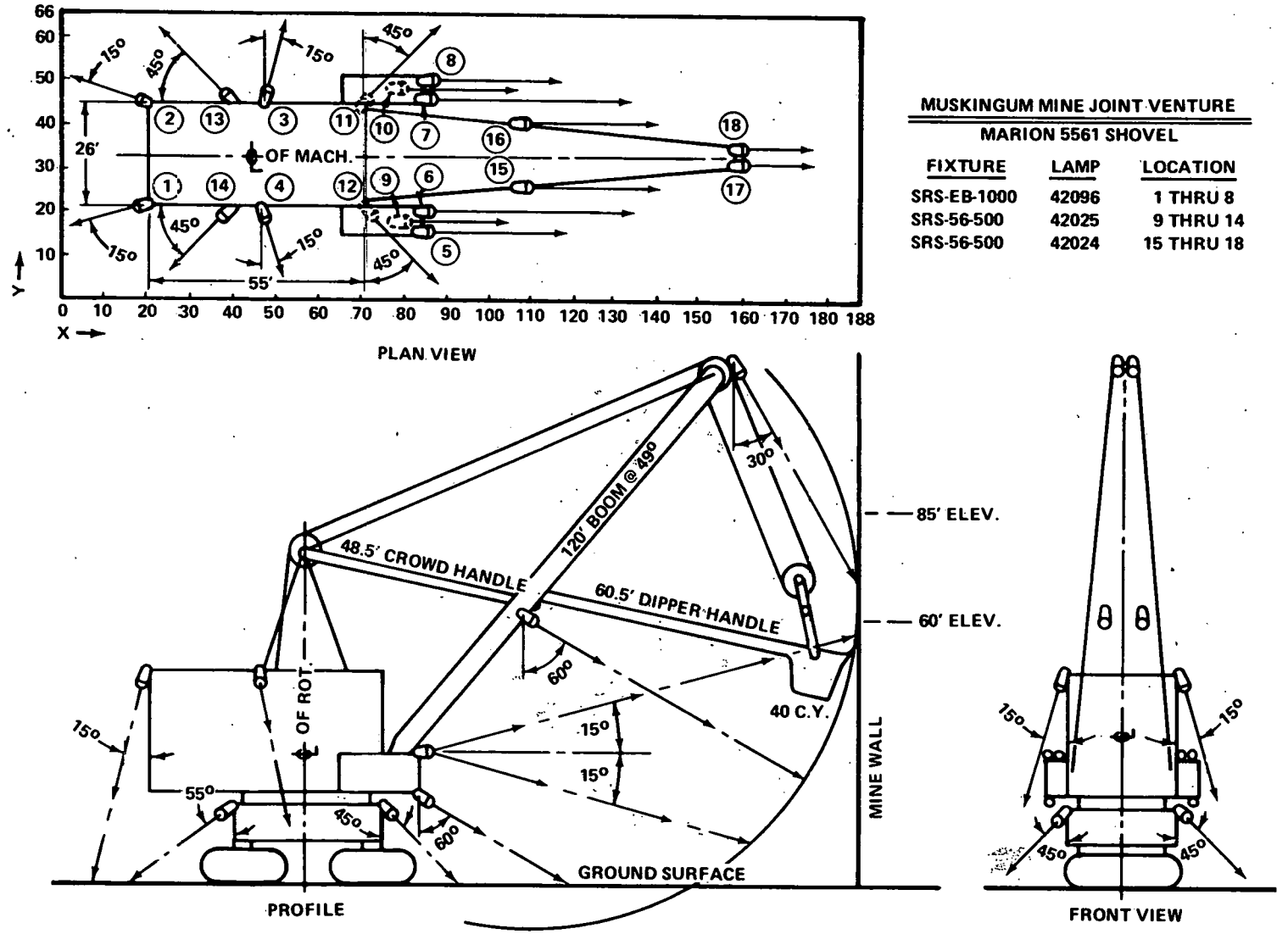
PLAN VIEW



R-VALUE = REFERENCE DISTANCE TO AIM POINT.

TYPICAL SECTION THRU BEAM CENTERLINE

"CALL" PROGRAM CO-ORDINATE SYSTEM
FIGURE 5



MARION 5561 LIGHTING LAYOUT
FIGURE 6

FIXTURE MOUNTING IDENTIFICATION

DATE 04/03/78 PAGE 1

LIGHTING PROFILE FOR GASH & ASSOCIATES #5561 MARION SHOVEL MUSKINGUM MINE OPERATION GROUND SURFACE WITH ALL LIGHTS

ILLUMINATION VALUES REPRESENT CUMULATIVE HORIZONTAL RAW FOOTCANDLES NO MAINTENANCE FACTOR CONSIDERED

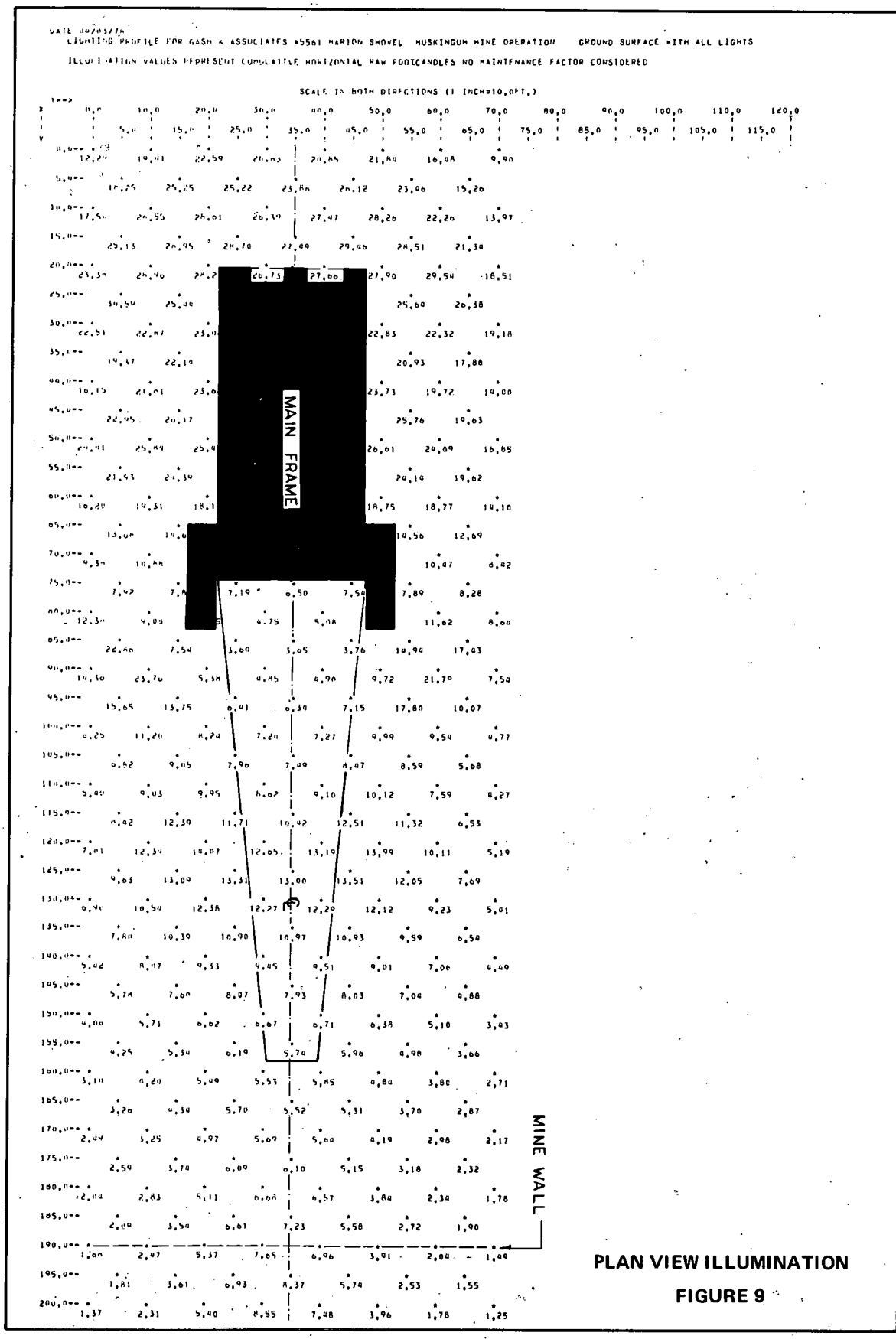
LP #	TYPE	LOC-X	LOC-Y	HEIGHT	ELEV	AZIMTH	IV	R-VALUE	LAMP DESCRIPTION
1	1153	20,00	20,00	98,00	15,00	165,00	0	12,86	SRS-E8-1000(10029-2),CLEAR METAL HALIDE (42096)
2	1153	20,00	46,00	98,00	15,00	165,00	0	12,86	SAME AS LAMP NUMBER 1
3	1153	48,00	46,00	98,00	15,00	15,00	0	12,86	SAME AS LAMP NUMBER 1
4	1153	48,00	20,00	98,00	15,00	285,00	0	12,86	SAME AS LAMP NUMBER 1
5	1153	83,00	16,00	34,00	75,00	0,00	0	126,89	SAME AS LAMP NUMBER 1
6	1153	83,00	18,00	34,00	90,00	0,00	0	*******	SAME AS LAMP NUMBER 1
7	1153	83,00	48,00	34,00	90,00	0,00	0	*******	SAME AS LAMP NUMBER 1
8	1153	83,00	50,00	34,00	75,00	0,00	0	126,89	SAME AS LAMP NUMBER 1
9	1055	83,00	17,00	26,00	60,00	0,00	E	45,03	SR8-56-500(10102), QUARTZ WIDE FLOOD (42025)
10	1055	83,00	49,00	26,00	60,00	0,00	E	45,03	SAME AS LAMP NUMBER 9
11	1055	75,00	46,00	20,00	45,00	45,00	E	20,00	SAME AS LAMP NUMBER 9
12	1055	75,00	20,00	20,00	45,00	315,00	E	20,00	SAME AS LAMP NUMBER 9
13	1055	41,00	46,00	20,00	55,00	135,00	E	28,56	SAME AS LAMP NUMBER 9
14	1055	41,00	20,00	20,00	55,00	225,00	E	28,56	SAME AS LAMP NUMBER 9
15	1056	108,00	25,00	52,00	60,00	0,00	C	107,39	SR8-56-500(10102), QUARTZ MEDIUM FLOOD (42024)
16	1056	108,00	41,00	52,00	60,00	0,00	C	107,39	SAME AS LAMP NUMBER 15
17	1056	158,00	31,00	121,00	30,00	0,00	C	69,86	SAME AS LAMP NUMBER 15
18	1056	158,00	35,00	121,00	30,00	0,00	C	69,86	SAME AS LAMP NUMBER 15

FIGURE 7

COMPUTER PRINT OUT NOMENCLATURE

LP # = LAMP NUMBER
TYPE = PHOENIX CONTROL NR.
LOC-X = LAMP LOCATION (X-COORDINATE) (FT.)
LOC-Y = LAMP LOCATION (Y-COORDINATE) (FT.)
HEIGHT = LAMP MOUNTING HEIGHT (FT.)
ELEV. = ELEVATION AIMING ANGLE (DEGREES)
AZIMUTH = AZIMUTH AIMING ANGLE (DEGREES)
IV = BEAM ORIENTATION INDICATOR (APPLIES TO PAR-TYPE LAMPS AND PARABOLIC REFLECTORS ONLY).
IV = 0, MAJOR BEAM AXIS IS PARALLEL TO DIRECTION OF BEAM CENTERLINE.
IV = 1, MINOR BEAM AXIS IS PARALLEL TO DIRECTION OF BEAM CENTERLINE.
FOR PAR TYPE LAMPS:
IV = 1, NORMAL LAMP POSITION
IV = 0, ROTATE LAMP 90°
FOR PARABOLIC REFLECTORS:
IV = 0, NORMAL FIXTURE POSITION
IV = 1, ROTATE FIXTURE 90°
R-VALUE = REFERENCE DISTANCE FROM NADIR POINT BELOW LAMP TO AIM POINT.
LAMP DESCRIPTION = FIXTURE MODEL; LAMP TYPE AND PPC PART NR.; BALLAST TYPE (IF REQ'D) AND PPC PART NR.

FIGURE 8



PLAN VIEW ILLUMINATION

FIGURE 9

WORK FACE ILLUMINATION

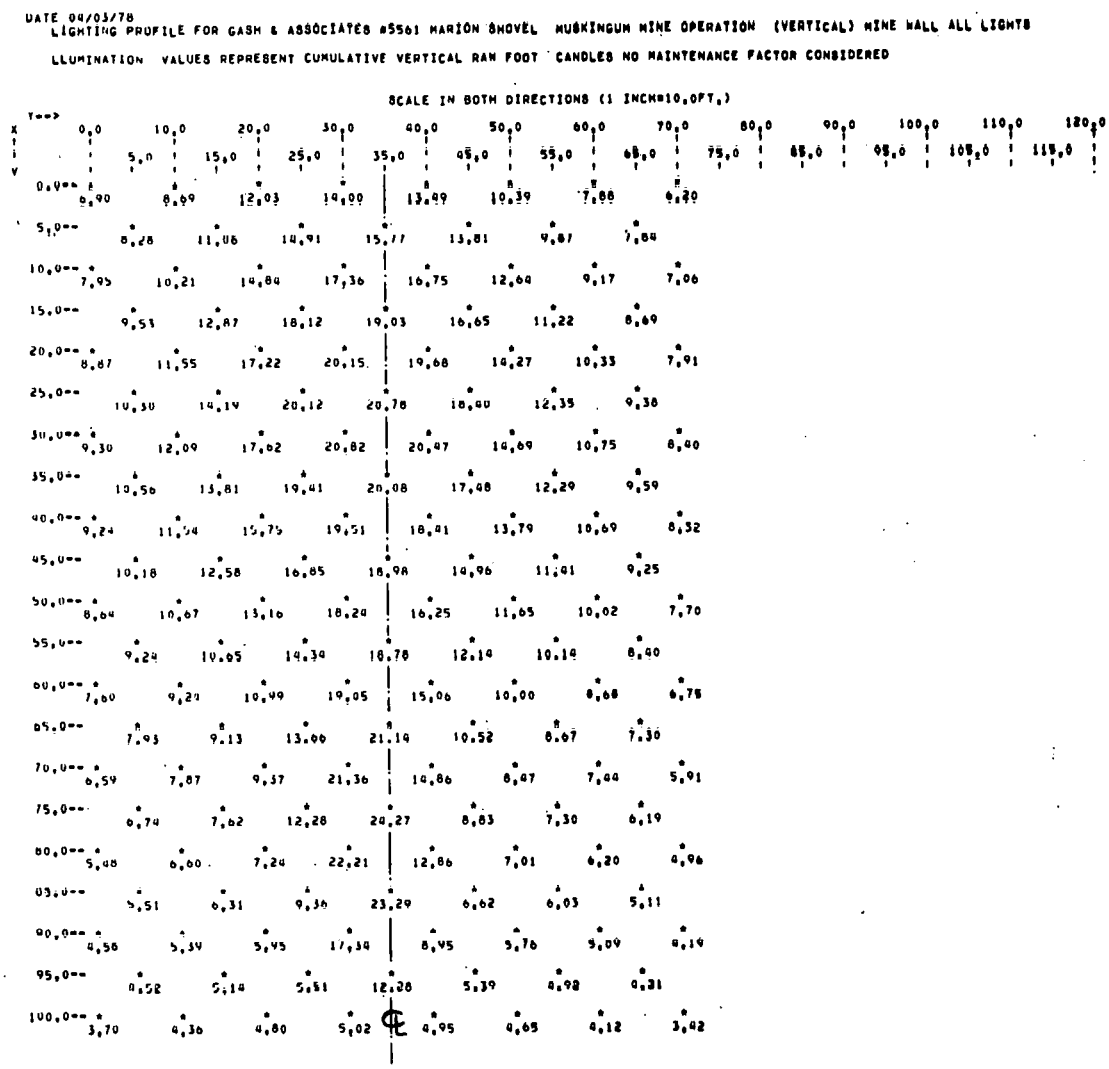


FIGURE 10

PROPOSED 30CFR REGULATIONS ILLUMINATION REQUIREMENTS
DRAGLINES, SHOVELS AND WHEEL EXCAVATORS

TABLE I

LOCATION	Average Illumination Intensity In Footcandles	*Uniformity Ratio (UR)	METHOD OF MEASUREMENT
1. All areas 20 ft. in all directions from the main frame.	5.0	10/1	Surface to be divided into fields of not greater than 100 sq. ft. Light sensitive cell shall be held in a horizontal plane approximately 30 inches from surface to be measured. Average illumination intensity for each field shall be the average of five readings; one taken at each of the four corners and one at the center.
2. All work or travel areas beneath the main frame.			
3. Interior areas except operating cabs.	10.0		
4. Exterior walkways, catwalks, and ladders on main frame and boom.	5.0	10/1	Average illumination intensity shall be average of at least four evenly spaced readings taken at intervals of 10 feet or less. Light sensitive cell shall be horizontal, approximately 30 inches from measured surface.
5. Interior walkways.	10.0		
6. Area beneath boom from 20 ft. from main frame to furthest point equipment can excavate or discharge material.	1.0	10/1	Average illumination intensity shall be average of at least four evenly spaced readings at intervals of 10 feet or less along the center line of the boom. The boom shall be placed over level terrain and the first reading shall be taken 20 feet ahead of the main frame.

*Uniformity ratio shall be calculated as the average to minimum illumination intensity within the 100 sq. ft. field (area).

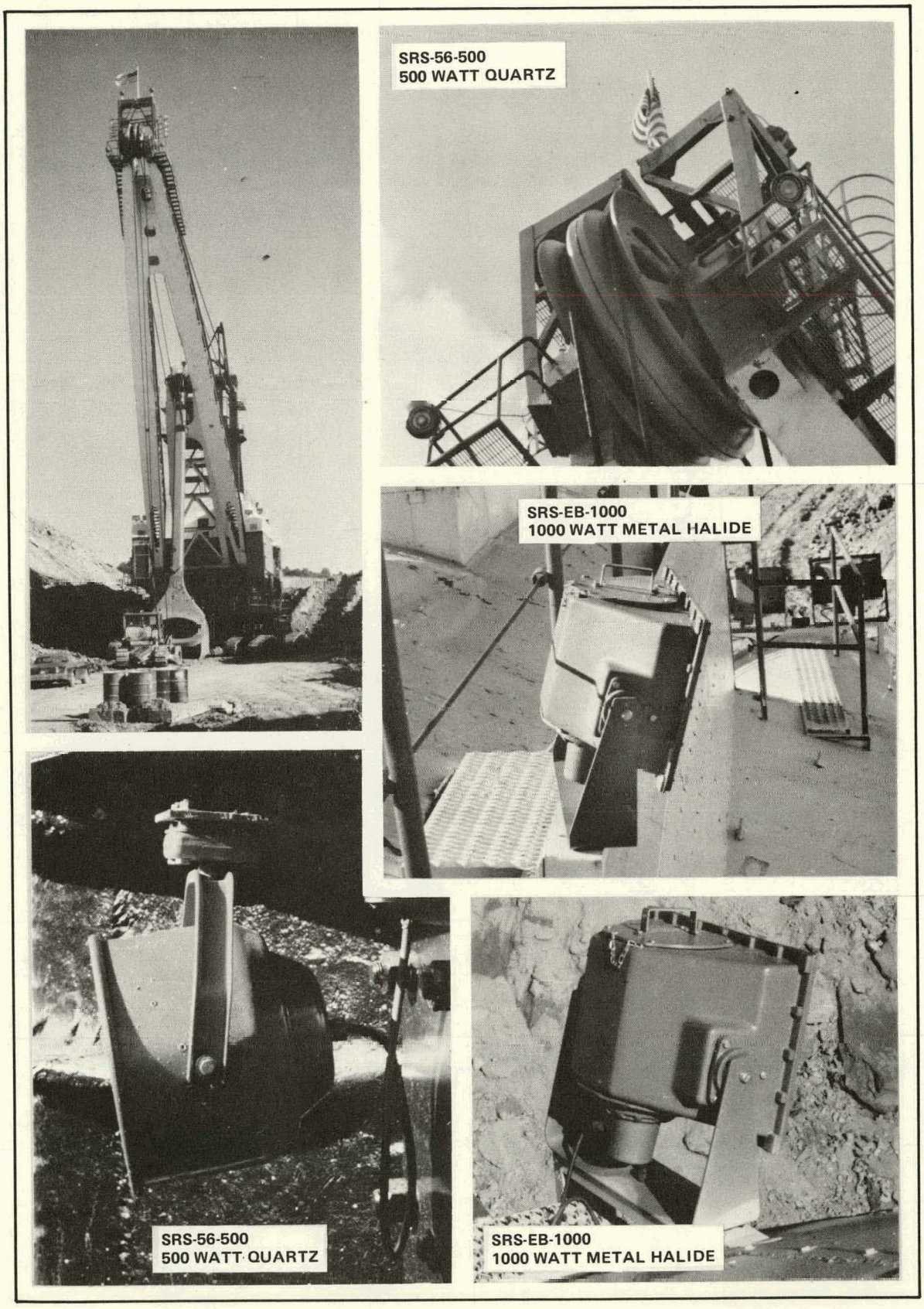

SRS-56-500
500 WATT QUARTZ
SRS-EB-1000
1000 WATT METAL HALIDE
SRS-56-500
500 WATT QUARTZ
SRS-EB-1000
1000 WATT METAL HALIDE

FIGURE 11

EVALUATION OF MINE ILLUMINATION SYSTEMS USING NUMERICAL MODELING

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Pittsburgh Mining and Safety Research Center
Bureau of Mines
U.S. Department of the Interior
Pittsburgh, PA

Robert Goldstein
Mathematical Applications Group, Inc.
Elmsford, NY

ABSTRACT

The analysis of a machine-luminaire configuration is a time-consuming process requiring the construction of a machine mockup, installation of a designated lighting arrangement, and an extensive set of illumination measurements. This entire process, however, is well suited to computer analysis. By simulating the machine geometry and the characteristics of the desired luminaires, the incident light levels can be calculated at the required set of points. This Bureau of Mines paper presents a general description of the techniques involved in performing this simulation.

INTRODUCTION

The Mine Safety and Health Administration (MSHA), U.S. Department of Labor, provides a method of evaluating an illumination system prior to its installation in an underground coal mine. This method requires the construction of a machine mockup that conforms to the shape and dimensions of the actual mining machine on which the lighting fixtures will be mounted. The designated luminaires are mounted on the mockup at their specified locations, with their proper orientations. The mockup, with its associated illumination system, is then centered in a simulated mine entry, which is a black room that excludes all exterior light. This room has movable roof and walls which can be adjusted to determine the maximum and minimum dimension of the mine entry in which the machine can be operated.

The illumination levels are determined by dividing the surfaces (face, roof, floor, and ribs) into square fields having areas of $4ft^2$. Incident light measurements are taken at the corners of each square by orienting the light sensor for a maximum reading. The average illumination for each square is determined by averaging the readings at its corners. The walls and the roof of the simulated entry are adjusted until the average illumination of each square field is at least 2fc.

It is obvious that the method of evaluation is a time-consuming process. As a result, the Bureau of Mines awarded a contract to the

Mathematical Applications Group, Inc. (MAGI) for the development of a computer simulation program for evaluating coal mine illumination systems. MAGI has previously developed computer techniques for simulating three-dimensional geometric structures to analyze geometry-dependent physical properties. Therefore, the goal of the project was to incorporate these computer techniques into a system for accurately predicting coal mine illumination levels.

A program of this nature must be able to perform the following functions:

a. Process a numerical model of a mining machine which is supplied as input.

b. Accept a specification of the types and locations of illumination sources.

c. Determine which luminaires contribute to the illumination at a specified set of points.

d. Record the indicent illumination levels at each detector point in an organized, easily interpreted format.

MACHINE MODELING TECHNIQUES

By experimentally determining the candlepower of a light source, one can easily predict the unobstructed illumination at a specific point by the Lambert cosine law:

$$E = \frac{I}{d^2} \cos \theta$$

This states that the illumination E is directly proportional to the candlepower I of the source and the cosine of the angle of incidence θ, and inversely proportional to the square of the distance d between the source and the point. However, in order to accurately predict the illumination provided by a set of luminaires at any point in a mine entry, one must also account for the effects of shadowing due to the machine itself. Thus, an accurate representation of the mining machine must be created.

The mining machine is represented by a numerical model. The model is created by a "building block" approach in which basic predefined shapes (boxes, cylinders, cones, etc.) are combined according to prescribed rules to form more complicated objectes. In constructing the model, each of these basic shapes is assigned specific dimensions and a location in a three-dimensional coordinate system. The totality of these forms then represents a complete numerical description of the desired model.

As an example, refer to Figure 1 for the construction of a simplified continuous miner model. The ripper head of the miner is modeled by a cylinder referred to as CYL 1. Its location is first defined by the x, y,

z coordinates at the center of either base. Thus, if the right base of the cylinder is chosen, its location would be defined by the point (4,1,1). The dimensions of the cylinder are then given by its axis vector and its radius. The axis vector is defined by the x y z coordinates, which give its magnitude and direction from the location point. Since the cylinder is 8 ft long with its axis parallel to the x axis in the negative direction, the axis vector is defined by (-8, 0, 0). The radius of the cylinder is 1 ft.

The cutter boom of the continuous miner is represented by a box, which is referred to as Box 2. It can be located by defining any corner of the box. If the corner indicated in Figure 1 is selected, its location is defined by the point (2.5, 3.5, 5). Its dimensions are then defined by three vectors which originate at the location point and give the magnitude and direction of the length, width, and height of the box. The length of the box is 2 ft in the negative y direction and 4 ft in the negative z direction. Thus, its length is defined by the coordinates (0, -2, -4). Since the box is 5 ft wide in the direction parallel to the nagative x axis, its width is defined by the coordinates (-5, 0, 0). Similarly, the height of the box is defined by (0, -1, 0.5).

The main frame and the discharge boom of the machine are modeled by Box 3 and Box 4, respectively. These boxes are given locations and dimensions by the same method used for Box 2. Thus, the entire input to the program for the machine model of Figure 1 would consist of the following data:

```
CYL 1:  Location = (4, 1, 1)
        Axis Vector = (-8, 0, 0)
        Radius = 1

BOX 2:  Location = (2.5, 3.5, 5)
        Length = (0, -2, 4)
        Width = (-5, 0, 0)
        Height = (0, -1, 0.5)

BOX 3:  Location = (3, 3.5, 17.5)
        Length = (0, 0, -12.5)
        Height = (0, -3.5, 0)

BOX 4:  Location = (1, 3.5, 23.5)
        Length = (0, 0, -6)
        Width = (-2, 0, 0)
        Height = (0, -1, 0)
```

These data are read by the program and stored for the calculation phase. The accuracy of the machine model can be verified by generating a line drawing of the model on a video display. A detailed model of a continous miner, which was generated by the numerical modeling process, is illustrated in figure 2.

LUMINAIRE MEASUREMENTS

Experimental measurements must be made for a representative sample of each type of luminaire. These measurements determine the luminaires' isointensity characteristics in order to define an average luminaire for each type. The candlepower is determined as a function of the angular light distribution of the luminaire. This is accomplished by positioning a photometer at a fixed location and varying the orientation of the luminaire by the use of a goniometer. Some luminaires are equipped with a protective metal cage. The measurements are taken with the cage in place to account for its effects on the variation in light output. Incident light measurements (in foot-candles) are made for every 10 degrees of orientation, resulting in a spherical, latitude-longitude arrangement. These data are then reduced to define the candlepower of the luminaire for each angular location by

$$I = E\ d^2$$

where

E = incident light measurement (fc)
d = distance from the photometer to the luminaire (ft)

These data are then stored in a luminaire library.

LUMINAIRE LOCATION AND ORIENTATION

Once the luminaire types are modeled and stored in the luminaire library, only the types of the luminaires desired and their locations and orientations on the machine need to be specified. Figure 3 shows two machine lights (Type 1) and two headlights (Type 2) mounted on the model of a continuous miner. The location of a luminaire is defined by x y z coordinates, using the same coordinate system utilized in the machine modeling process. Thus, the location of the center of luminaire 1 is defined by the point (-2.5, 3.75, 15.5). Each luminaire is assigned an aim and an axis vector which define its orientation on the machine. Since the axis vector of luminaire 1 is directed toward the face, parallel to the z axis in the negative direction, it is defined by the coordinates (0, 0, 1). Its aim vector is directed toward the roof parallel to the positive y axis; therefore, the aim vector is defined by the coordinates (0, 1, 0). The entire input that defines the locations and orientations of the four luminaires illustrated in Figure 3 would consist of the following:

```
LUM 1:  Type = Type 1
        Location = (-2.5, 3.75, 15.5)
        Aim = (0, 1, 0)
        Axis = (0, 0, -1)

LUM 2:  Type = Type 2
        Location = (-2, 3.75, 7)
        Aim = (0, 0, -1)
        Axis = (0, 0, -1)
```

LUM 3: Type = Type 1
Location = (2.5, 3.75, 15.5)
Aim = (0, 1, 0)
Axis = (0, 0, -1)

LUM 4: Type = Type 2
Location = (2, 3.75, 7)
Aim = (0, 0, -1)
Axis = (0, 0, -1)

The location of the luminaires can also be defined by an alternate method. This requires displaying the machine model on the graphics terminal and using movable crosshairs to define the luminaire's position on the model. However, the aim and the axis vectors must still be defined.

Prior to starting the illumination calculations, the program determines the validity of the locations of the luminaires. It tests whether any point on the axis of each luminaire lies outside the mine entry or inside any geometric shape comprising the machine model. If either case occurs, the luminaire has been improperly located, and the program is terminated.

Is should also be noted that high-intensity discharge lamps and incandescent lamps can be treated as point emitters owing to their compact sources of illumination. However, the point source approximation for fluorescent luminaires becomes invalid owing to the length of the arc tube. Therefore, the program automatically divides the tube into smaller segments and treats such segment as a point emitter.

ILLUMINATION CALCULATIONS

Once the luminaires are positioned on the machine model and the dimensions of the mine entry are specified, the illumination calculations begin. As an example, assume three luminaires located on a model of a continuous miner and a detector point D located on the right rib, as illustrated in figure 4a. The calculations proceed in the following manner:

(1) The distance d from luminaire 1 to the detector point D is calculated by

$$d_1 = (x_1 - x_D)^2 + (y_1 - y_D)^2 + (z_1 - z_D)^2$$

where x_1, y_1, z_1 are the coordinates that define the location of luminaire 1 and x_D, y_D, z_D are the coordinates that define the location of the detector point. The distances d_2 and d_3 are then determined in a similar fashion.

(2) Since the location of the luminaires with respect to the detector point is known, the candlepowers (I_1, I_2, and I_3 of the luminaires are determined from the prestored angular distribution data.

(3) The program tests whether the candlepower of any luminaire is equal to zero; if so, the illumination contribution E of the luminaire is assigned the value of zero. If the candlepower of the luminaire is greater than zero, the illumination contribution of the luminaire is calculated ignoring the cosine correction, by

$$E = \frac{I}{d^2}$$

Since luminaire 3 is a headlight which is oriented toward the face, the detector point D does not lie within the angular distribution pattern of the light as shown in figure 4b. Thus, the prestored angular distribution tables would contain the value of zero for luminaire 3 in this particular situation. Therefore, E_3 is assigned the value of zero. Since point D is within the angular distribution patterns of luminaires 1 and 2, the illumination contribution of each luminaire is calculated as follows:

$$E_1 = \frac{I_1}{d_1^2} \text{ and } E_2 = \frac{I_2}{d_2^2}$$

(4) To determine the possible effects of machine shadowing, a ray tracing is performed for each luminaire whose illumination contribution E is greater than zero. A ray is traced from the location point of each applicable luminaire to the detector point. If a ray intersects any component of the machine model, the contribution of the luminaire is assigned the value of zero. Thus from figure 4c, E_1 is set equal to zero.

(5) As mentioned earlier, the present method of measurement requires the use of a cosine-corrected photometer. These measurements are made by orienting the probe in a trial and error fashion until the maximum reading is obtained. The reading thus displays the summation of the illumination contributions of each luminaire, taking into account the cosine of their angles of incidence.

To duplicate this procedure, the detector point D is surrounded by an imaginary hemisphere which is divided into degress of latitude and longitude. Each pair of angles determines a possible orientation of the axis of the detector probe. For a given axis direction, the program computes the angle of incidence θ for each luminaire and the following quantity:

$$E_D = \sum_n E_n \cos \theta_n$$

This procedure is repeated for each latitude-longitude pair, which results in approximately 200 orientations being tested. Thus, the program scans through a series of angles as shown in figure 4d. For the sake of simplicity, the diagram is illustrated in a two-dimensional mode. Since luminaire 2 is the only luminaire that contributes to the illumination at the detector point D, the maximum reading is obtained with the

normal of the probe aimed directly at this luminaire. Therefore, the angle of incidence is equal to zero, and the maximum illumination at point D is equal to E_2.

After all the detector points on a given entry surface have been completed, the averages within each 2-ft by 2-ft square are computed and the entire matrix of results is printed. The program then repeats the entire process until all five entry surfaces have been covered. The output format for a 6-ft-high by 12-ft-wide coal face is illustrated in figure 5. Any average illumination level that falls below the required level of 2 fc is flagged with an asterisk for easy identification.

RESULTS

To verify the accuracy of the illumination calculations, simple structures were mocked up, consisting of cardboard boxes placed on the floor of a simulated mine entry. Readings were taken on a 2-ft by 2-ft grid to coincide with the computer output pattern. The computed results from one of these tests are shown in figure 6. The calculations were in good agreement with the experimental readings, with differences generally less than 8 percent. A few scattered locations, especially those at low illumination levels, showed larger differences. Other tests at MSHA's Illumination Laboratory produced similar results.

Experimental measurements for the various types of luminaires have shown that their deviations from the average light output can vary as much as 10 percent. Thus, the computer simulation provides a more standard means of evaluation than present mehtods, since the average light output for each type of luminaire is used for the calculations.

The computer simulation program will significantly decrease the time involved with the present method of evaluating mine illumination systems. Computer-generated models of mining equipment, with sufficient detail to produce accurate results, can be created in approximately 2 hours, as opposed to 8 hours for the construction of a less detailed machine mockup. The numerical models can be stored indefinitely and retrieved as needed for future evaluations, whereas a mockup must be constructed for each different evaluation owing to storage and material limitations. Modifications to existing models (the addition or removal of a canopy, etc.) can be accomplished without having to recreate a new model.

CONCLUSION

The computer simulation program described should be a viable alternative to experimental measurements for evaluating mine illumination systems. Once the machine is numerically modeled and the characteristics of the desired luminaires are determined, the user need only specify the locations and orientations of the luminaires on the model to accurately predict the incident light levels on all mine surfaces. The program accounts for the effects of shadowing by tracing simulated light rays from each luminaire to an array of detector points, and the output of the illumination levels is recorded in an organized and easily interpreted format.

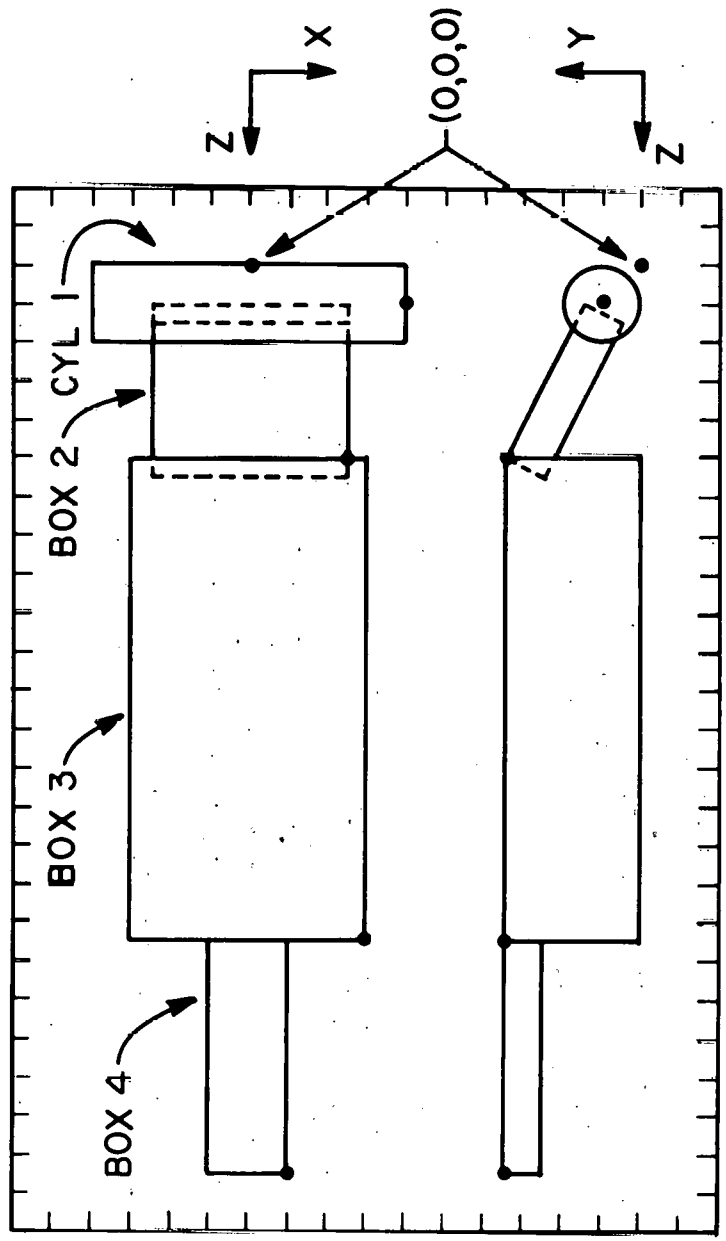


FIGURE 1.--Simplified model of a continuous miner

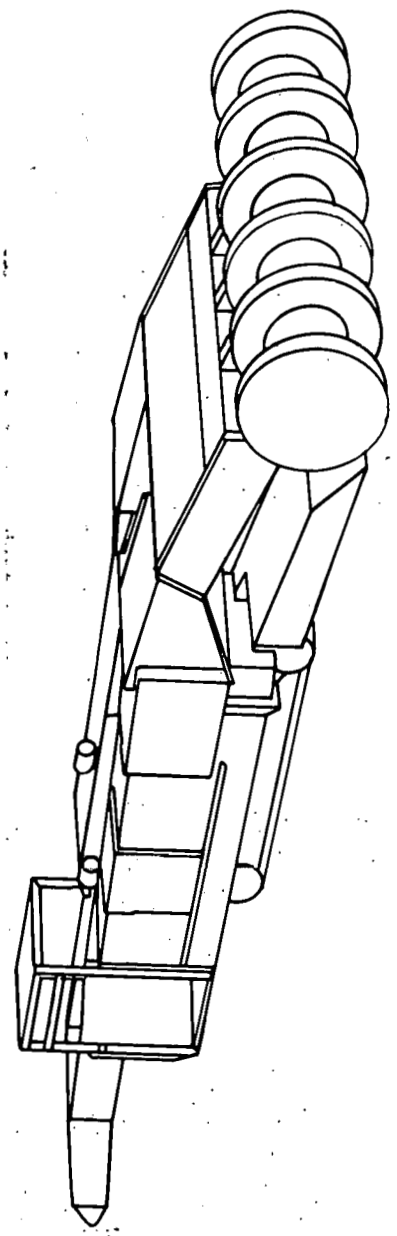


FIGURE 2.--Actual computer generated line drawing of a continuous miner

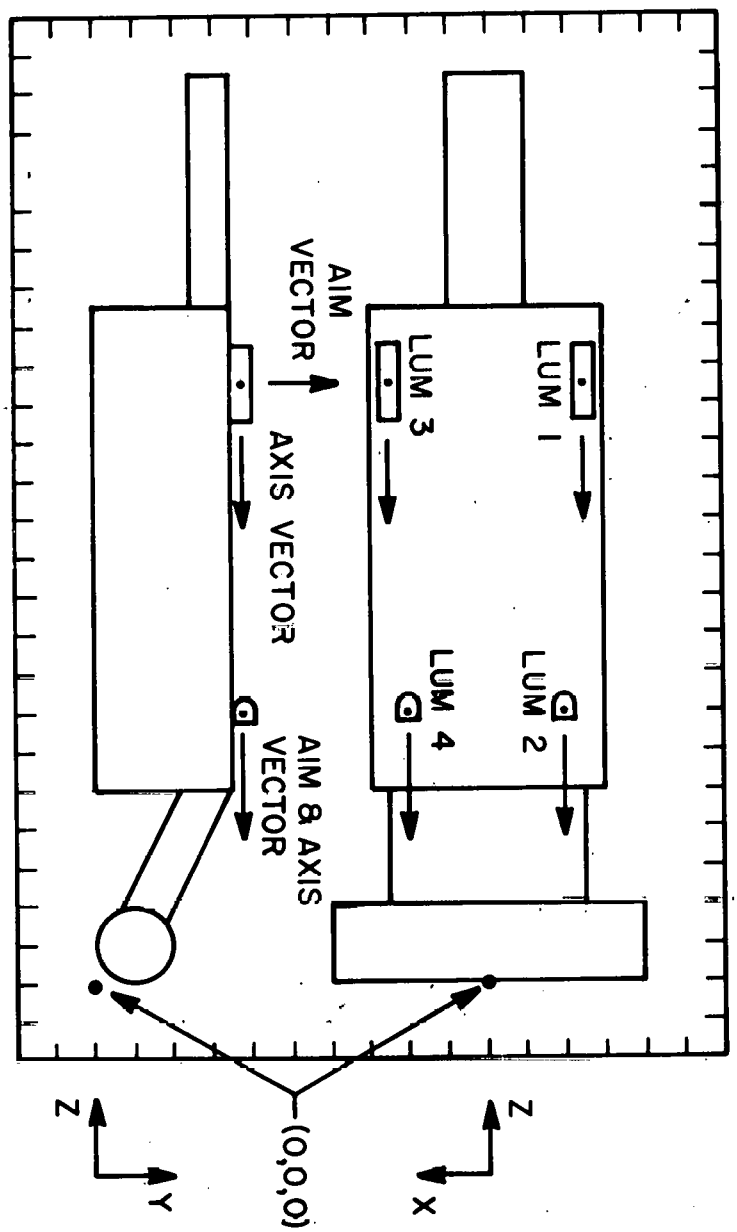


FIGURE 3.--Location of luminaires on the machine mockup

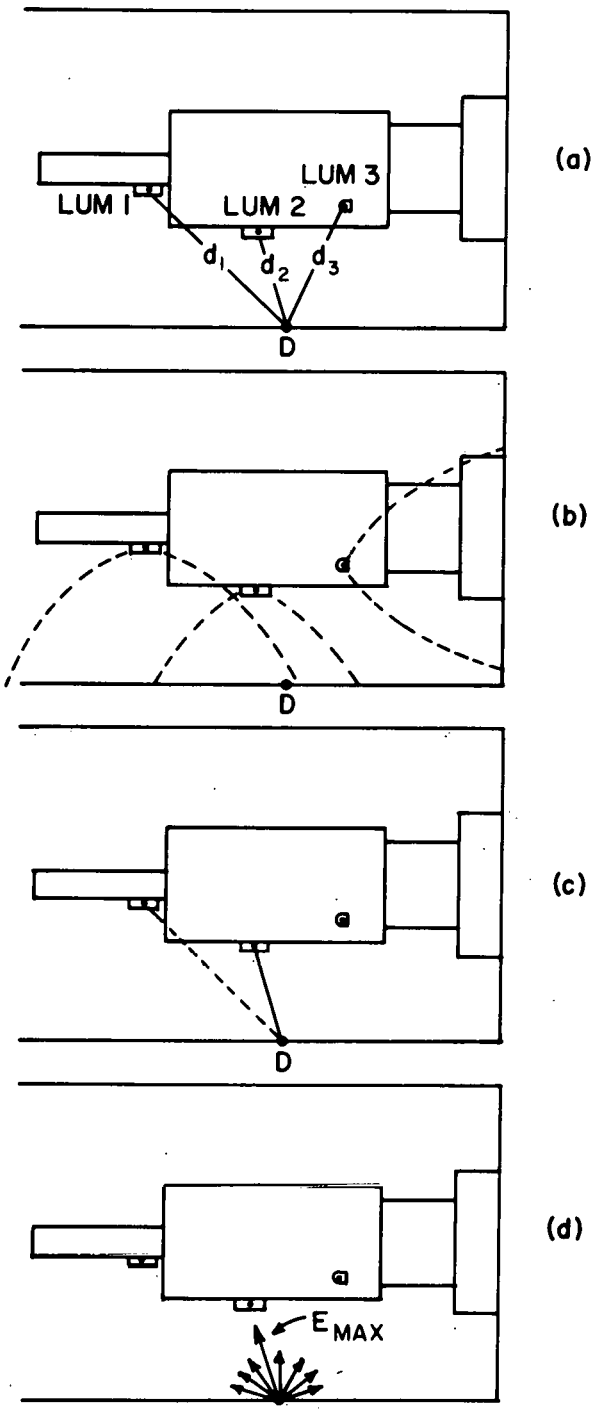
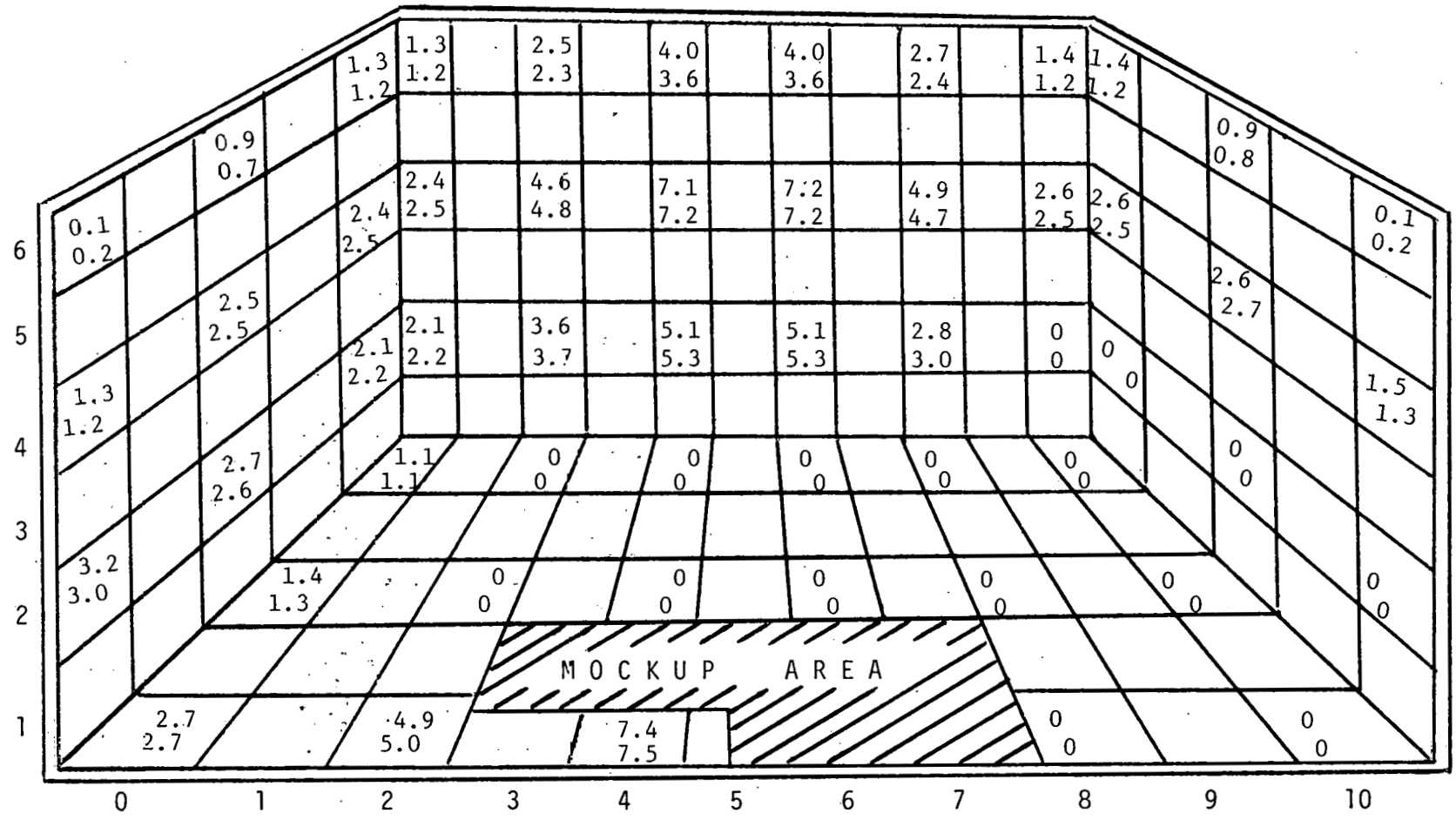


FIGURE 4.--Illumination calculations

F A C E

	0	2	4	6	8	10	12
	1.0	1.2	2.8	4.8	2.7	1.1	1.1
Avg.	1.2*	2.1	3.9	3.9	2.0	1.2*	
	1.2	1.4	3.0	5.0	2.9	1.1	1.3
Avg.	1.4*	2.3	4.2	4.2	2.2	1.4*	
	1.4	1.5	3.2	5.6	3.1	1.4	1.5
Avg.	1.3*	2.3	4.2	4.1	2.2	1.3*	
	1.0	1.3	3.0	4.9	2.9	1.3	1.1

FIGURE 5.--Output format



Upper number is computed (fc); lower number is measured (fc)

FIGURE 6.--Calculated and measured results

ILLUMINATION REGULATIONS AND METHODS

OF DETERMINING COMPLIANCE

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ABSTRACT

The underground illumination regulations, Title 30, Code of Federal Regulations, Sections 75.1719 through 75.1719-4, define illumination requirements, the surfaces or areas to be illuminated and the minimum acceptable levels of illumination in the working places of underground coal mines. Revisions to the procedures used to issue Statements of Test and Evaluation (STE's) were published in the Friday, December 30, 1977, Federal Register, Volume 42, No. 251, which permitted the use of diffusers or louvers on lighting fixtures to minimize discomfort glare. The revisions also adopted a new assumed reflectance value for underground coal surfaces and amended the formula for determining luminous intensity and incident light. The effective date of the underground coal mine illumination regulations was extended from April 1, 1978, to July 1, 1978, to allow additional time for compliance.

INTRODUCTION

By Section 317(e) of the Coal Mine Health and Safety Act of 1969, the Congress of the United States required the Secretary of the Interior to establish minimum illumination standards for the working places of underground coal mines. These standards are now Sections 75.1719 through 75.1719-4 of Title 30, Code of Federal Regulations. These Sections may be summarized as follows:

Section 75.1719-1 prescribes illumination to be provided in the working places of underground mines.

Section 75.1719-2 specifies the requirements for lighting fixtures.

Section 75.1719-3 outlines the methods of measuring light to determine compliance.

Section 75.1719-4 specifies requirements for mining machines, hard hats and cap lamps.

SECTION 75.1719-1 ILLUMINATION IN WORKING PLACES

Illumination shall be provided in each working place of underground coal mines while self-propelled mining equipment is being operated in the working place and people are working in such places. Self-propelled mining equipment is defined as equipment that possesses the capability of moving itself or its associated components from one location to another by electric, hydraulic, pneumatic or mechanical power supplied by a source located on the machine or transmitted to the machine by cables, ropes or chains. This illumination shall be in addition to that provided by personal cap lamps and will require all surfaces in the miner's normal field of vision to have a luminous intensity of at least 0.06 footlamberts.

The type of self-propelled mining equipment dictates the areas which shall be illuminated in the working place. The following is a breakdown of areas to be illuminated based upon machine type:

1. Continuous Miners and Coal-Loading Machines (see figure 1)

   Section 75.1719-1(e)(1) requires that the following areas be illuminated while continuous miners and coal-loading machines are operated in the working place:

   a. The face, and

   b. The ribs, roof, floor and exposed surfaces of mining equipment from the face to the outby end of the machine's bumper when shuttle car haulage is used or to the first transfer point when continuous conveyor haulage is being used.

2. Auger-Type Continuous Miners (see figure 2)

   The application of Section 75.1719-1(e)(1) to auger-type continuous miners was reconsidered for the following reasons:

   a. The glare encountered from systems designed to light up all the required surfaces for the complete entry width.

   b. Lighting systems designed to illuminate required surfaces the complete entry width interferred drastically with the miner's necessary visual communication.

   c. The vast majority of auger-type continuous miners utilize roof-support plans requiring two or more rows of posts on each side of the machine. Section 75.1719-3(b)(8) prohibits measurements where

Figure No. 1. Loader or continuous miner. The area to be illuminated is from the face to the outby end of the bumper.

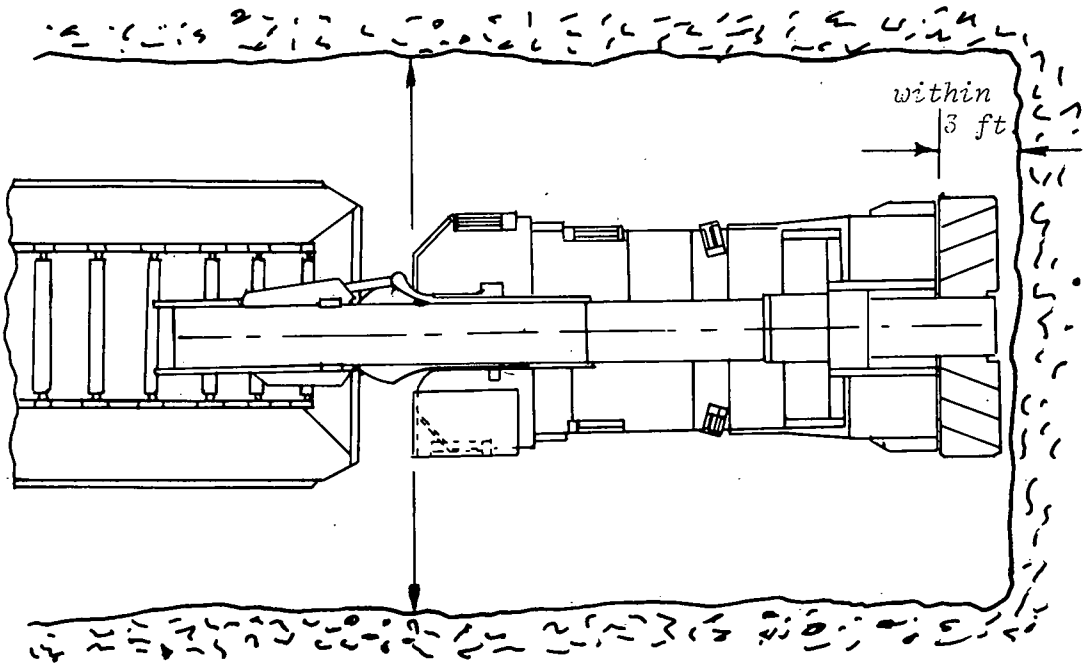
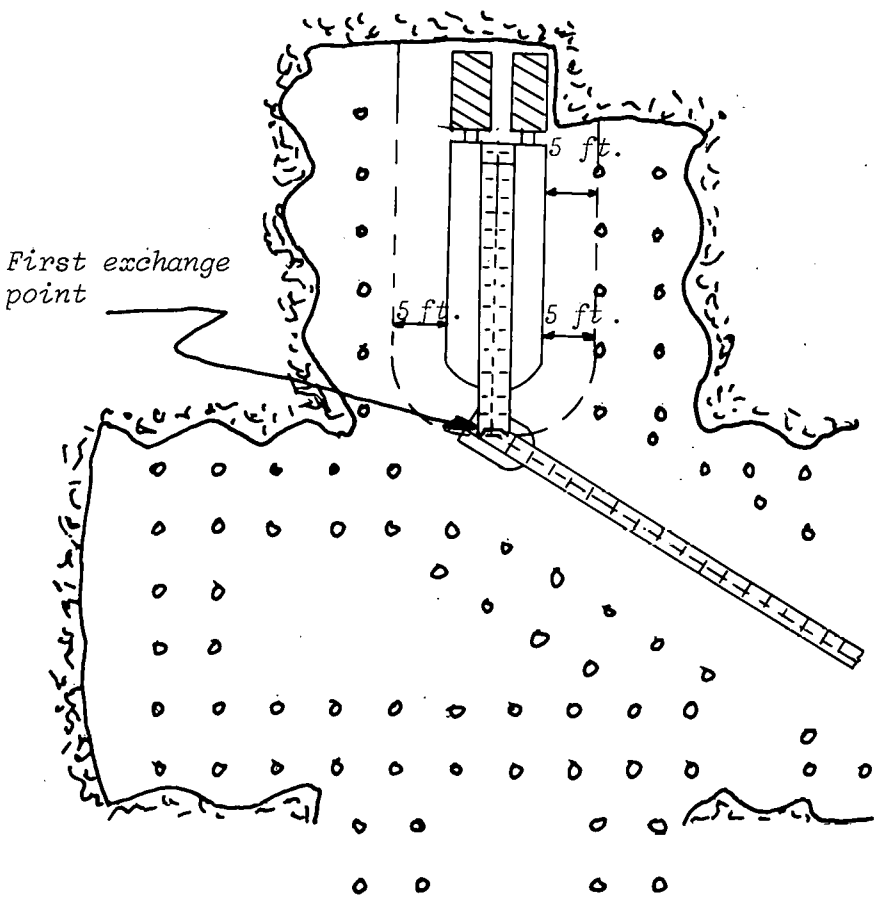


Figure No. 2. Auger-type continuous miner.



shadows are cast by roof-support posts. The taking of accurate measurements of unshaded rib surfaces would be impractical.

The areas that shall be illuminated while auger-type continuous miners are being operated in the working place shall be limited to the face, rib, roof, floor and exposed surfaces of mining equipment which are within an area, the perimeter of which is a distance of 5 feet from the sides of the machine when measured parallel to the floor. The area that is required to be illuminated shall extend from the face to the first coal transfer point.

3. Self-Loading Haulage Equipment Used As A Loading Machine (see figure 3)

Section 75.1719-1(e)(3) requires that the following areas be illuminated while self-loading haulage equipment is used as a loading machine in the working place:

a. The face, and

b. The ribs, roof, floor and exposed surfaces of mining equipment that are between the face and a point 5 feet outby the machine.

The term "self-loading haulage equipment" refers to scoops, loading, hauling, dumping machines, etc. The requirements of Section 75.1719-1(e)(2) apply when the self-loading haulage equipment is used to load coal or other material in the working place. Self-loading haulage equipment which is utilized only for haulage, cleanup or transporting supplies in the working place shall meet the requirements of Section 75.1719-1(e)(6).

4. Cutting Machines and Coal Drills (see figure 4)

Section 75.1719-1(e)(3) requires that the following areas be illuminated while cutting machines and coal drills are operated in the working place:

a. The face, and

b. The ribs, roof, floor and exposed surfaces of mining equipment that are between the face and a point 5 feet outby the machine.

5. Shortwall And Longwall Mining Equipment

Section 75.1719-1(e)(4) requires that the following areas be illuminated while shortwall and longwall mining equipment is operated in the working place:

Figure No. 3. Scoop that is used as a loading machine. Section 75.1719-1(e)(2)

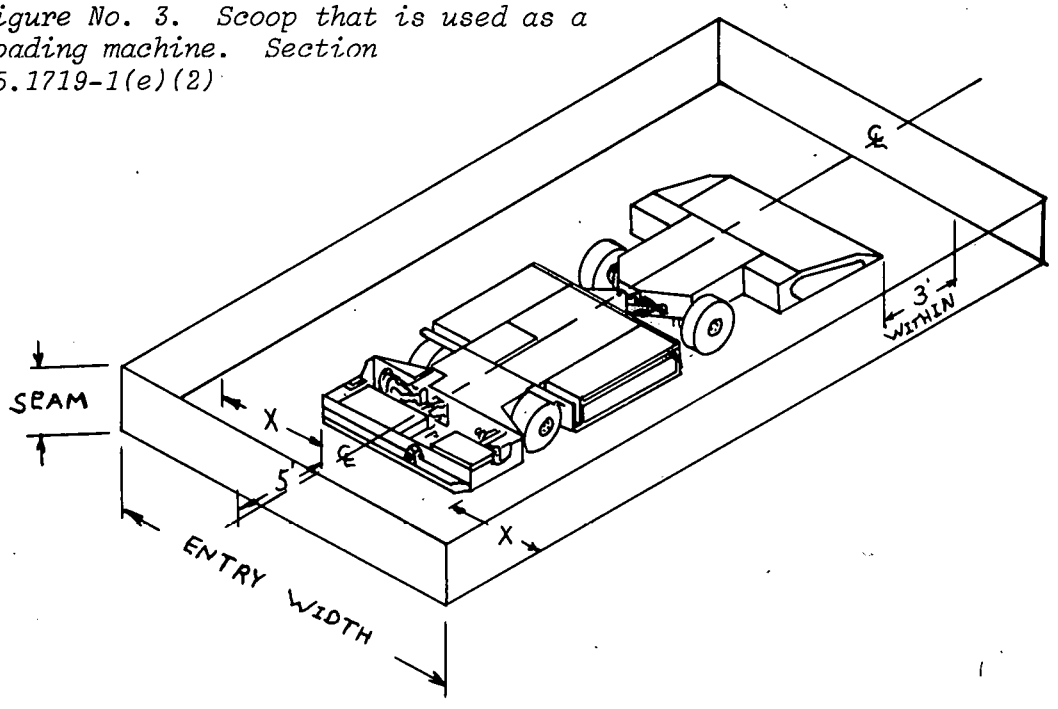
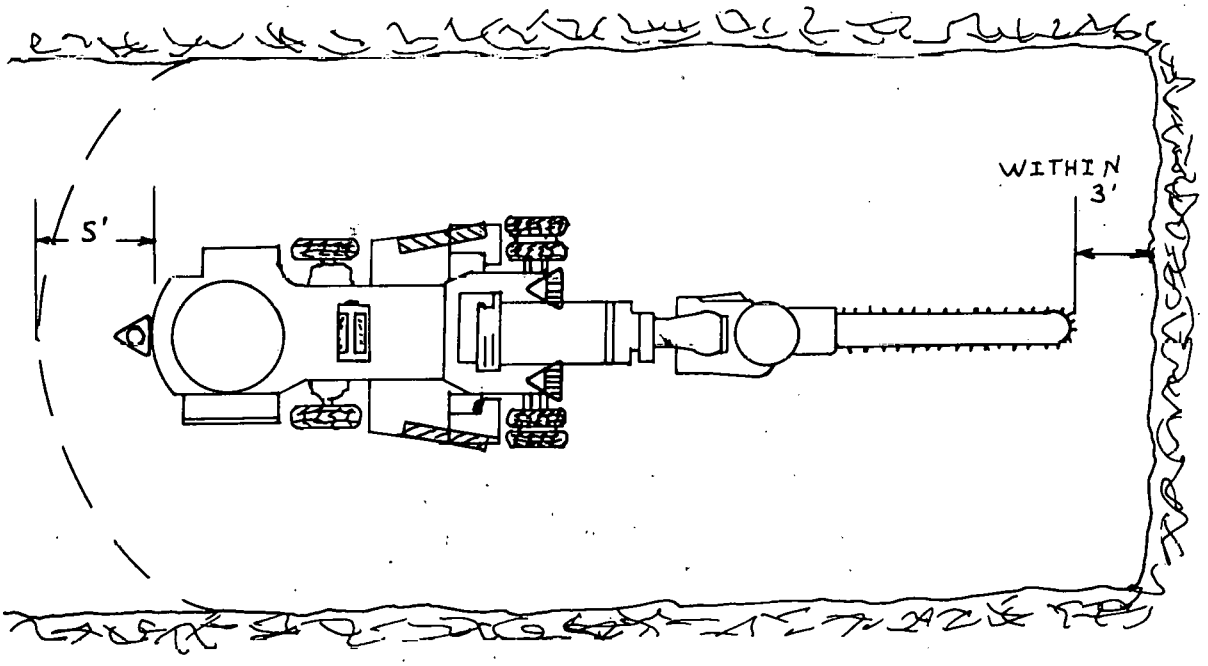


Figure No. 4. Cutting machine. The area to be illuminated is from the face to 5 feet outby the machine.



a. Longwall Mining Equipment (see figures 5, 6 and 7)

(1) The face, and

(2) The area between the gob-side of the travelway and the side of the block of coal from which coal is being extracted for the entire length of the self-advancing roof-support system, and

(3) The control station, and the headpiece and tailpiece of the face conveyor, and

(4) The roof and floor for a distance of 5 feet horizontally from the control station, headpiece and tailpiece.

b. Shortwall Mining Equipment (see figures 8 and 9)

(1) The area between the gob-side of the travelway and the side of the block of coal from which coal is being extracted for the entire length of the self-advancing roof-support system as specified in Section 75.1719-1(e)(4).

(a) The areas to be illuminated as required by Section 75.1719-1(e)(4) would include the face, roof, floor and exposed surfaces of mining equipment between the face and the gob side of the designated travelway. This illumination may be accomplished by stationary lighting only as shown in figure 8.

(b) The required areas of Section 75.1719-1 (e)(4) in shortwall mining systems utilizing continuous mining machines may be illuminated as follows:

i. For the section of the shortwall from which coal is to be extracted, the illuminated area is between the gob-side of the travelway and the face.

ii. The continuous mining machine will have machine-mounted lighting fixtures installed in accordance with the requirements of Section 75.1719-1(e)(1).

iii. For shuttle-car haulage systems, the areas around the shuttle car shall be illuminated in accordance with Section 75.1719-1(e)(6).

Figure No. 5. Longwall System. The area to be illuminated is from the face to the gob-side of the travelway.
Designated Travelway "A", the area to be illuminated is A'
Designated Travelway "B", the area to be illuminated is B'

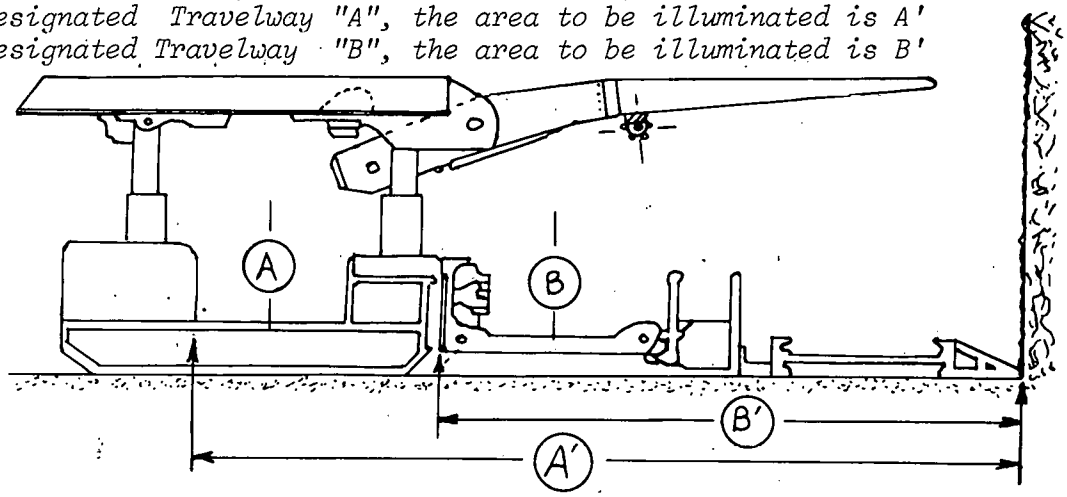
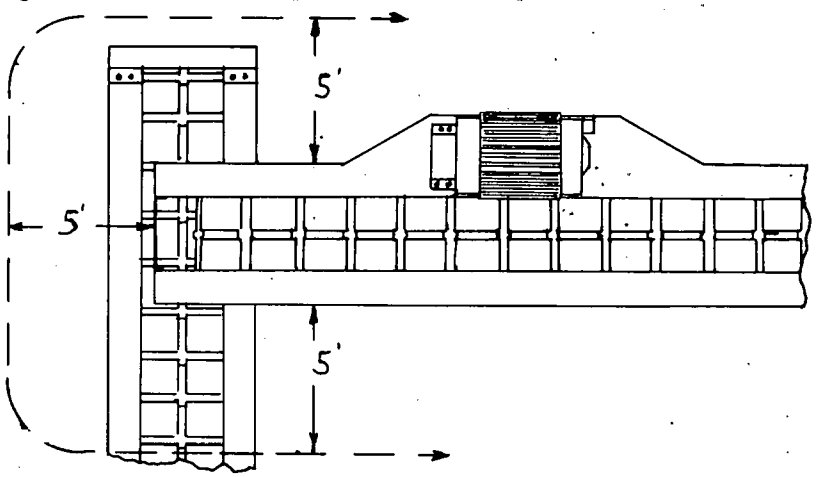


Figure No. 6. Headpiece and tailpiece.



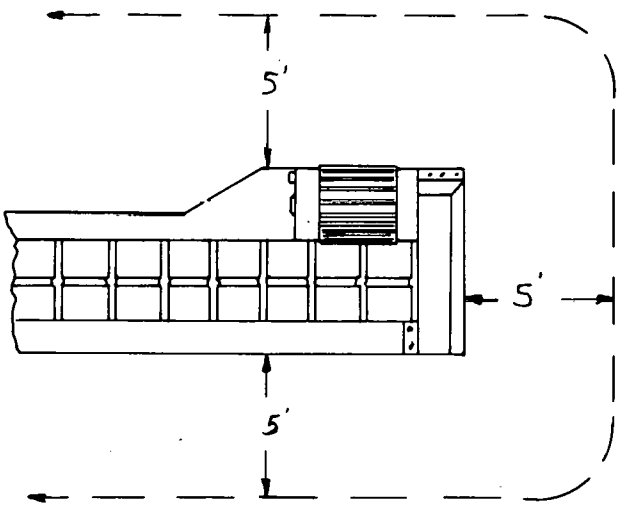


Figure No. 7. Control station.

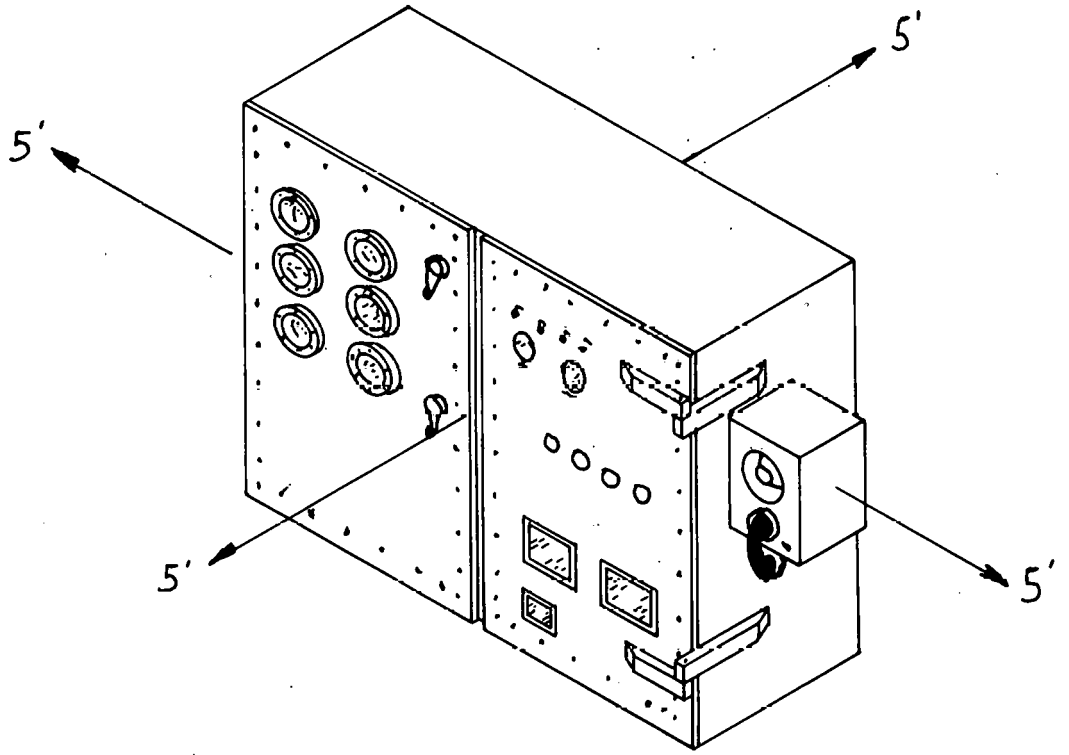
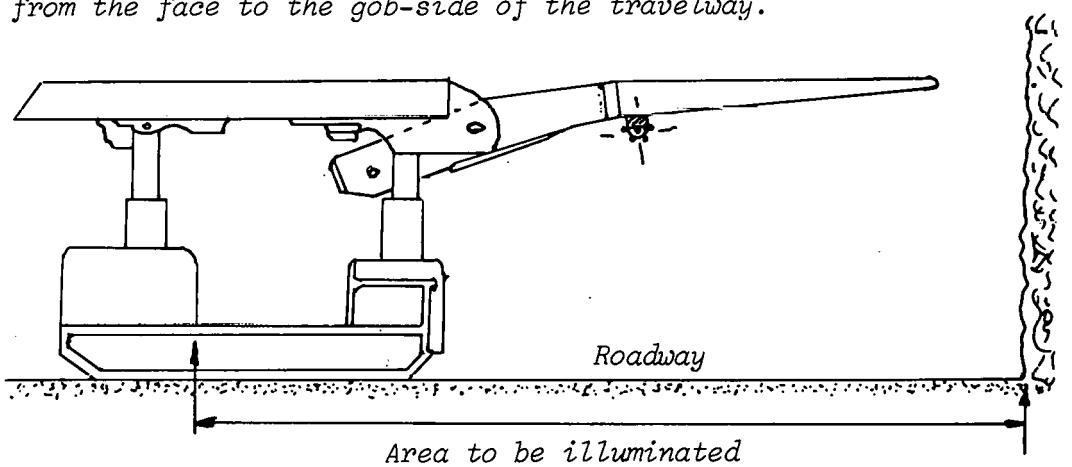


Figure No. 8. Shortwall system. The area to be illuminated is from the face to the gob-side of the travelway.



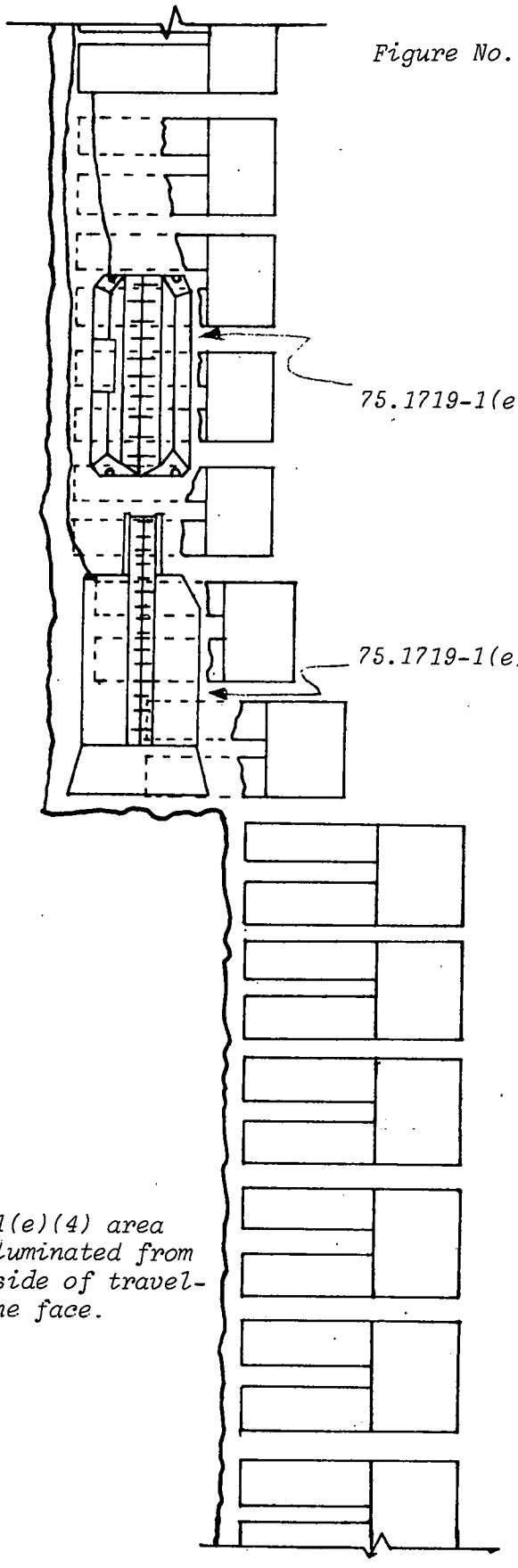


Figure No. 9. Shortwall (plan view)

6. Roof-Bolting Machines (see figure 10)

Section 75.1719-1(e)(5) requires that the following areas be illuminated while roof-bolting machines are operated in the working place:

a. In working places 5 feet or less in height, the face, ribs, roof and exposed surfaces of mining equipment shall be illuminated when such surfaces are within a distance of 5 feet from the machine when measured parallel to the floor.

b. When the height of the working place exceeds 5 feet, the face, ribs, roof, floor and exposed surfaces of mining equipment are to be illuminated when such surfaces are within a distance equal to a height of the working place from the front and sides of the machines. In all cases, surfaces within 5 feet of the rear of such machines shall be illuminated.

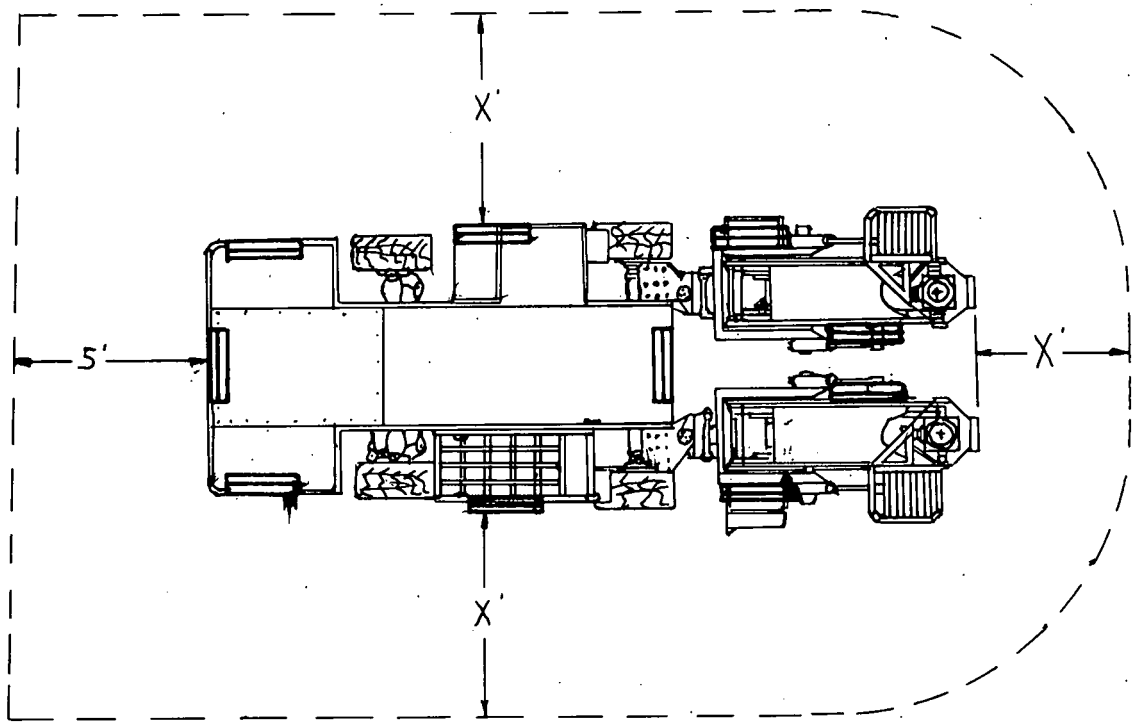
7. Other Self-Propelled Equipment (see figure 11)

Section 75.1719-1(e)(6) specifies the areas that must be illuminated when self-propelled mining equipment, other than equipment specifically covered under Sections 75.1719-1(e)(1) through 75.1719-1(e)(5), is operated in the working place. "Other self-propelled equipment" would include shuttle cars, battery tractors, maintenance vehicles, trailers pulled by tractors, etc. The requirements of Section 75.1719-1(e)(6) are as follows:

a. Lighting fixtures shall be installed on each piece of self-propelled equipment which will illuminate the face or rib coal surfaces within 10 feet of the front and the rear of the machine to a surface brightness of not less than 0.06 footlamberts.

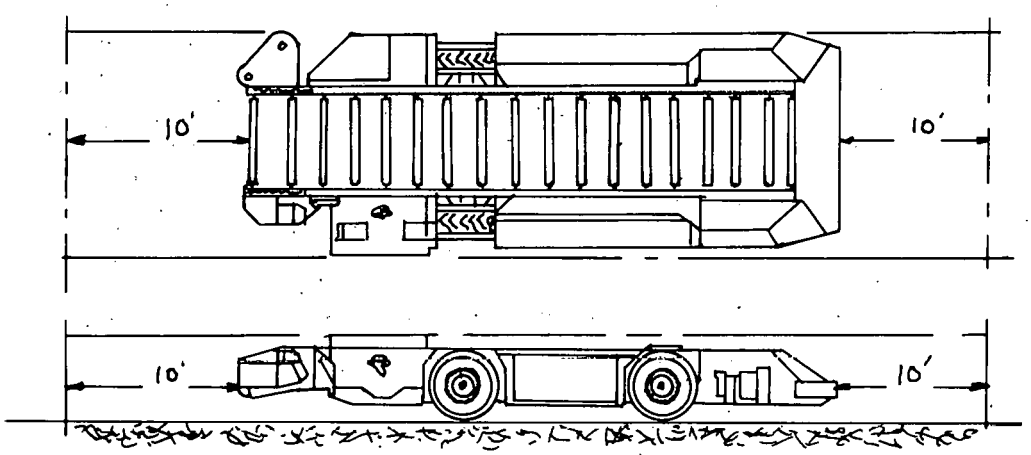
b. The height and width of the coal surface to be illuminated as required by Section 75.1719-1(e)(6)(ii) shall be equal in both directions of travel and directly in front of the equipment. The height of the coal surface shall be determined by the maximum height of the shuttle car or other self-propelled equipment which would include sideboards or canopies. The width of the coal surface shall be determined by the maximum width of the shuttle car or other self-propelled equipment which would include bumpers or tires.

Figure No. 10. Roof-bolting machine.



(1) Seam height is 5 feet or less x = 5 feet.
(2) Seam height is greater than 5 feet x = seam height.

Figure No. 11. Other self-propelled equipment. The area to be illuminated is face or rib surfaces within 10 feet of the front and rear of the machine. The height and width of the area of the coal surface which shall be illuminated shall equal the maximum height and width of the machine on which the luminaires are installed.



c. The lights in the direction of travel shall be operated at all times when equipment is being operated in the working place.

SECTION 75.1719-2 LIGHTING FIXTURES REQUIREMENTS

All lighting fixtures shall be permissible and either machine-mounted or stationary.

1. Machine-Mounted Lighting Fixtures

Machine-mounted fixtures are mounted on board a machine and receive power from the machine. These types of fixtures shall be electrically grounded to the frame of the machine as follows:

a. The intent of Section 75.1719-2(c)(3) is to ensure that machine-mounted lighting fixtures remain effectively grounded when the fixtures become loosened or completely separated from the machine frame. In applications for STE's, MSHA will accept a flexible shunt in lieu of a grounding conductor contained in the power cable for existing machine-mounted luminaires, providing the installation conforms to the following criteria:

(1) The shunt is a minimum of a No. 6 AWG jacketed cable that has a minimum of 125 strands.

(2) The connection points of the grounding shunt shall be independent of the luminaire-mounting base.

(3) The length of the grounding shunt is sufficient to ensure that the power cable to the lamp would be severed first when the lamp is detached from the machine. The minimum length of shunt shall in all cases provide a 6-inch movement of the lamp in any direction when detached from the machine.

b. All lighting fixtures will be considered electrically grounded to the machine when the installation meets the following requirements:

(1) The grounding conductor is an integral part of the power cable extending to the lighting fixture.

(2) The grounding conductor shall be continuous from the grounding post inside the lighting fixture to a grounded point inside the explosion-proof enclosure, from which the machine-mounted lighting system receives its power. A lighting fixture will not be acceptable for establishing the grounded point of a lighting circuit.

(3) The explosion-proof enclosure that establishes the grounded point of the lighting circuit shall be securely bolted or welded to the machine.

2. Stationary Lighting Fixtures

Stationary lighting fixtures are attached to the mine roof, ribs or roof-support equipment, such as chocks or shields on longwalls.

a. Since stationary lighting fixtures will be subject to contact by miners while energized (often by miners standing or crawling on wet mine floors), it is of paramount importance that power systems supplying power to stationary lighting fixtures be designed so that additional hazards to miners are not created. Consequently, the voltage of circuits supplying power to such lighting fixtures is restricted to 70 volts to ground for alternating-current circuits (Section 75.1719-2 (c)(2) and 300 volts nominal circuit voltage for direct-current circuits (Section 75.1719-2(d)).

b. Section 75.1719-2(c)(2) also requires that alternating-current circuits, energized at 100 volts or more and used to supply power to stationary lighting fixtures, originate at a transformer having a center or neutral tap grounded through a proper grounding resistor which shall be designed to limit fault current to not more than 5 amps. A grounding circuit shall originate at the grounded terminal of the grounding resistor, extend along with the power conductors, and serve as a grounding conductor for the frames of all equipment receiving power from the circuit.

c. Section 75.1719-2(e) requires that cables conducting power to stationary lighting fixtures, other than intrinsically safe devices, be considered trailing cables. Section

75.1719-2(e) requires that trailing cables conducting power to stationary lighting fixtures be protected against short circuits, overloads and grounded phase. The ground trip arrangement shall be designed to de-energize the lighting circuit at not more than 50 percent of the available ground fault current.

SECTION 75.1719-3 METHODS OF MEASUREMENT AND DETERMINING COMPLIANCE

The Illumination and Communications Branch of the Beckley Electrical Testing Project (BETP), MSHA, at Beckley, West Virginia, tests and evaluates lighting systems to determine the luminous intensity of surfaces required to be illuminated in accordance with Sections 75.1719-1 (e)(1) through 75.1719-1(e)(6) and to determine that the lighting fixtures are designed and installed to minimize discomfort glare in accordance with Section 75.1719-2(g).

Prior to December 30, 1977, Technical Support did not test and evaluate lighting systems to determine whether they were designed and installed to minimize discomfort glare. A Notice was published in the Federal Register dated December 30, 1977, which advised mine operators and equipment manufacturers of changes made in the procedures for the issuance of STE's by Beckley Technical Support. The Notice advised the industry that MSHA's assumed reflectance of the coal surfaces of underground working places was increased from 2 percent to 4 percent, which is based on the average reflectance rather than the worst case reflectance of the working place. This change reduced the amount of incident light required on all surfaces for the issuance of STE's from approximately 4 footcandles to 2 footcandles. The reduction in the amount of required light is intended to permit the industry to equip light fixtures installed in the miner's normal visual field with diffusers or louvers for the purpose of eliminating discomfort glare.

STE's issued after December 30, 1977, specify the following:

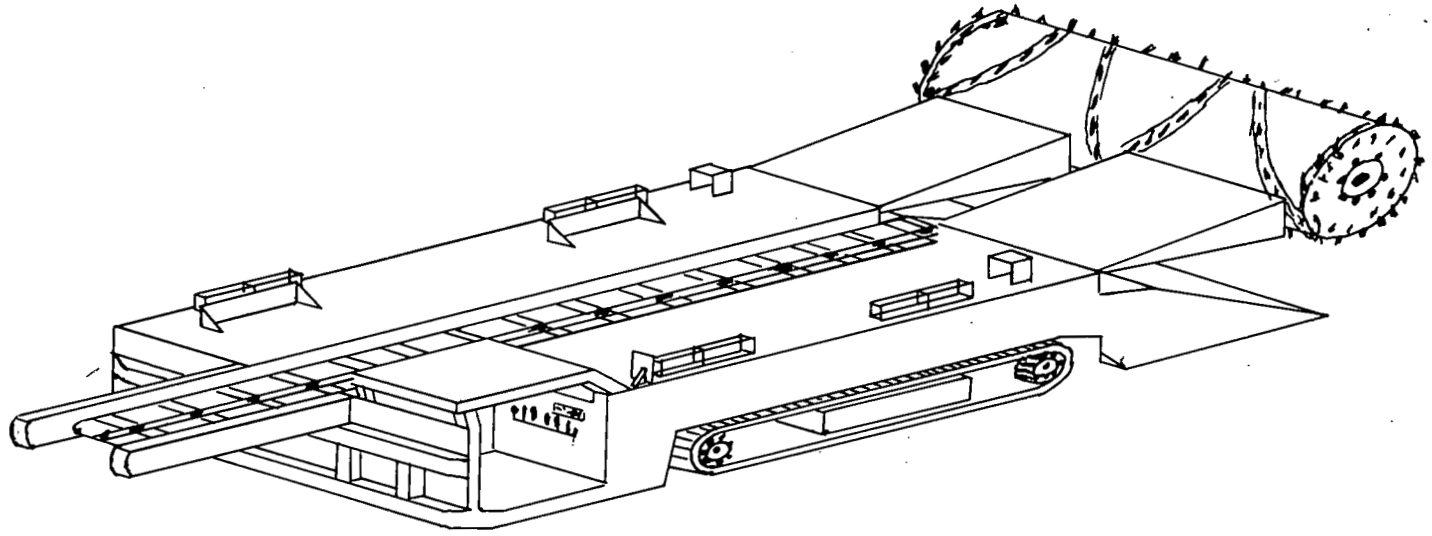
1. The type and location of diffusers and louvers necessary to minimize discomfort glare.

2. The size and location of back shields and/or blackout shields designed to prevent the miner from seeing the light fixture lens from his normal operating position.

3. Any additional guarding of lighting fixtures for mechanical protection (see figure 12).

For lighting systems which have been tested, evaluated and judged to meet all the criteria of Sections 75.1719 through 75.1719-3, a STE will be issued by BETP. Each STE specifies the maximum height and width of the working place, the type and model of the machine (for machine-

Figure No. 12. Continuous miner machine showing back shields.



mounted lighting systems), the location and orientation of lighting fixtures, the type of diffusers or louvers and other conditions under which the lighting system must be installed and operated to provide compliance with the applicable provisions of the illumination regulations. When an accepted lighting system is installed on a machine, a 3- X 5-inch metal plate containing pertinent information regarding the lighting system shall be attached to the machine near the permissibility plate (see figure 13).

STE's are based upon incident light level measurements. Incident light level measurements are not taken within 1 foot of the machine as specified by Section 75.1719-3(b)(9). This is inclusive of the floor area which is within 1 foot of the machine contour. Also, the immediate roof area which is within 1 foot of the top of the machine is not required to be illuminated.

1. Incident light measurements of the face, ribs, roof, floor and exposed surfaces of mining equipment are made with the machine or mock-up located in the center of the actual working place or the simulated working place with the cutting, loading or drilling head toward the face.

2. All machines or machine mock-ups, with the exception of roof bolters and other self-propelled equipment which are classified under Section 75.1719-1(e)(6), are placed within 3 feet of the face.

3. All canopies and traveling roof-support systems are raised to their highest position. All extending booms or drilling heads are placed in the fully retracted position.

4. All booms, drilling heads, auger drums and ripper heads are placed in the down position.

Generally, most machine-mounted lighting systems provide low quantities of incident light, usually below 2 footcandles, with a 1- to 2-foot perimeter around the contour of the machine on the floor. The floor area approximately 3 feet out from the machine contour usually has much higher quantities of incident light. It is allowed, in STE applications, to average the lower light levels with the higher light levels as long as the actual light quantities can be averaged to obtain 2 footcandles for any 4 square feet field outside the 1-foot contour of the machine (see figure 14).

SECTION 75.1719-4 MINING MACHINES, CAP LAMPS REQUIREMENTS

Section 75.1719-4 contains the following additional requirements which are intended to provide increased visibility for miners:

1. Paint used on exterior surfaces of mining machines shall have a minimum reflectance of 30 percent, except

Figure No. 13. Replica of metal plate to be installed on a machine equipped with a lighting system tested and evaluated by Technical Support.

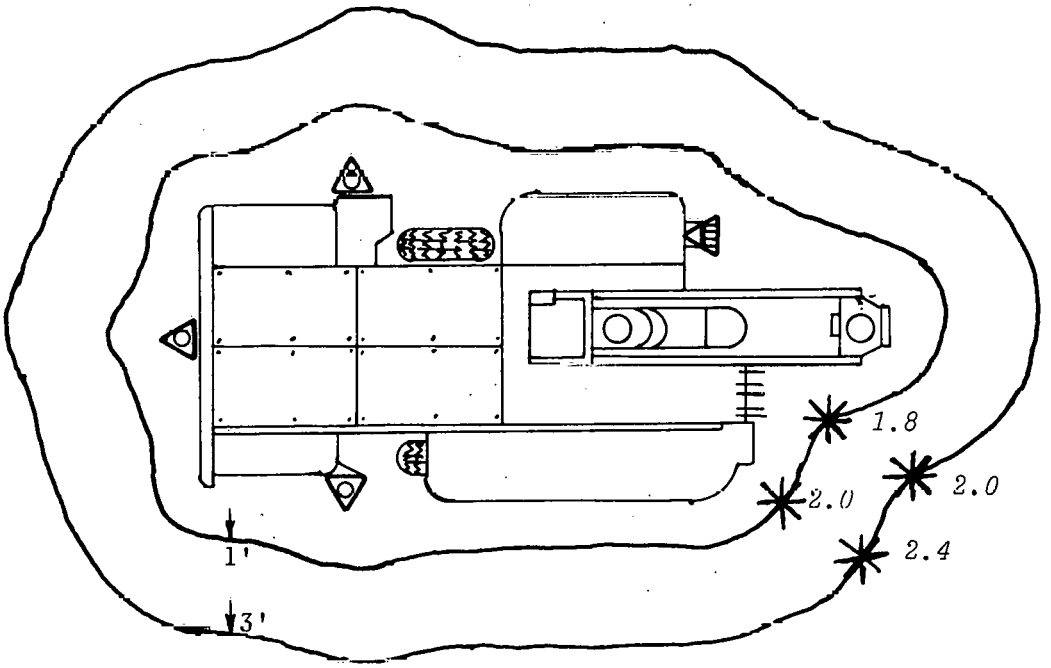
United States Department of Labor Mine Safety and Health Administration		
ILLUMINATION SYSTEM		
STE	CANOPY	
ENTRY DIMENSIONS	H - Max.	W - Max.
	H - Min.	W - Min.

QTY	MFG. NAME	TYPE

Metal STE plate codes.

W----------With
WO----------Without
RC----------Remote control
FLU----------Fluorescent
HDL----------Headlight
MV----------Mercury Vapor
INC----------Incandescent
SOD----------Sodium

Figure No. 14. Averaging technique.



cab interiors and other surfaces which might adversely affect visibility.

2. When stationary lighting fixtures are used, red reflectors mounted in protective frames or reflecting tape having an area of not less than 10 square inches shall be installed on each end of the mining machine, except that continuous mining machines, loading machines and cutting machines are required to be provided with red light-reflecting material only on the outby end.

3. Each person who goes underground shall be required to wear an approved personal cap lamp or equivalent portable light.

4. Each person who goes underground shall be required to wear a hard cap that shall have a minimum of 6 square inches of reflecting tape or equivalent paint or material on each side and back.

CONCLUSION

There are seven major manufacturers of lighting fixtures. Each lighting fixture has some definite design advantages which provide the mine operator with a selection. Each mine operator must determine which fixture best suits his mining needs. The STE program is a service provided by MSHA whereby the operator can procure lighting systems with MSHA's assurance of compliance with the lighting regulations. The mine operator is not required to obtain a STE. An operator may design and install a lighting system which meets all the criteria of Sections 75.1719 through 75.1719-3 and be in compliance with the lighting regulations. The Illumination Laboratory of Technical Support, MSHA, at Beckley, West Virginia, is available to help anyone in the mining industry with lighting problems.

The installation of quality lighting systems in the working places should be conductive to the development of safer, more productive coal mines.

"INDUSTRY'S VIEWS OF ILLUMINATION REGULATIONS AND THEIR IMPLEMENTATION"

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ABSTRACT

Section 317 (E) of the Federal Coal Mine Health and Safety Act of 1969 requires that all working places in a mine be illuminated while persons are working in such places. After nearly seven years of development and in-mine experimentation by the USBM, MSHA and the industry, Section 317 (E) became effective on July 1, 1978. At present it is still doubtful if the mine operators can install lighting hardware on the majority of machines in a practical and maintainable configuration which will enhance the safe and efficient performance of the miners' tasks in the working place.

INTRODUCTION

Section 317 (E) of the Federal Coal Mine Health and Safety Act of 1969 became effective on July 1, 1978. This section of the Act prescribes standards under which all working places in a mine shall be illuminated while persons are working in such places.

The first proposed rulemaking for illumination of underground coal mines was published in the Federal Register for December 30, 1970. In light of written comments, suggestions and objections to this proposed rulemaking, the proposed standards were withdrawn and reproposed in the Federal Register for October 27, 1971. In light of further comments, suggestions and objections, a public hearing was held on April 4, 1974 and standards were again reproposed and published in the Federal Register for April 1, 1976. Promulgation of the lighting standards took place on October 1, 1976, and were to become effective April 1, 1978. Further changes in Mining Enforcement Safety Administration (MESA) procedures for the testing and evaluation of illumination systems on December 30, 1977 again delayed the effective date of the illumination standards until July 1, 1978. The complex history of the coal mine illumination standards just outlined provides evidence of the many technical difficulties encountered in implementation of coal mine lighting standards.

FIELD EVALUATION OF EARLY MINE LIGHTING SYSTEMS PROVES UNACCEPTABLE

Consolidation Coal Company's (Consol) initial efforts in the area of face illumination began in early 1972 when Consol and the United States Bureau of Mines (USBM) agreed to a cooperative study of underground face illumination. Early mine lighting systems were aimed at investigating practical lighting for underground face equipment. Although mine lighting systems were continually being installed by Coal operators and the USBM since 1972, it was not until April 1, 1976 that the proposed illumination standards and light measuring techniques were made public. When these illumination standards were defined and measuring techniques were available, Consol checked their lighting systems underground and determined that none of the early lighting systems were in compliance with the proposed illumination standards.

After determining that all of our experimental face lighting systems were not in compliance, we began adding additional lighting hardware in order to meet compliance with the proposed regulations. We were not able to meet compliance with the proposed regulations with practical lighting systems.

In May of 1976, Bituminous Coal Operators Association (BCOA) members asked MESA to demonstrate the practicality of compliance with the mine illumination regulations. United States Steel Corporation, American Electric Power Corporation and Consolidation Coal Company offered to work with MESA in a joint effort to develop the capability of providing adequate face illumination in a safe, workable manner which would not detract from efficiency of operation.

BCOA members involved in this cooperative study submitted necessary machine drawings, sketches, etc. to MESA and MESA performed "black box" studies and specified the type and locations of luminaires to be installed on the test machines. MESA was confident that they could specify lighting systems which would be in compliance and would be practical so as not to detract from efficiency of operation.

Test illumination systems were installed and after several months of operation, the Joint Industry Health and Safety Committee[1] toured and reviewed all three test illumination systems. One mine visited by the Joint Union - BCOA Committee was an encouraging success from the standpoint of lighting, the mine had a high reflectivity coal seam and fluorescent lights were employed to achieve acceptable light levels in the working place. A second fluorescent illumination system visited by the Committee was generally disliked by the Union employees but was tolerated. The final illumination system reviewed (employing mercury vapor and fluorescent lights) was a total failure; it was obviously counterproductive to safety. The men, after having given a fair trial of letting the lighting system stay on for about 30 minutes, refused to operate the machine any longer.

[1] The Joint Industry Health & Safety Committee is comprised of coal industry, MSHA, and UMW of A members.

The result of the Joint Industry Health & Safety Committee mine visits was to determine that machine-mounted lighting fixtures used to provide the required 0.06 footlamberts light level could create discomfort and disability glare and create additional hazards to miners.

MESA STATEMENT OF TEST & EVALUATION PROCEDURES REVISED TO LOWER GLARE PRODUCED BY ILLUMINATION SYSTEMS

Based on evidence presented by the Joint Industry Health & Safety Committee, MESA acknowledged that the coal mining industry and the mine lighting fixture manufacturers had encountered problems in minimizing discomfort glare to miners. These problems were encountered even when MESA tested and evaluated lighting systems were installed on machines to achieve compliance with lighting requirements. MESA stated that experience with machine-mounted illumination systems in underground coal mines had shown that excessive brightness of the light sources causes discomfort glare and can create additional hazards to miners by actually reducing the miners ability to observe rotating machinery parts, moving machines, ropes and other equipment in the working place. Excessive brightness of the light sources also hamper the miners ability to communicate with one another by the customary cap lamp signals.

MESA determined that these problems could be solved by increasing the assumed surface reflectance value of coal surfaces under the simulated conditions and formula used in the evaluation procedures. A higher reflectivity factor of 0.04 would permit the use of diffusers or louvers on the lighting fixtures to reduce and minimize discomfort glare and still provide the required light levels to comply with lighting requirements. MESA set forth these modified Statement of Test and Evaluation (S.T.E.) Procedures in the Federal Register for Friday, December 30, 1977.

FIELD INVESTIGATION SUGGESTS BASIC APPROACHES TO MINE ILLUMINATION SYSTEMS

Consol has gained considerable experience in the installation and field maintenance of coal mine face illumination systems during the past several years. Although a continuing effort is being made, certain conclusions regarding basic mine illumination systems employed on specific face equipment have been drawn.

CONTINUOUS MINERS

Consol has installed and field tested all basic types of machine-mounted face illumination systems and we have concluded that a combination system consisting of very high output (V.H.O.) fluorescent lights (mounted along the body of the machine) and sodium vapor headlights is the most acceptable to both mine workers and maintenance personnel. Standard output fluorescent lights are considered more acceptable than V.H.O. fluorescent lights by mine workers, however, because of their lower light output, it is impractical to mount a sufficient number of fixtures on a continuous miner to meet compliance with lighting regulations.

Sodium vapor headlights have been chosen over incandescent headlights in many instances because they direct more light to the coal face and because a fewer number of sodium vapor headlights must be maintained in the critical face area to meet compliance. Field reports have judged sodium vapor headlights to improve vision of the coal face where dust and mists from water sprays are particular problems. Certain problems have developed during field tests of the sodium vapor lighting fixtures. The resolution of these problems may limit the usefulness of the sodium vapor luminaire.

Approximately 25% of the continuous miners used by Consol are equipped with integral bolters. Due to the fact that the bolter operators are located inby the miner operator, particular attention must be directed to reducing glare produced by the machine-mounted lighting system. In this case of a miner with integral bolters, the same basics of lighting apply; however, one or two additional fluorescent lights must be provided along the body of the miner to achieve uniform light coverage.

In low and undulating seam conditions, we have not field tested a face illumination system which is acceptable to mine workers and will meet compliance with illumination regulations. Although Consol has installed several different varieties of low seam lighting systems, the men are blinded by the lights because of their close proximity to individual luminaires.

LOADING MACHINES

Several Consol operations employ pickup loaders behind their continuous miners to render the flow of coal from the face continuous. Except for the gathering head area, the pickup loader can employ the same illumination system as the continuous miner. Fluorescent or incandescent luminaires must be used to illuminate the gathering head area of the loader so that glare for the miner operator may be minimized.

ROOF BOLTERS

The illumination of roof bolters is a particularly difficult task because men are required to be in many different locations around the machine. Men will also be required to be traveling from the immediate vicinity of the bolter outby to secure supplies. We have determined that the lowest intensity lights should be used to illuminate roof bolters. Twin Boom Bolters and bolters equipped with temporary roof support are even more complex to practically illuminate than are standard bolters. A majority of bolter operators interviewed within Consol prefer illumination of only their immediate work area (where the hole is being drilled and bolted) to total face illumination.

SHUTTLE CARS AND SCOOPS

Few shuttle cars or scoops in Consol are employed as self-loading haulage equipment used as a loading machine. For this reason, Consol's scoops and shuttle cars will be illuminated under the requirements of "other self-propelled equipment" and will require one additional 150 watt

incandescent light on either end of the machine. In the majority of cases, incandescent lights will be wired in series and neon indicator lamps will be used to identify lamp failures.

LONGWALL

Lighting regulations require longwalls to be illuminated for the length of the self-advancing roof support system between the gob-side of the travelway and the face. Illumination is also required in the headgate and tailgate entries for a specified distance.

Consol believes that the fine safety record of longwall mining speaks for itself. The introduction of electricity on chocks and shields for the purpose of providing lighting will introduce risks which far outweigh any possible benefits.

The energy level needed to provide illumination is sufficient to cause incendiary sparking. The amount of cable, estimated to be between 800-2500 ft., and hardware needed to illuminate a 500 foot face is impressive. The number of luminaires may exceed 100, with attendant sockets and junction boxes. The complications associated with each unit (chock/shield) being moved (up and down as well as forward) exposes this system, and especially the cables, to damage far in excess of lighting systems installed on mobile equipment. With this exposure, it is not reasonable to expect that any lighting system would operate year-in and year-out without damage that could result in incendiary sparking. Some day sparking will occur when sufficient methane is present and a major disaster could result.

To minimize the potential hazard just described, Consol believes that it is imperative to install intrinsically safe longwall lighting systems. Unfortunately, it is not possible to have 100% intrinsically safe longwall lighting system.

PROBLEMS REMAIN WITH FACE ILLUMINATION

Although the illumination of underground face equipment has begun on a large scale, many problem areas are as of yet unresolved.

One major problem associated with the lighting of face equipment is the illumination of D.C. powered equipment. Although several D.C. power supplies have recently been developed or are presently under development, sufficient in-mine testing has not been completed to demonstrate their feasibility for use underground.

Another concern to many operators is that the majority of V.H.O. fluorescent lights are operated from a dual lamp ballast. If two fluorescent lights are wired to the same dual lamp ballast, both lamps may extinguish when only one lamp has failed and a positive identification of faulty lamp cannot be made if both lamps are not replaced.

Although MSHA has recognized the need to minimize glare produced by underground illumination systems and has revised S.T.E. procedures to reduce glare, excessive brightness produced by certain lighting systems is of major concern.

As mentioned before, glare from light sources creates additional hazards to miners by actually reducing their ability to observe rotating machinery parts, moving machines, ropes and other equipment in the working place. Excessive brightness of the light sources also hampers the miners' ability to communicate with one another by cap lamp signals.

FACE ILLUMINATION MUST BE SAFE AND PRACTICAL

Consol's field experimentation with face illumination systems has demonstrated that the major problem in meeting compliance with MSHA illumination regulations is not one of mounting enough lighting fixtures on a given piece of face equipment to generate the necessary light output. The problem is to install lighting hardware on equipment in a practical and maintainable configuration which will enhance the safe and efficient performance of the miners' tasks in the working place.

If practical illumination systems are to be installed in our underground mines, many problems concerning illumination safety and practicality must be resolved. It is naive to assume that underground lighting systems can be proven acceptable for use solely on the basis of laboratory testing and installation on mockups of mining machines. It may be possible to achieve compliance with illumination regulations by showing the location of luminaires on machine blueprints and mounting this hardware on wooden machine mockups. It is an entirely different undertaking to install this same lighting hardware on actual mining machines under actual mining conditions. Very few mining machines which have been in service for any length of time remain unmodified, therefore, many lighting installations will require custom fitting of lighting hardware.

The most favorable time to install lighting hardware is when the face equipment is being initially fabricated or when it is undergoing major rebuild in an outside machine shop.

If mounting is done in the shop environment, lights can be properly protected by modifying the machine cover plates or structure and lighting cables can be properly guarded.

It is unfortunate that the majority of lighting installations will, of necessity, be done in-mine, which is the worst circumstance in which to do custom installation.

MEASUREMENT OF RAIL BOND IMPEDANCE

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ABSTRACT

Many underground mines use electric rail haulage incorporating the track as the return conductor. This approach makes it essential that the rail joints have as little resistance as possible in order to properly carry the return current. Resistance is significantly reduced by bonding the joints, and is required by Federal law. Bonds can become broken or dislodged for many reasons, however. These open bonds can cause significant voltage drop in the rail at the unbonded joint. This drop can result in power loss, and can sometimes seriously affect the operation of ground-check monitors and ground-fault relays. Because most bonds are buried beneath several inches of coal, gravel, rock dust or mud, it becomes very costly for the mine operator to have these bonds checked frequently.

West Virginia University has undertaken the task of developing instrumentation which will quickly and accurately assess the impedance of each bond. This paper contains an analysis of the severity of the problem, a description of the instrumentation developed to measure bond electrical characteristics, and the results of some preliminary measurements made underground.

INTRODUCTION

Many underground coal mines use electric rail haulage incorporating the track as the return conductor. This approach makes it essential that the rail joints have as small a resistance value as possible in order to properly carry the return current. The resistance of the joint is greatly affected by its mechanical connection. Calculations show that the resistance of a loose rail joint is significantly reduced by placing a bond across it. Also cross bonds, bonds interconnecting the two parallel track, placed at approximately every two hundred feet, aid in keeping both sides of the haulage track at the same potential. At the same time the cross bonds insure that the return conductor or section of track will not be separated from the rectifier which would occur if a bond were to break.

Bonds can become broken or take on large resistance values for various reasons, including vibrations from heavy machinery, objects being dropped on the bond, or improper installation of the bond. These

open bonds can cause a large voltage drop in the rail at the unbonded joint. Should several of the bonds be broken, this voltage drop can cause large power losses. Sparking may also occur between rails if the difference in potentials becomes large enough. Leakage currents flowing into the earth from the rail can find their way to various machine frames resting on the mine floor. This current may return to the safety ground bed via the ground conductor in the trailing cable to the machinery, causing electrolytic corrosion of the ground bed and increasing its resistivity. This added current to the ground conductor of the trailing cable may also render ground check monitors inoperative. Direct currents on the ac ground can also effect the protective relay coordination by placing a current in the safety ground resistor. This current may cause saturation of the current transformer reducing its sensitivity. The voltage drop across the safety ground resistor caused by this current may also cause the circuit breaker to trip out, turning off the power to the machinery.[1] Figure 1 shows some of the possible leakage current paths for an underground mine power system.[2] All the above effects can present a danger or hazard to the mine personnel who are working within the mine.

BONDS

Presently there are only three methods of providing rail bonds. These are:

A) Direct weld - Whether an arc or thermit weld, this method provides a permanent connection between the two sections of rail. The direct weld provides a good physical contact, yielding a low resistance path for the current to flow through.

B) Adhesive chemical bond - Consisting of an epoxy resin paste, the chemical bond immobilizes the rail ends at the joint. To be effective, the rails must be cleaned and sanded so that when the epoxy sets, the mechanical connection between rails will provide a low resistive path. The epoxy also fills the voids and aids in making the joint moisture proof. Chemical bonds are most used where conditions prohibit a thermit type weld.

C) Wire rail-bond - Consists of a 2/0 or 4/0 wire cable, usually 30 inches in length which is welded to each rail across the physical connections of the rail joint. When the bond is in place its length is shortened to 24 - 26 inches.

While all the above methods are in practice today, the direct weld and chemical adhesive bonds tend to be used on sections of track which are considered as a permenant installation or in a section of the mine where a retreat is not being planned. The wire bond is the most common method used today.

Table 1 gives the electrical characteristics of the rail and the rail joint. [1]

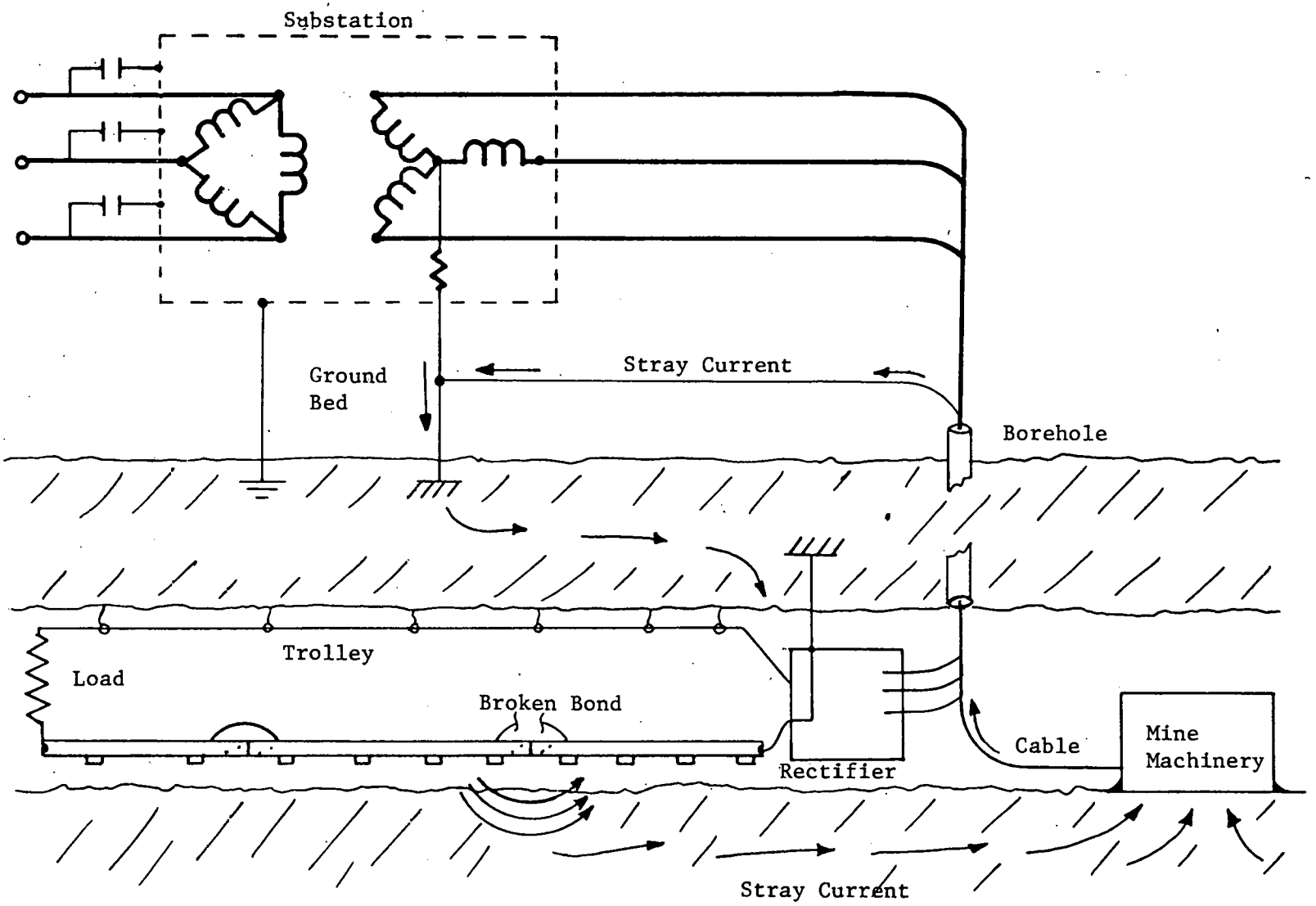


Figure 1.
Possible Paths for Stray Current in a Mine Power System

TABLE 1

ELECTRICAL CHARACTERISTICS OF HAULAGE RAIL

Wt. of Rail Per Yd. (lbs)	Resistance (ohms/1000 ft.)	Resistance of Unbonded Joint (ohms)
30	0.018	0.072
40	0.0135	0.054
50	0.0108	0.0432
60	0.0090	0.036
70	0.0077	0.0308
80	0.0067	0.0268
90	0.0060	0.024
100	0.0054	0.0216

A copper bond of 4/0 wire has a resistance of .052 ohms/1000 ft. From this is is found that the resistance of a 30 inch rail bond to be:[3]

$$\frac{(.052)\Omega}{1000\ \text{ft}} \times \frac{1\ \text{ft}}{12\ \text{in}} \times 30\ \text{in} = .13\ \text{m}\Omega\ \text{(milliohm)}$$

By placing the rail bond in parallel with a rail joint for a 30 lb. rail the total resistance is found to be:

$$\frac{(.13\ \text{m}\Omega) \cdot (.072\Omega)}{(.13\ \text{m}\Omega) + (.072\Omega)} = 0.12977\ \text{m}\Omega.$$

These calculations show that the resistance which needs to be detected is very small, but there is a relatively large difference between a bonded joint and an unbonded joint. For a 30 lb track, the unbonded joint has the resistance of approximately 4000 ft of track. (Information received from the manufacturer of Chemical Adhesive bonds indicate that when properly applied there is no difference between the rail resistance and that of the bonded joint).

With the large difference in resistance between a bonded verses unbonded joint in mind, researchers at WVU set-out to develop circuitry which could detect a broken or dislodged rail bond, by measuring the resistance of the joint. At precent the only way of examining a bond is by visual inspection, which tells nothing about electrical conductivity. Since most bonds are buried beneath several inches of coal, gravel, rock dust, or mud and water, this inspection requires alot of man hours and becomes very costly for the mine operator.

INSTRUMENTATION

Figure 2 is a block diagram of the circuitry developed at WVU. Since the haulage rail is used as the return for ground it will contain ac as well as dc currents. This current will be constantly changing in magnitude and direction depending on the equipment being operated

Block Diagram of Instrumentation

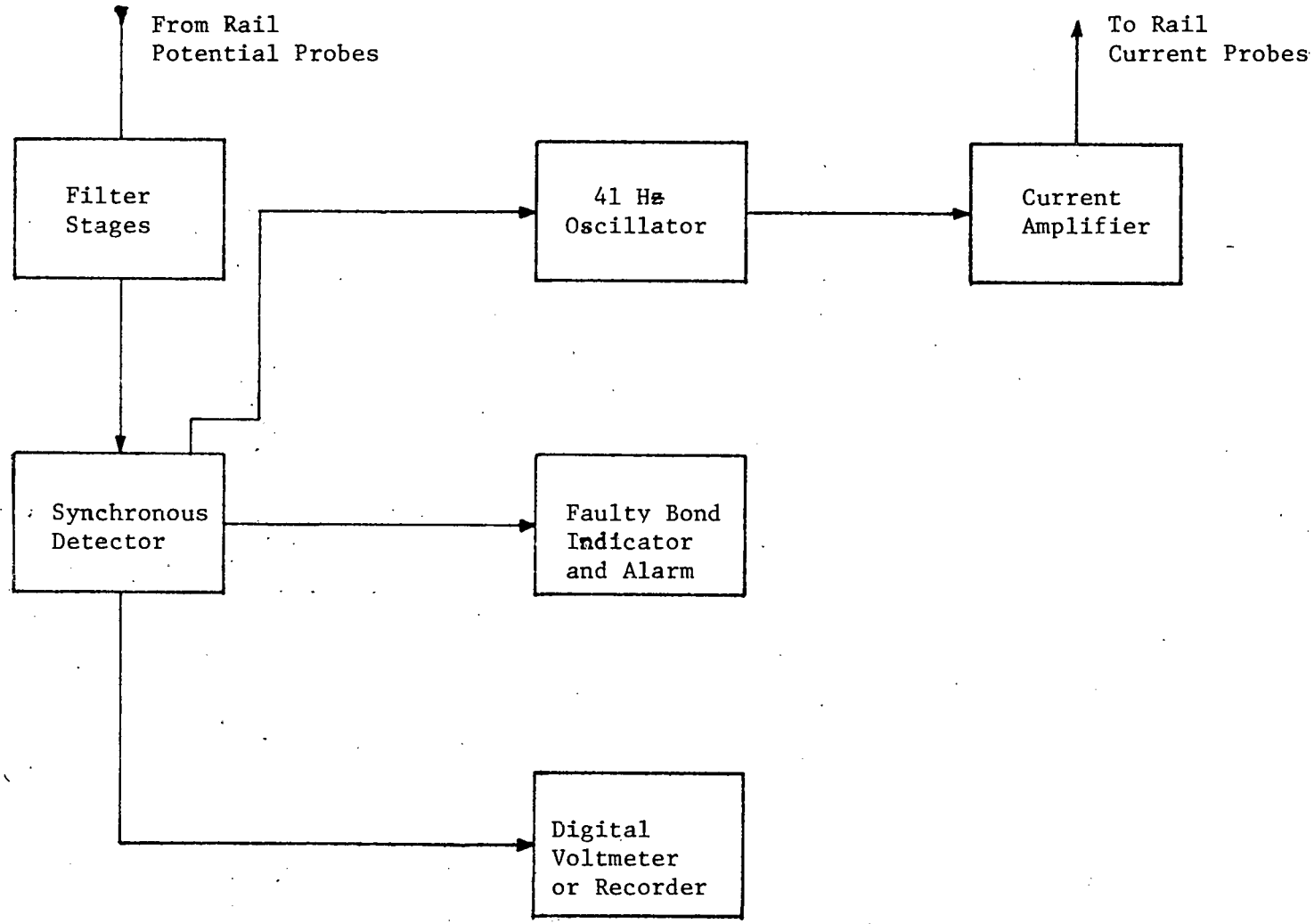


Figure 2.

at the time. For this reason a constant current source which uses a prime frequency to reduce interference from harmonics and sub-harmonics of 60Hz is used. A 41Hz sine wave signal is developed from Intersil 8038 function-generator integrated circuit and passed through a buffer to prevent loading. The signal is then placed into a constant current source, utilizing class B operation. The output of the amplifier is fed through a transformer which boosts the current. The current is then fed to the two current electrodes which contact the rail. The voltage which is picked off the rail via the two voltage electrodes is passed through a low-pass filter with a corner frequency at 52Hz. This is done to help reduce the 60Hz signals which are present on the track, while at the same time to pass the 41Hz signal which the circuitry uses. The filter input also contains some transient suppression to protect the electronic circuits against the power surges present on the track. The filtered voltage is then fed into a synchronous detector. The detector employed operates by synchronous rectification of the signal to be detected. This can be accomplished by effectively multiplying the input signal (plus noise and interference) by a pure signal (output of the oscillator) which is exactly the same frequency and in phase with the expected signal. This multiplication causes a full wave rectification of the signal which is to be detected and the cancellation of low frequency interference. (The higher frequencies were attenuated by the low pass filter). The low frequency cancellation occurs because the change in amplitude of the low frequency signal during a half cycle of the higher frequency (41Hz) is so slight, that multiplication by the higher frequency signal produces zero output for the low frequency input. If the injected signal frequency (41Hz) is present along with the low frequency interference. The synchronous rectifier will still produce a full wave rectified output of the injected signal.

The filtered input voltage is fed into a pair of analog switches. The 41Hz signal is passed through a phase shifter and into two zero crossing detectors. The control signals from the zero crossing detectors operate the analog switch array synchronously with the incoming 41Hz. The output of one of the analog switches is fed into an inverting amplifier, while the output of the other switch is fed into a non-inverting amplifier. The invert/non-invert characteristic thus generated produces synchronous rectification, at the signal frequency, and subsequent low frequency cancellation. The amplifier output are summed and fed into a low-pass filter with a corner at .8Hz to give only a dc output which corresponds to the incoming signal which was detected. The output of the filter is fed to a comparator which allows the operator to set a range point at which an alarm or indicator would be turned on when the output voltage was above a preset level. This would indicate that the bond's resistance was too high or out of tolerance. The output of the filter also feds to a scaling amplifier and then either to a recording voltmeter or a digital panel meter, where the output voltage would be read as the bonds resistance, according to the scaling used.

A four-probe method is used to inject the current and measure the voltage from the rail. The four electrodes, or in this case wheels

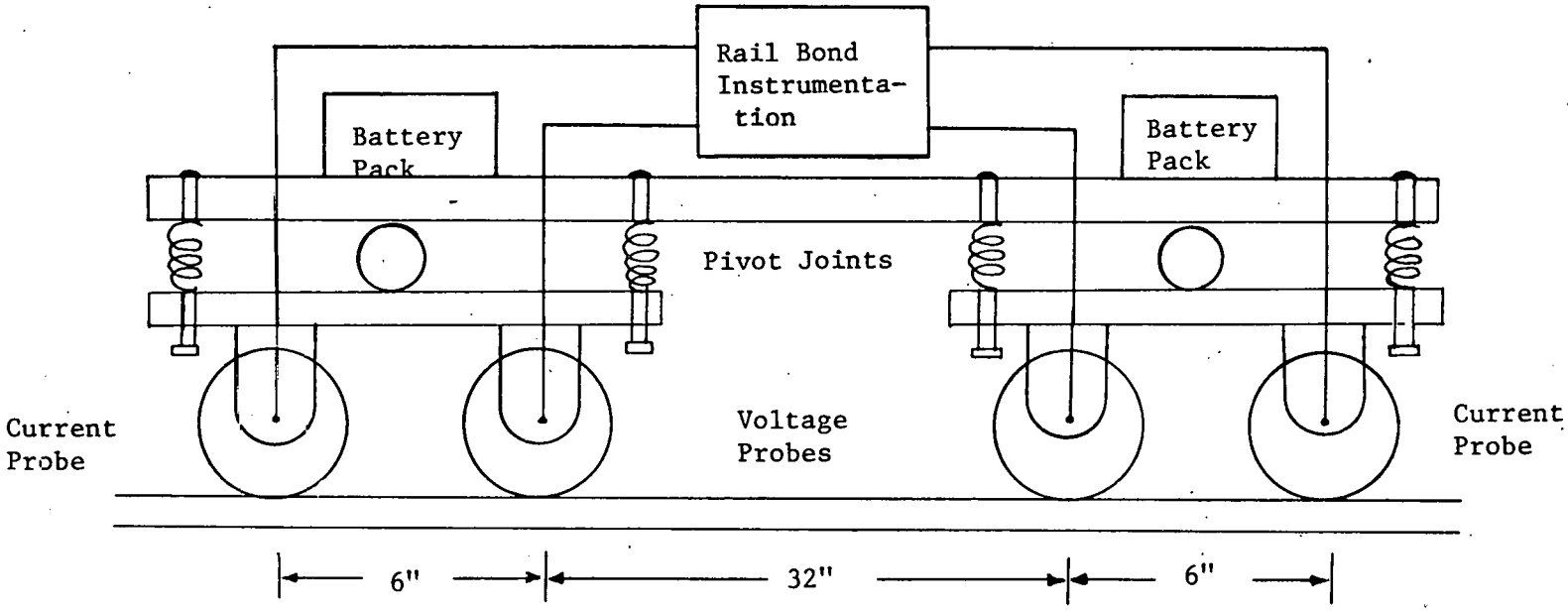

Rail Bond
Instrumenta-
tion
Battery
Pack
Battery
Pack
Pivot Joints
Current
Probe
Voltage
Probes
Current
Probe
6"
32"
6"

Figure 3.

are set-up on a cart which is shown in Figure 3. The inner two wheels are to measure the voltage, and are spaced far enough apart to allow the entire bond to be between them. The outer two electrodes are used to inject the current into the rail. A piece of V-shaped stainless steel is used to make contact between the wheels and the electronic circuitry. A spring pivot joint is placed at each end of the cart to allow for changes in the track height while keeping all four electrodes in contact with the rail. The cart also houses the battery storage compartments. All electronic circuitry is mounted on printed circuit cards and housed in an approved enclosure along with the current transformer and alarm system.

INITIAL LAB TESTS

The electronic circuitry has been tested on various resistive loads and on a bond obtained from a local mine. The output voltage was within a reasonable tolerance of what was expected and minor modifications have been made to reduce offset voltages in the amplifiers and scaling circuit. The comparitor circuit worked as expected, and the operator can adjust the set point to allow for range of variations in resistance before tripping the alarm. The cart is being modified for stability. Presently, WVU is awaiting approval which will allow field data to be obtained from a local mine.

CONCLUSIONS

With the need for a good low resistive return to ground for haulage track operations and the rising cost of labor which is used to inspect the haulage track for broken bonds, researchers at WVU feel that the electronic circuitry for measuring rail bond resistance will be a welcomed advance in technology for coal mines in the future.

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ACKNOWLEDGEMENT

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ELECTRIC SHOCK PREVENTION

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ABSTRACT

Electric shocks are a serious problem in the mining environment. Because of extremely wide ranging use of electrically powered equipment in the mining industry, the hazard to personnel due to electrical shock and related injuries are not restricted to any single area but cover the entire spectrum of mining activity. Electrical accidents, resulting in personal injury, due to improper or complete lack of grounding of electrically operated mining machines, form a significant percentage of the overall electrical accident picture.

The object of this paper is to describe methods of detection of unsafe electrical conditions with respect to the grounding of electrically operated mining equipment. It also includes a discussion of prototype instruments which effectively monitor the condition of the grounding safety devices currently in use on mining equipment and warn of dangerous conditions as soon as they appear.

The paper is based on work accomplished under USBM Contract Number J0357115. The contract was initiated under the Pittsburgh Mining and Safety Research Center Program. It was administered under the technical direction of PM and SRC with Mr. Roger L. King as the Technical Project Officer. Mr. Daniel B. Dawkins was the Contract Administrator for the Bureau of Mines.

PERSONNEL HAZARD DUE TO ELECTRIC SHOCK

Much work has been done in the area of human response to electricity. The following values are given as a general indication of typical threshold values and serve only to define areas of interest for instruments being developed.

Threshold of perception to 60 Hz AC between major extremities - 0.5 mA.

Muscle disfunction "let go current" - 9 mA.

Respiratory paralysis - 30 mA.

Heart disfunction - 75 mA.

Cardiac arrest - 4 amps.

Hazardous potential threshold, skin contact - 40 V.

Skin contact resistance - 1K-5K depending upon conditions.

Body resistance between major extremities - 500 ohms (exclusive of skin).

Skin effects disappear as voltage increases, higher potentials being able to pierce the skin's protective layer.

The object of this paper is to discuss methods of detection of unsafe electrical conditions with respect to the grounding of electrically operated mining equipment. It also includes design information of prototype instruments which effectively monitor the condition of the grounding safety devices currently in use on mining equipment and warn of dangerous conditions as soon as they appear.

A device of interest is a voltmeter that is intended to be permanently mounted on the mining machine and continuously monitor the electrical potential of the frame of the mining machine relative to the earth. The device also contains a presettable threshold level which is capable of giving a warning or opening a relay contact whenever the threshold potential is exceeded.

Another is a device intended for use on DC operated machinery employing safety grounding diodes. The diode monitoring function is physically included in the machine mounted instrument housing with the single resulting package serving the dual purpose of frame potential measurement and safety diode condition monitoring.

GROUNDING DIODE CHECKING

Present Electrical Equipment

The vast majority of electrically operated mining equipment in the field today is still manufactured on a unit by unit basis. While basic form, fit and function of individual machines remain fixed for relatively long periods of time, the individual electrical components included in each machine produced are subject to slight but constant variation. This is true of virtually all manufacturers including even the larger volume equipment producers. The variations in machine components are often necessitated by the manufacturer's responsibility to delivery large expensive equipment on a regular schedule regardless of random inventory shortages of specific component types. "Equivalent" substitutions are made whenever necessary to expedite production. In addition, the extremely large volume of repaired and rebuilt equipment makes the exact parts complement or electrical schematic of any operating piece of heavy mining equipment difficult to predict. Any equipment intended for installation in existing mining equipment must be universal in application and easily adaptable to changing conditions.

Electrical Control Boxes

A large number of mining equipment manufacturers purchase entire control boxes, as a complete unit, from an independent supplier. These boxes are treated as a component part with the internal design particulars left to the box supplier. Here, also, actual numbers of particular internal components vary from unit to unit. However, as far as safety grounding diodes are concerned, virtually all diodes are functionally equivalent.

Grounding Diodes

Safety grounding diodes are used in DC powered mining equipment such as shuttle cards, to prevent voltage from appearing on the machine frame. They are connected between the frame and the DC return line so that if the hot side of the DC supply comes in contact with the frame, the diode will go into heavy conduction and shunt the power to ground via the safety ground lead and/or the DC return until the breaker at the load center trips.

Although many different individual grounding diode part numbers are in current use, the basic electrical characteristics of all units are virtually identical.

Differences in part numbers are due to individual manufacturer's number systems, physical packaging differences between various units produced by the same manufacturer and difference in maximum voltage and current ratings of units installed in various machines. A typical practice followed by

one control box manufacturer is to supply a single grounding diode on machines up to 50 hp, two diodes in parallel for 50-75 hp machines, and three in parallel for machines up to 100 hp.

1. Failure Modes of Grounding Diodes

Silicon rectifiers of the type used as grounding diodes are generally fabricated by either compression bonding techniques or brazed construction. Compression bonding involves mounting the actual semiconductor inside a welded case such that a least one contact to the semiconductor wafer is made through a compression type contact. Brazed construction involves use of hard soldering techniques. Field experience seems to indicate little difference between construction techniques with ultimate device failures generally being characterized by either a shorted diode junction or a device whose pigtail lead has burned open due to excessive current under fault condition. Little field information is presently available regarding partial failures of grounding diodes on mining equipment except for some cases involving the detection of excessive reverse leakage prior to ultimate device failure.

Failures may be classified as either electrical or mechanical nature.

a. Electrical Failure

(1) Overvoltage

Overvoltage failure of silicon power rectifiers results from the application of reverse voltages exceeding the avalanche breakdown rating of the diode. In safety ground applications, this most often results from the extremely high level short duration voltage transients so common in current mine power systems. Once the reverse blocking voltage of a device has been exceeded, current begins to flow across the semiconductor junction. The resulting high level localized power dissipation results in a burn usually near the passivated edge of the element. Continued or repeated application of excessive voltage spreads the burn area, finally resulting in complete device failure. External indications of overvoltage damage prior to complete device failure are a reduction in reverse voltage blocking capability and an increase in reverse leakage current as the junction deteriorates.

(2) Overcurrent

In the mine usage, overcurrent failure results from electrical fault currents flowing through the diode. Failure due to overcurrent results from excessive heat concentrated in the junction region of the semiconductor element resulting in a general degradation passivation. This damage can result from either large continuous currents or high surge currents for

relatively short periods of time. The currents required to cause overcurrent failure are orders of magnitude larger than those involved in failure due to overvoltage. This is because of the low forward voltage drop of the diodes; however, the ultimate effect is very similar. Damage due to overcurrent usually encompasses the major portion of the junction area and is manifested, prior to total device failure, by a reduction in the reverse blocking voltage of the device and an increase in reverse leakage current. Very high surge currents result in bulk burns of the semiconductor material and produce almost instantaneous failure.

b. Mechanical Failure

(1) Loss of Hermeticity

This type of failure generally results from physical abuse or improper mounting, destroying the integrity of the package seal. Failure is generally very slow to appear as the effect of the broken seal is usually not immediately apparent. The broken seal allows internal contact materials to oxidize resulting in higher flow conditions. This results in device failure due to excessive temperature. This type of failure should be rare in safety grounding applications as the diodes involved are not intended for and do not generally carry large continuous forward currents. They are, however, often subjected to physical abuse. In safety ground applications, a possible result of contact oxidation progressing for an extended period of time would be a gradual increase in forward voltage drop of the diode, due to increasing internal resistances; followed by final device failure in the open state, due to complete loss of internal contact.

(2) Fractured Element

Excessive mounting torque or poor mounting surface condition can deform the device package and physically fracture the semiconductor element. This results in a sudden voltage type failure at the facture point. Excessive physical stresses present at the time of mounting may not immediately fracture the element but cause device failure at some later date from a sudden temperature change in the semiconductor element such as that normally induced by relatively mild current surges.

(3) Insufficient Mounting Torque, Poor Mounting Surface

Insufficient mounting torque or load force results in high temperature due to lack of heat transfer between the diode case and its heat sink.

This type of failure should be rare in safety ground usage because of the general absence of large continuous currents in these

devices. Excessive temperature operation from any cause is detectable by a marked increase in reverse leakage as junction temperature rises. Leakage currents double for every 10°C increase in junction temperature. Leakage currents reduce as the junction temperature decreases provided maximum operating junction temperatures (approximately 190°C) are not exceeded.

VOLTAGE MEASUREMENTS

Earth Contact Resistance

A study was made to determine the earth contact resistance to be expected from earth electrodes of various types. This was done to gain insight into the magnitude of the problem associated with establishing a local earth reference in an operating mine from a moving piece of equipment. Measured values were obtained for four separate electrode configurations at two different locations in the Federal #2 Mine of the Eastern Associated Coal Company near Fairview, West Virginia. The electrodes tested included a metal whip antenna, a heavy chain dragging along the fllor, a conductive wheel, and a hoe or plowing type electrode. With the exception of the chain, separate measurements were made with the electrodes contacting the roof as well as the floor of the mine at each location. The two locations includes a recently mined area within 800 feet of the active face of the mine, and a second site near the base of the main shaft which had not been the scene of active mining for several years. A summary of the data is found in Table A.

Table A. Ranges of Earth Contact Resistances for Various Electrodes

Electrode	Range of Earth Contact Resistances	
	Floor	Roof
Whip	3K - 1000 M	3K - 2000 M
Chain	500K - 2000 M	-
Wheel	4K - 500 M	40K - 2000 M
Hoe	400K - 1000 M	400K - 250 M

In addition to the measured electrode resistance value, it was noted in the report that the earth potential within several feet of a line machine frame is significantly altered such that the immediately adjacent earth assumes the potential of the live frame.

This information was verified by other researches at West Virginia University during independent study work in the area of ground potential gradients near haulage rails in coal mines, and is attributed to the relatively high resistivity of the earth in and around the coal.

This finding is significant in that it definitely indicates that any frame potential measurement instrument intended for mine usage and using the earth as a reference must not obtain its earth reference point from directly under the machine but at some remote location. From present data, it appears that an earth conduction distance of 10-15 ft. from the frame is sufficient, and further testing may show the mine roof above the machine to be the most convenient location, even though contact resistance data indicates the roof contact in general to be of higher resistance than floor contact.

Machine Ground Contact Resistances

One parameter affecting the performance of any instrument monitoring frame and/or ground circuit potentials in mine applications is the contact resistance of the machine frame-earth interface. This is of interest because it provides a second ground contact path from the machine frame which does not include the cable ground conductor.

Investigations into ground contact resistances of various large open pit shovels and drag lines on soils of various resistivity gave measured ground contact resistances of from 100 ohms to less than one ohm. Estimates of the contact resistance of smaller, crawler mounted machines such as bulldozers, miners and loaders, having smaller total weight and corresponding smaller contact area, indicate expected values of ground contact resistance to be in the range of one ohm to 10,000 ohms.

Instruments designed to monitor frame potential and/or leakage currents must not be adversely affected by frame-earth contact resistance in this range, and may, in fact, be capable of using it to some advantage.

OPEN PIT GROUND CONDUCTOR VERIFICATION

It became apparent during the Phase I and Phase II investigations of our contract that an instrument designed to easily monitor the condition of long, open pit mining machine grounding conductors would be of interest to the Bureau of Mines. It also appeared that the sensors and techniques developed during this program may have application in this area.

Looking at the machine frame grounding system from the machine end of the trailing cable gives two electrical paths to earth. The first is through

the cable ground conductor and the second is a parallel path from the frame to earth through direct frame-earth contact. It was desirable to design an instrument which can measure the cable ground path resistance regardless of the value of the parallel frame-earth path. Bendix did this through the use of a technique similar to that used to measure ground contact resistance of driven rods. Figure 3 shows a simplified schematic of the system.

If a small potential relative to earth is impressed on the frame of a piece of equipment in direct contact with the earth, a small current will flow. The ratio of the impressed frame potential to the resulting current flow would result in a measure of its earth contact resistance. If the earth contact is made by two separate paths, one directly to earth and one via the cable ground conductor, the ratio of the impressed frame potential to the current flow in only the cable ground conductor will result in a measure of only the cable associated earth contact resistance, including the ground conductor and ground bed resistances.

Measurements made in open pit metal mines by Pennsylvania State University indicate residual currents flowing in machine safety ground conductors due to induced voltages, capacitance effects, and ground potential gradients are generally less than 10 amperes, peak value. Since ground check measurements using this instrument will be made without the machine in use, values of residual currents flowing in the ground conductor at the time of measurement are expected to be quite low, with phase current being essentially zero. For this reason, the full scale sensitivity of the cable current sensor may be kept quite low, allowing for accurate detection of the instrument-induced current variations which are expected to be in the range of 0.2 to 1 ampere.

The small suitcase sized prototype instrument is designed to measure the safety ground return resistance level, as seen from the machine end of the cable. The instrument is intended for open-pit mine usage and the resultant ground return resistance measurment would include any and all paths to earth, as seen from the machine frame, including the total trailing cable, the power center safety ground bed resistance, and the effects of any additional equipment tied to the same safety ground bed. The measurement does not include the earth contact effects of the machine frame at the measurement location. Use of the instrument requires the installation of a pass-through current sensor around the safety ground conductor of the trailing cable and requires the availability of some good quality earth contact within a reasonable distance from the machine location.

A battery powered current source is connected between the machine frame and the current probe. The impressed current is limited to less than 1 ampere. If a good quality current probe electrode, such as a buried metallic pipe or heavy piece of fuel-powered equipment (bulldozer) is unavailable within

OPEN PIT GROUND CONDUCTOR VERIFICATION

SIMPLIFIED SYSTEM SCHEMATIC

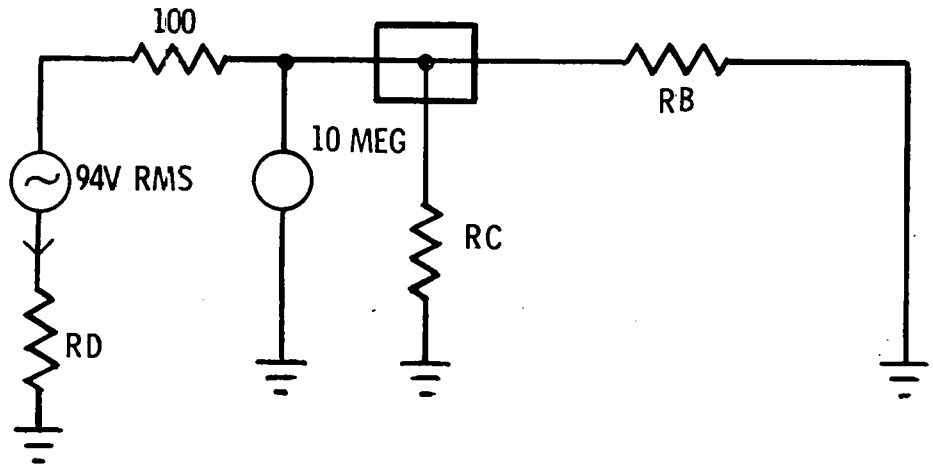


Figure 3

RB = Trailing Cable and Safety Ground Bed Resistance

RC = Machine to Ground Resistance

RD = Current Ground Rod Resistance

Let RA = RB||RC

Let R = RA + RD + 100

Current from Inverter = 94 ÷ R

Frame Voltage = Current X RA

RB = Frame Voltage ÷ Current in Cable

a distance of several hundred feet, a ground rod will have to be driven for use as the current probe. The current impressed between the current probe (earth) and the machine frame will divide and return to earth via the trailing cable ground conductor and the direct frame/earth contact. Figure 4 shows a simplified system block diagram.

The current flowing in the trailing cable ground conductor will be monitored using a pass-through current sensor. The electrical potential of the frame relative to an earth potential probe driven near the machine site is monitored employing a very high input resistance voltmeter (10 Meg).

The instrument current source is a standard 400 Hz inverter with a 100VA rating. A 100 ohm resistor has been placed in series with the convert output to limit the current to less than 1 ampere. Filtering is used to insure the system will measure only the 400 Hz components and not be influenced by 60 Hz stray currents.

CONCLUSIONS

Detection and prevention of electrical shock hazards is an ongoing program of great importance. The equipment and techniques described briefly in this paper are but one small effort.

The current field tests of the diode monitoring equipment will yield information as to actual operating environment characteristics of the system.

It is apparent that effort is needed to arrive at a viable way of establishing a local ground potential reference point in the mines to determine possible shock hazard conditions on the machines.

With respect to the Open Pit Ground Conductor Verification System, field tests are needed to prove out the equipment and techniques. It may be possible to use similar equipment to rapidly assess the condition of safety ground beds.

Continuing effort between the Bureau, industry, and the mining community will make the mines a safer and more productive operation.

ACKNOWLEDGEMENT

The information presented in this paper is based on work carried out under Bureau of Mines Contract J0357115 with Mr. Roger King as the Technical Project Officer.

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Feasibility Study of Electric Shock Prevention

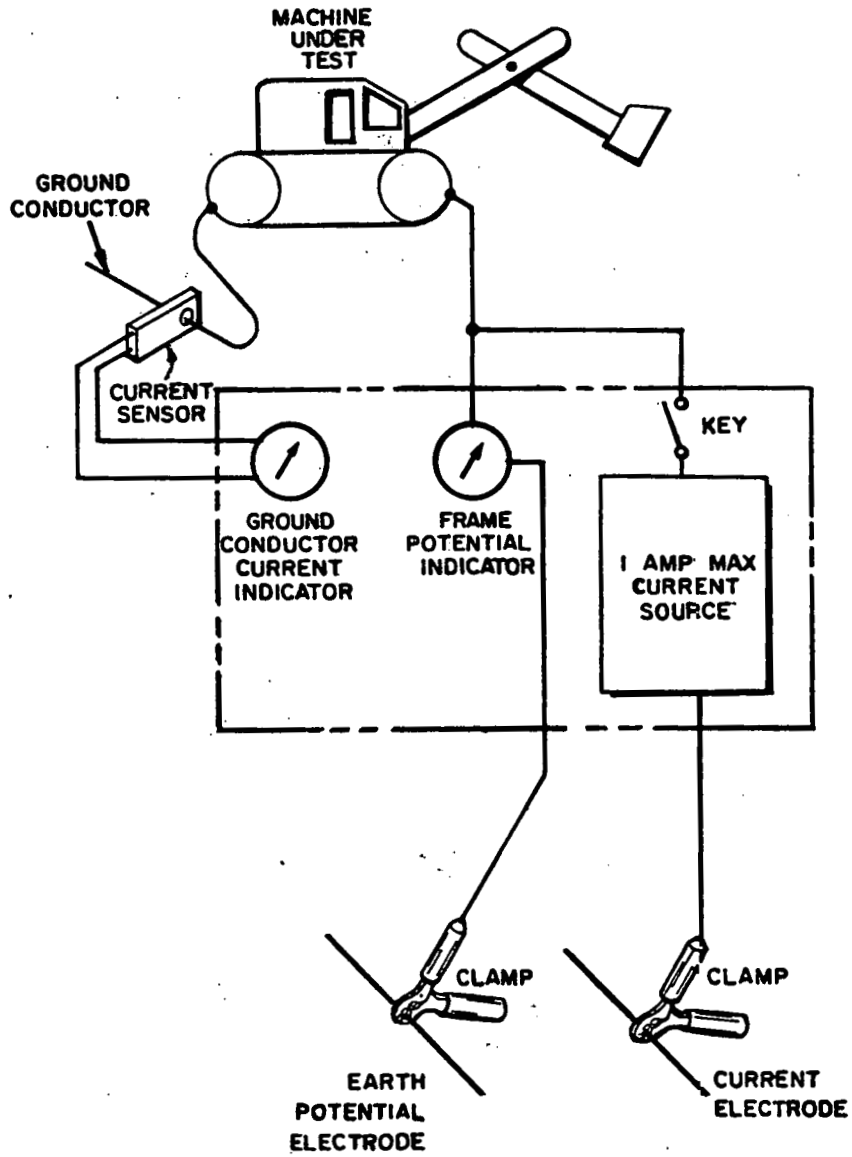


Figure 4

OPEN PIT GROUND CONDUCTOR VERIFICATION, BLOCK DIAGRAM

A FEASIBILITY STUDY TO IMPLEMENT A COAL MINE ELECTRICAL HAZARD PREDICTION SYSTEM USING A MICROPROCESSOR

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ABSTRACT

A microprocessor based system for use in underground coal mines is being studied as an in-mine tool for the prediction of incipient safety hazards from component failures of in-mine electrical power systems. A system designated as the Sequential Events Detection Analysis and Prediction System (SEDAPS) for the prediction of incipient electrical component failure is being developed at The Pennsylvania State University around a minicomputer framework (1). The microprocessor-based system being studied eliminates many disadvantages found in the prototype SEDAPS, which is a minicomputer based system, while maintaining its hazard prediction capability. The most attractive aspect of this design is the feasibility of installing it in and about the mine.

This paper presents the hardware system design of the microprocessor based SEDAPS along with a sketch of the microprocessor selection process. The system design is being developed by The Pennsylvania State University under a grant from the U.S. Bureau of Mines with Mr. G. J. Conroy as technical project officer.

INTRODUCTION

Incipient electrical component failure is a constant safety problem associated with in-mine electrical power systems. SEDAPS is being developed to reduce this problem by predicting component failure prior to the development of hazardous conditions. The present minicomputer-based system is too bulky and costly for use in the in-mine environment. The problem then, is to determine the feasibility of a microprocessor (μP)-based SEDAPS which can be used in and about the mine.

Incipient electrical component failures are predicted by sampling all of the voltage and current waveforms which provide power to each mining machine. A fast Fourier transform (FFT) is performed on each of these sampled waveforms. The results of the FFT's are then processed by a complex algorithm which compares the voltage-current characteristics of the components being monitored with the voltage-current characteristics of these same components in an undergraded state. The results of these comparisons are used in a pattern classification algorithm to determine the onset and type of safety hazard.

System Requirements

Adequate component monitoring requires that each of 24 voltage and/or current monitoring channels be sampled at a rate of 960 Hz for 133 m sec. Each 16-consecutive-sample group, of the 128 consecutive samples taken on each channel, must be processed through a 16 point FFT and through the complex failure prediction algorithm in an additional time of 266 m sec.

Hardware Design

The μP-based SEDAPS hardware shown in Figure 1 may be viewed as serving three functions. The sensors, amplifier, analog-to-digital converter (ADC), and ADC interface are responsible for signal acquisition. The μP and FFT hardware make up the signal processing portion of the system. The light emitting diode (LED) display and μP keyboard are used as a man-machine interface for indicating incipient failures and allowing manual interrogation of the μP.

Initial speed, accuracy and cost considerations limit the μP selection process to 16-bit μPs. Only 4 of the initial 9, 16-bit μPs under consideration survived the initial weeding out process which considered instruction execution times, availability of software and hardware support, and the availability of a fixed instruction set. The four remaining μPs, the CP 1600, microNova, TMS 9900 and WD 1600, were evaluated for both arithmetic and nonarithmetic speed. The arithmetic calculation time, as shown in column 2 of Table 1, was estimated by determining all of the arithmetic operations required by the failure prediction algorithm and using the instruction execution times of each μP. The arithmetic speed of each μP was enhanced by using the FFT hardware to perform all required multiplications rather than using each μP's multiply instruction. The data handling speed (i.e., nonarithmetic speed) of each μP was estimated by realizing a characteristic portion of the failure prediction algorithm using the instruction sets of each μP. The data handling time required by each μP is shown in column 3 of Table 1. The number of instructions executed by each μP in performing this characteristic program portion is shown in column 4 of Table 1.

Table 1

COLUMN NUMBER	1	2	3	4
μP	ASSUMED CLOCK FREQUENCY (MHz)	ARITHMETIC CALCULATION TIME (m sec.)	CHARACTERISTIC DATA HANDLING TIME (m sec.)	CHARACTERISTIC INSTRUCTION COUNT
TMS 9900	3.0	45.8	61.8	12,601
CP 1600	2.5	59.1	62.5	23,220
WD 1600	3.0	67.9	74.4	17,606
microNova	2.08	37.8	129.7	46,305

Table 1 shows that, for the SEDAPS application, the TMS 9900 and CP 1600 are closely matched in terms of speed. The TMS 9900, however, requires fewer instructions and does not require the hardware and software complexity necessary when the FFT hardware is used to perform

high-speed multiplications. The TMS 9900 was chosen for the SEDAPS application. Realization of the entire failure prediction algorithm using the TMS 9900 instruction set indicates that the program execution time is approximately 224.65 m sec. (2).

It was quickly determined in the microprocessor selection process that no μP is capable of performing all of the FFT arithmetic calculations within the time allotted. A high-speed hardware implementation which shares the system memory was therefore found to be necessary. Sharing of the system memory is accomplished by the start, hold and hold acknowledge control lines shown in Figure 2. The start line allows the FFT hardware to begin processing data. When the FFT hardware needs access to the system busses, it sends a hold level to the μP. The μP electronically disconnects itself from the system busses and sends a hold acknowledge level to the FFT hardware. The FFT hardware then electronically attaches itself to the system busses and accesses memory. The FFT hardware relinquishes bus control by detaching itself from the system busses and then removing the hold level to the μP.

The FFT hardware is required to perform 192, 16-point FFT's at the end of each sampling period. Equivalently, it must perform 36,864 additions and 24,576 multiplications in a few tens of milliseconds in order to meet the time requirement. A preliminary FFT hardware design and timing diagram that meet this criteria are shown in Figures 3 and 4.

The read only memory (ROM) of Figure 3 is used to store the instructions which are required to perform one butterfly. A butterfly is a set of calculations which can be used 32 consecutive times to realize a 16-point FFT. The synchronous load N-bit counter is used to access these instructions in the proper sequence. The data address generator of Figure 3 insures that the correct data is available at each stage in the butterfly computation. The 12-bit multiplier-accumulator is the arithmetic element which is controlled by the program storage ROM and performs the calculations required by each butterfly. The SIN/COS ROM provides the sine and cosine constants used in the FFT computation. The BFYDON and ALLDON flags are used to signal the end of each butterfly and the end of FFT calculations respectively. The control exercised by the BFYDON and ALLDON flags on the FFT hardware may be ascertained by inspection of Figure 4. Note that the 4N to N multiplexer in Figure 3 is used to supply the starting address of the various routines stored in the program storage ROM. Using a 5 MHz clock with the FFT hardware permits the FFT calculations to be executed in 14.75 m sec. (2). This time is well within the calculation time permitted.

SUMMARY

The work discussed here shows the feasibility of using a μP as a compact in-mine tool which can meet the speed required for the prediction of electrical component failure. It can also be shown that this μP-based system can meet the arithmetic error constraints required of such a system (2). Although problems such as dust in the mine environment and power supply regulation have yet to be fully investigated, it is felt that any problems in system packaging can be overcome with a little engineering effort.

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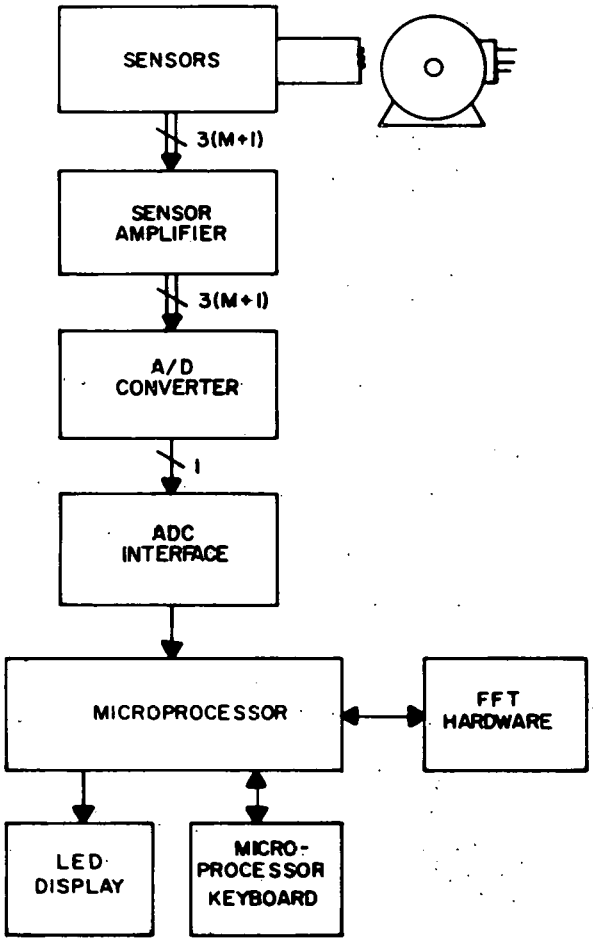


Figure 1. Block Diagram of the Microprocessor-Based SEDAPS Hardware.

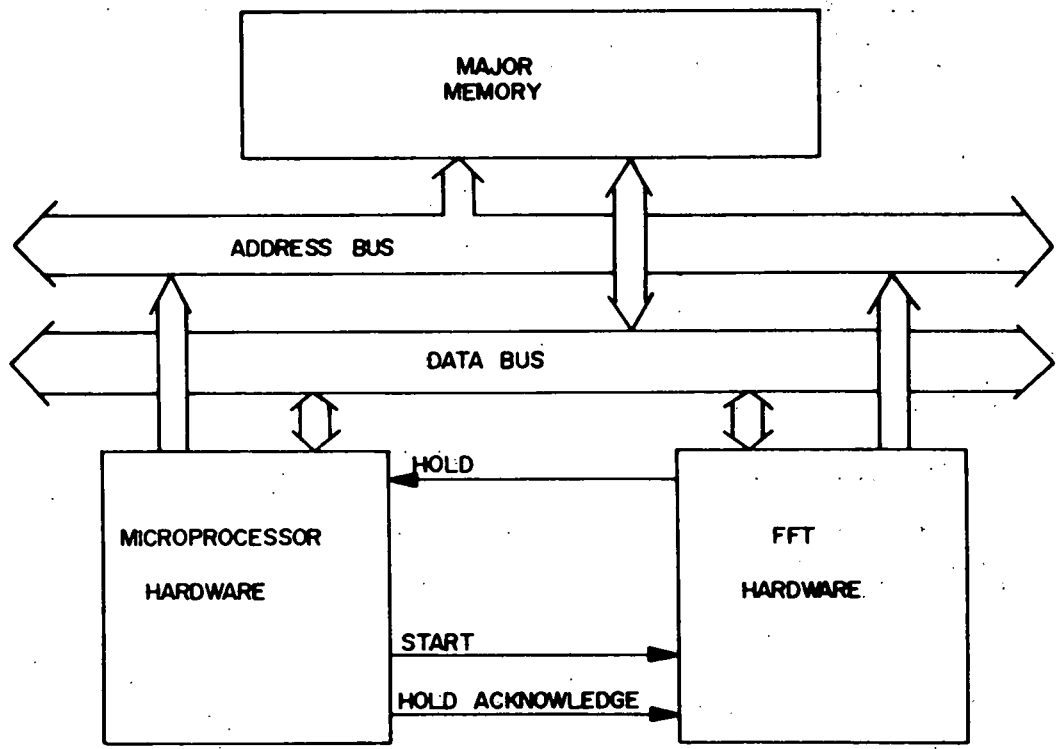


Figure 2. Microprocessor-FFT Hardware Block Diagram.

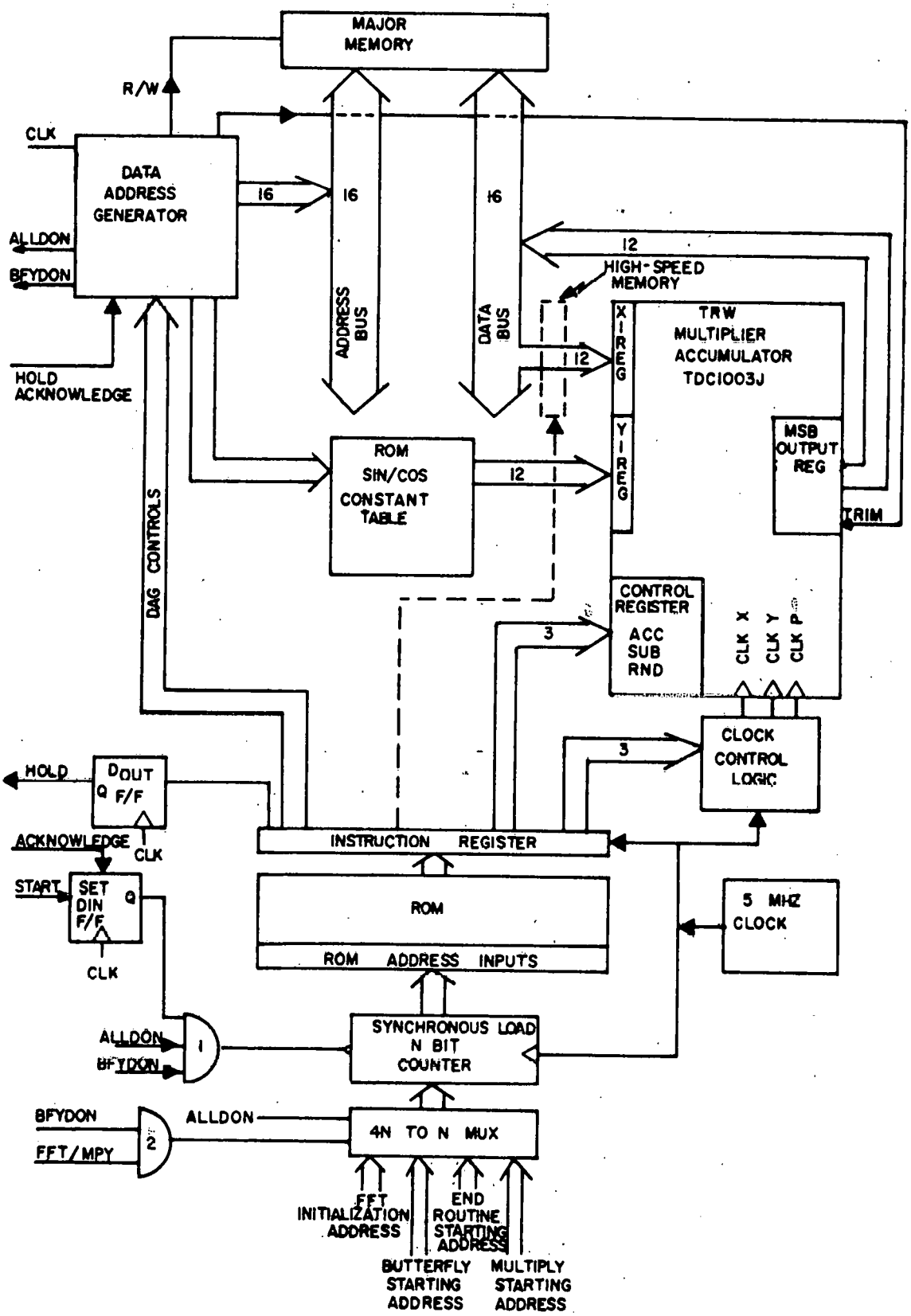


Figure 3. Preliminary FFT Hardware Design.

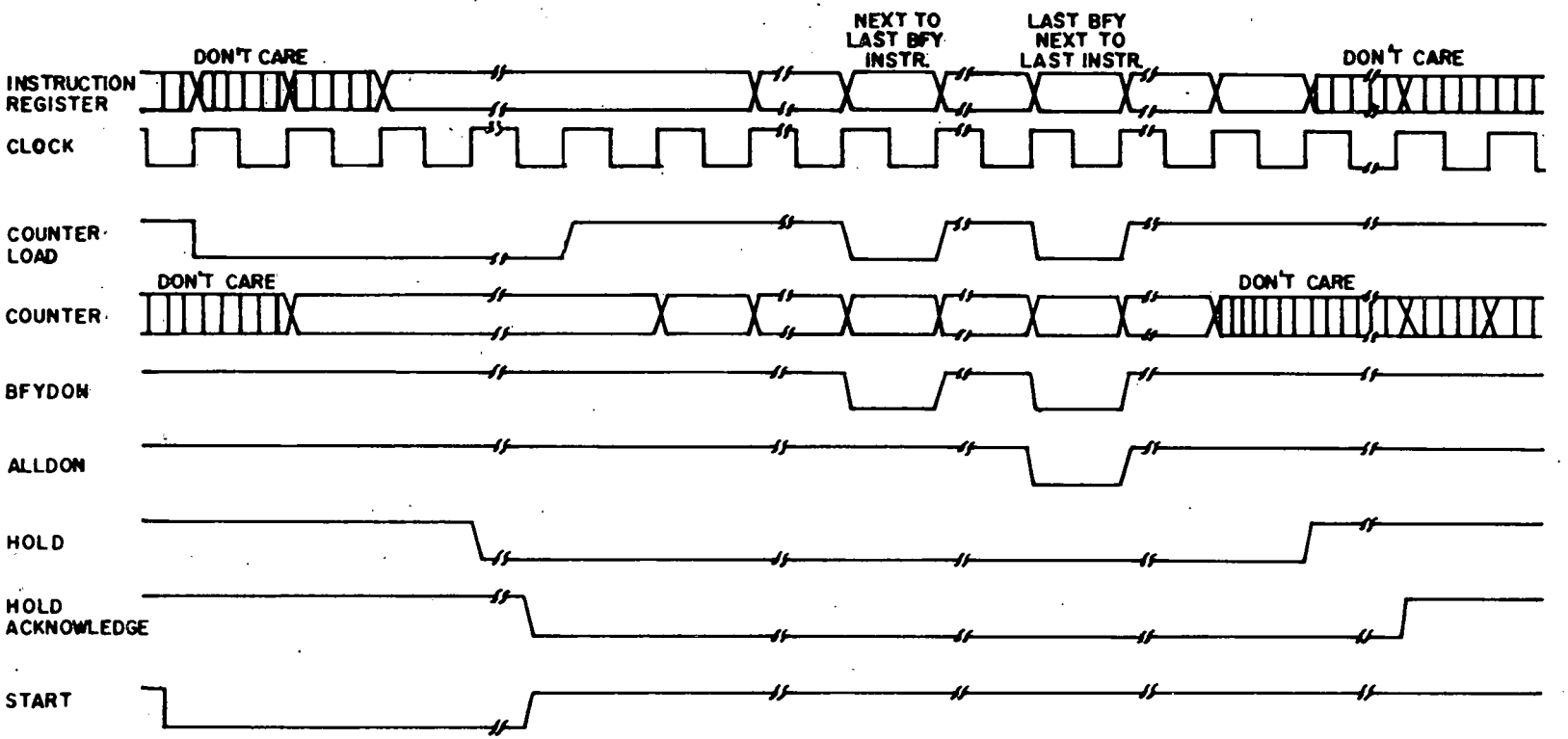


Figure 4. Preliminary FFT Hardware Timing Diagram.

MONITORING ELECTRICAL GROUND BED CORROSION

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ABSTRACT

Electrical grounding is essential for the safe operation of power systems, since it reduces the possibility of dangerously high potentials in human work areas. A faulty ground therefore poses a potential hazard. One reason for faulty ground connections is ground bed corrosion. The easiest way to detect corrosion is by inspection; however, in a working electrical power system the ground bed must remain intact at all times. The purpose of this study is to devise a non-invasive monitor for ground bed corrosion. Five possible methods for non-invasive measurement of ground bed corrosion are proposed: ground bed-to-soil potential; bed resistance; current integration; ultrasonics and soil resistivity. The study of these five methods suggests current integration, bed resistance, and ultrasonics to be the most reliable and feasible.

INTRODUCTION

Grounding is necessary in power systems to prevent dangerous potentials from appearing in human work areas. Connection of equipment frames to a low resistance ground is designed to keep the potential difference between equipment frame and earth small. If this ground connection becomes severed or faulty, a large potential may appear from equipment frame to ground, allowing current to flow through man. Current flow will generally be from hand (frame) to foot (earth), passing directly through the heart. Ground bed corrosion is but one possible way safety ground connections can become faulty. Development of a monitor to evaluate ground bed destruction, due to corrosion, eliminates yet another potential hazard.

All corrosion is electrolytic in nature. Energy, in the form of electrons, is used for chemical reactions at the metal surface. Because corrosion is electronic, a corrosion circuit, or cell, is formed. Potentials developed at the metal surface give rise to a current (electron flow) causing corrosion. The mere presence of a drop of water on a piece of iron sets up an electro-chemical cell causing current flow, thus corrosion.

Ground bed corrosion occurs when the bed becomes a current sink. Interconnected ground beds composed of two or more metals set up a galvanic corrosion cell. Two metals, each at a different energy level, cause a potential difference. This potential difference provides an electron flow (current) necessary for corrosion reactions. Leakage currents in an electrical ground system also contribute to ground bed corrosion. These currents result from potential differences in ground bed points. In an electrolytic medium the corrosion rate of a metal is proportional to the current flowing from that metal (1). Observations of coal mine electrical ground systems indicate from 0-10 amperes flowing during normal mine operations (2). By law current flow in excess of about 12 amperes is considered unsafe and determines a fault condition. However, currents below the ground fault trip point are unrestricted (3). The possibility of current flow in the ground system less than the ground fault trip point for extended periods of time increases the likelihood of severe corrosion.

CORROSION PROCESS

Three things are needed to define electrochemical corrosion of a metal. The electrochemical system must have an anode and cathode connected by some ionic medium. To complete the electrical circuit the anode and cathode must be externally connected with a low impedance path or source.

The corrosion process is electrochemical because of the continual change between electronic and ionic flow. Electrons flow from anode to cathode while conventional current flows from cathode to anode. Corrosion occurs at the anode-ionic interface through the process of oxidation, the removal of electrons from metal atoms. Examples of oxidation reactions are $Zn \rightarrow Zn^{++} + 2e^-$, $Al \rightarrow Al^{+++} + 3e^-$, and $Fe^{++} \rightarrow Fe^{+++} + e^-$. These freed electrons travel through the low-impedance path to the cathode where they are involved in the complementary process of reduction. Examples of reduction reactions are $H^+ + e^- \rightarrow \frac{1}{2}H_2$, $Cu^{++} + 2e^- \rightarrow Cu$, and $Fe^{+++} + e^- \rightarrow Fe^{++}$. Thus the simultaneous oxidation-reduction reactions complete the electrochemical circuit.

The speed and direction of electron flow along the external connection will determine the corrosion rate for each metal. Insertion of an electronic source increasing electronic movement toward the cathode results in increased corrosion. A steady flow of electrons in a cathodic direction (dc current) is very conducive to corrosive action. If alternating current (electron flow) is applied, electron flow alternates every half cycle. This means that each electrode alternates from an anodic to a cathodic state. Ideally with the application of pure ac neither electrode is purely anodic or cathodic, thus no corrosion should occur. Realistically the presence of a pure ac current reduces a metal's corrosion rate typically to 1% that of an equivalent (RMS) dc current (4). The tendency of a metal to corrode is related to its reduction potential, which has been determined for many substances (see Table I). The more negative the reduction potential the greater the corrosive activity.

TABLE I. Reduction Potentials of Some Common Metals. Increasing Reduction Potential Denotes Decreasing Corrosion.

Au^{+}_{+2}	(gold)	+1.68
Pt^{+2}	(platinum)	+1.20
Cu^{+2}	(copper)	+0.34
H^{+}	(hydrogen)	0.00
Pb^{+2}	(lead)	-0.12
Sn^{+2}	(tin)	-0.14
Fe^{+2}	(iron)	-0.41
Cr^{+2}	(chromium)	-0.56
Zn^{+2}	(zinc)	-0.76

Faraday observed a linear relationship between charge and metal dissolution. Subsequently, he developed an equation to predict weight loss in an electrolytic cell. Weight loss (W_L) is equal to the molecular weight of the metal (M) times the charge passed (Q) divided by Faraday's constant (F = 96,500 coul/mole) and the metal's valence number (V), $W_L = MQ/FV$.

From Faraday's equation the weight loss is proportional to charge. Faraday's equation only becomes valid given two conditions; 1) all electrons passed must be involved in metal dissolution; 2) the metal must be pure. Use of alloy metals and the presence of side reactions introduces error into the Faraday equation.

FIELD OBSERVATIONS

Ground Currents

Studies done by Penn State University on mine power system grounds revealed the presence of ac and dc current during normal mine operations. Since the dc component is most destructive one can neglect ac ground current. The assumption that ac corrosion can be overlooked comes from electronic theory, but one must be cautious. Mechanisms of ac corrosion are not well understood and it is the opinion of some that ac corrosion may be just as dangerous as dc corrosion (5). Ac current in the ground system can be attributed to asymmetries in the massive power supplies used in the mine industry. On the other hand, dc current is generally contributed from one source, dc haulage systems leakage.

To study dc haulage leakage current an experiment was conducted at Bruceton PA, site of a US Bureau of Mines test center. Installed to evaluate dc haulage equipment, Bruceton possesses an above-ground dc haulage track. The Bruceton test track contained no rail bonds, suggesting maximum leakage current. Measurements here revealed dc leakage of 0-2 amperes during train operations.

Rod Corrosion

Previous studies done in the laboratory revealed destructive

ground corrosion with modest dc current flow. Observations of ground bed corrosion in the field is a chance occurrence. Destruction of ground beds, for whatever reason, is not readily admitted. One observation of ground bed corrosion was recorded and is presented in Figure 1. Pictured is the top section of an eight foot by one-half inch Copperweld[R] ground rod, severed by corrosion.

CORROSION DETECTION

Introduction

The object of this paper is to devise a way to non-invasively monitor ground bed corrosion. The easiest way to detect corrosion is by examination; however, in working electrical systems the ground bed must remain intact at all times. Therefore the search began for possible alternatives to evaluate corrosion progress. After an extensive literary search, five measurable parameters existed; these five appear in Table II.

TABLE II
Evaluation of Ground Bed Monitoring Techniques

Monitoring Technique	Non-Invasive	Accuracy	Corrosion Prediction	Corrosion Monitor
Ground Rod - Soil Potential	5	3	3	0
Soil Resistivity	5	5	3	1
Current Integration	5	5	4	3
Ultrasonics* (rod length)	5	4	0	3
Ground Rod Resistance	5	4	3	2

(*) Ultrasonics is only applicable under certain ground bed conditions.

All five methods are rated from 0-5, (five being the best) evaluating their ability to be used non-invasively; to give reliable results; to predict the onset of corrosion; and to monitor corrosion progress. These ratings are derived from literary information and practical experience, their use is solely for narrowing possible alternatives.

All possible schemes adapt themselves easily to working ground beds with some reservations. Accuracy, for the most part, is good for all methods, but the last two columns narrow the field. The existence of corrosion depends on the presence of two conditions; adequate electrolytic contact and current flow. All methods, except length detection, monitor one or both conditions. The thesis given assumes corrosion exists; the problem is one of retardation and evaluation and therefore column 3 is irrelevant. Column 4 reveals four possible solutions which will be discussed in detail. Reasons

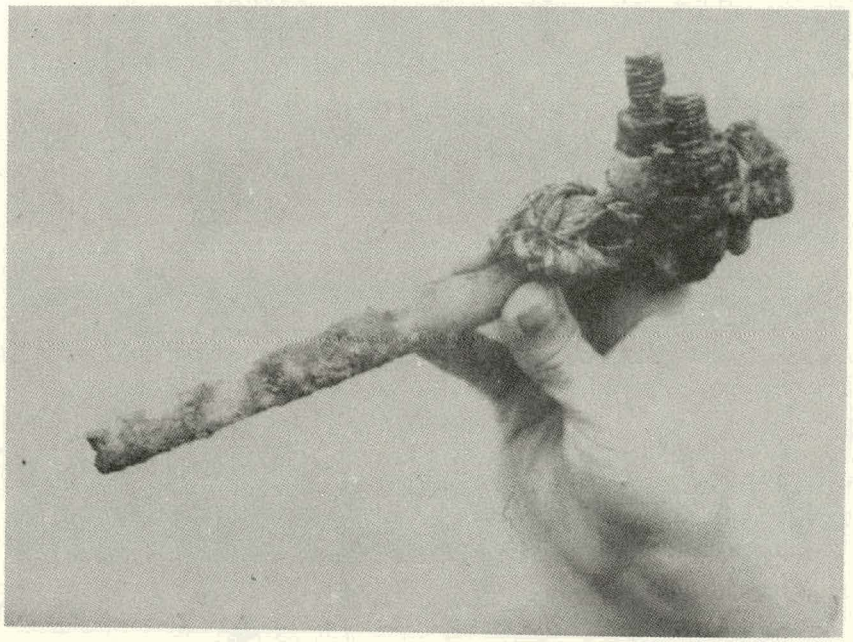


Fig. 1. Severed Ground Rod due to Corrosion.

for deleting the remaining solution follow:

Ground bed-to-soil potential measures the potential of ground beds with reference to a reference electrode (e.g. standard copper-copper sulfate, $Cu\text{-}CuSO_4$) close by. The numerical value obtained gives only an indication as to the type of corrosion present, if any (6). As long as the metal is available for corrosion the potential will prevail. Therefore this method is not useful for monitoring duties. Rod-to-soil resistance, although considered possible for corrosion monitoring, will not be discussed further. Work in this area is presently being conducted for the USBM at West Virginia University. To prevent multiple investigation discussion will be limited to the soil resistivity method, a relative of rod-to-soil resistance.

Soil Resistivity

Background

By induction one can ascertain that corrosion will be related to soil resistivity or, inversely, soil conductivity. Corrosion rate (Cr) is proportional to the amount of current (I) passed, which is proportional to the soil conductivity (K). Therefore the corrosion rate is inversely proportional to the soil resistivity (7). Increasing soil resistivity decreases the ability for current to flow, thus decreasing corrosion; conversely, a decrease in soil resistivity increases corrosion. By measuring the soil resistivity of a ground bed one can predict the likelihood of corrosion taking place (8).

The mechanism of metal corrosion calls for the dissolution of metallic ions into the electrolyte. The increase of metallic salts due to corrosion theoretically should lower the surrounding soil resistivity. Therefore the hypothesis is that by monitoring the soil resistivity one could observe ground bed corrosion. In order to investigate such a hypothesis tests were conducted using a simulated ground bed constructed in a wooden box.

Simulated Ground Bed Resistivity Tests

Using the simulated ground bed, resistivity measurements were taken before and after rod corrosion (9). Measurements were made using the Bison resistivity meter and the Wenner electrode configuration. Limited ground bed size dictated the use of small (#10), solid copper wire electrodes for resistivity measurements.

Because the major interest was in observing the effect of corroding rods on soil resistivity it was necessary to minimize all external factors. Several factors affect soil resistivity with time. Of these, moisture and temperature have a major effect (10). Temperature variation posed little problem since the ground bed simulators were kept indoors in a fairly constant-temperature environment. Moisture content proved to be the largest error factor. Soil moisture was lost rapidly due to the dry indoor climate, causing the soil to harden and crack. A plastic cover was used to retard evaporation.

As a secondary precaution a control box was used for all resistivity measurements to monitor the effect of moisture loss. The control box was prepared and measured in the same manner as the corrosion cell except for the absence of a ground rod and current.

Before and after corrosion, resistivity measurements were conducted for Copperweld^{R} and steel rebar electrodes. Both ground rods remained in the soil during corrosion box measurements to simulate actual conditions.

Distance resistivity profiles done before and after corroding ground rods revealed some known facts but did not conclusively show corrosion activity. Soil resistivity measurements are subject to great variability and error. Distance resistivity profiles done for the Copperweld^{R} ground rod clearly showed an increased soil resistivity near the rod after corrosion (see Figure 2). This phenomenon can be attributed to a loss of soil moisture due to heat produced during current passage and electroendosmosis (movement of water molecules in an electric field). Because of the encircling cathode the assumption is made that corrosion products will radiate equally in all directions from the anode, thus one would expect a corresponding symmetrical decrease in soil resistivity. In the X-axis Copperweld^{R} profile it appears that the soil resistivity is lowered due to corrosion, but the Y-axis profile does not indicate the same result. Therefore the reason for the resistivity drop is uncertain. A look at the corresponding control data shows resistivity to remain fairly constant; if anything, resistivity decreases with time. The control data shows no substantial decrease in resistivity at the box center confirming loss of moisture during corrosion (see Figure 2). Rebar distance resistivity profiles showed little change between before- and after-corrosion values. This, along with a fairly constant control, suggests insufficient time for corrosion activity to be a factor (see Figure 3).

It appears that soil resistivity changes occur over long time periods. Even though the rebar rod had undergone significant corrosion activity no evidence was visible in the resistivity readings. The large variability of resistivity readings makes analysis of profile graphs mostly speculation. In the past, investigators, e.g. G. N. Scott, have used statistical methods in an attempt to recover trends in soil resistivity (11). Scott showed that a plot of resistivity vs its cumulative frequency on log-probability paper helped predict corrosion trends (12). He found that by plotting the log of resistivities he was able to obtain a normal distribution. According to Scott, when a soil's resistivities are normally distributed it follows that such a soil is "uniform." In an attempt to analyze data from simulated ground bed resistivity distance profiles, Scott's methods were employed. Plotting resistivity vs cumulative frequency on log-probability paper did not produce a normal distribution. Therefore all graphs were constructed on standard probability paper for comparison. Figures 4 and 5 show before- and after-corrosion plots for Copperweld^{R} and rebar rods respectively. Analysis of both probability representations show a decrease in mean resistivity (numerically

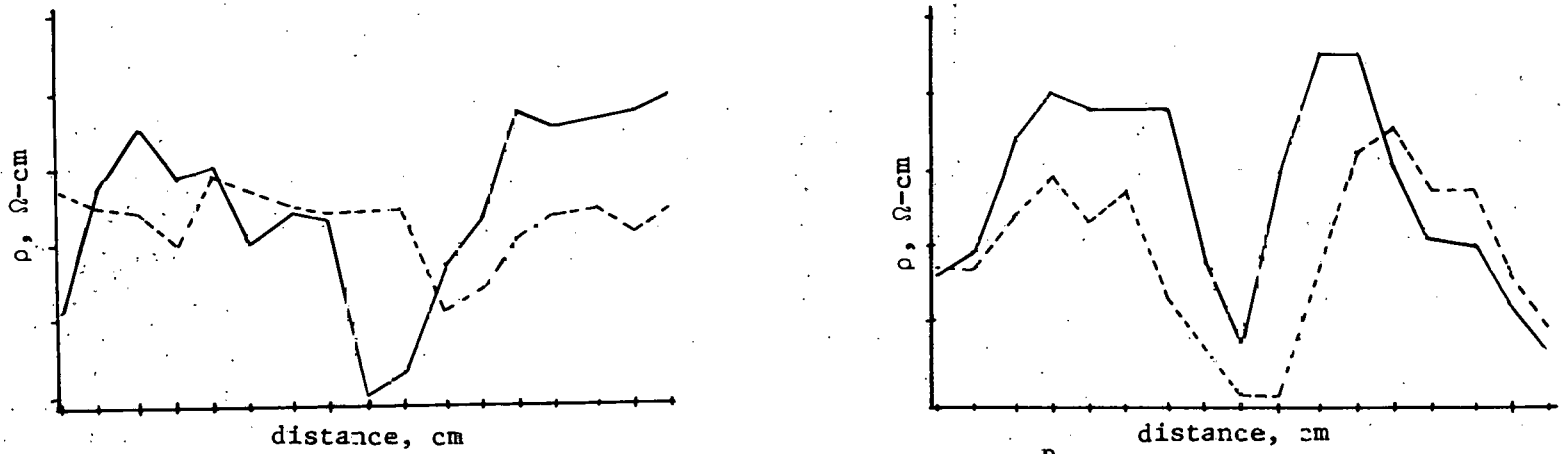


Figure 2. Distance resistivity profile for the Copperweld^{R} ground rod and control.

Before corrosion ——— After corrosion - - - - - - - -

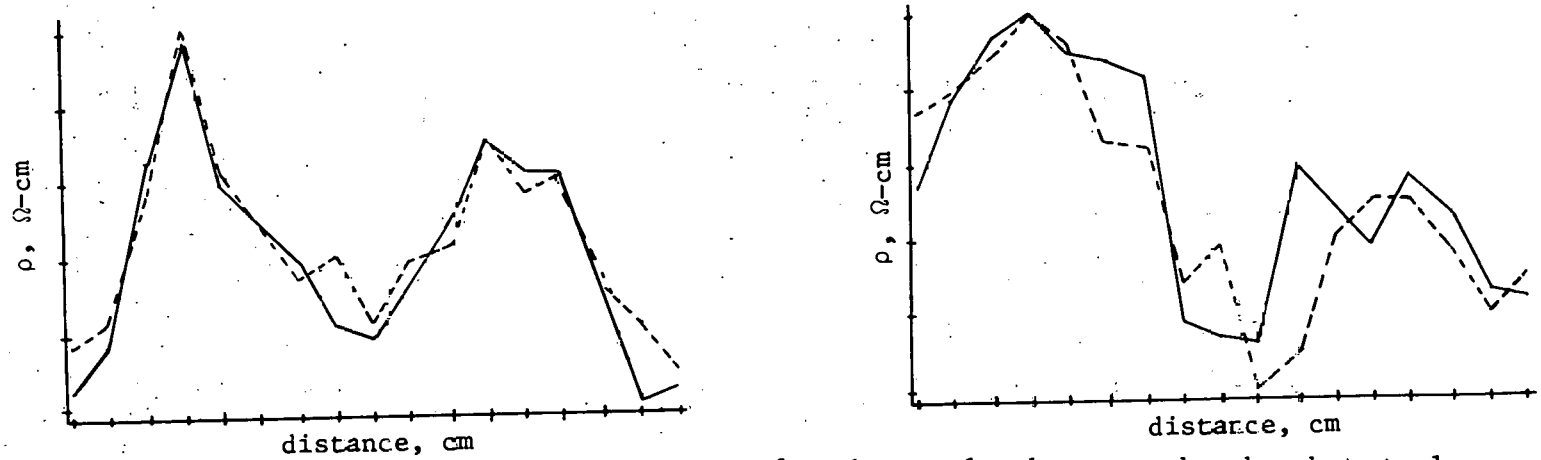


Figure 3. Distance resistivity profile for the steel rebar ground rod and control.

Before corrosion ——— After corrosion - - - - - - - -

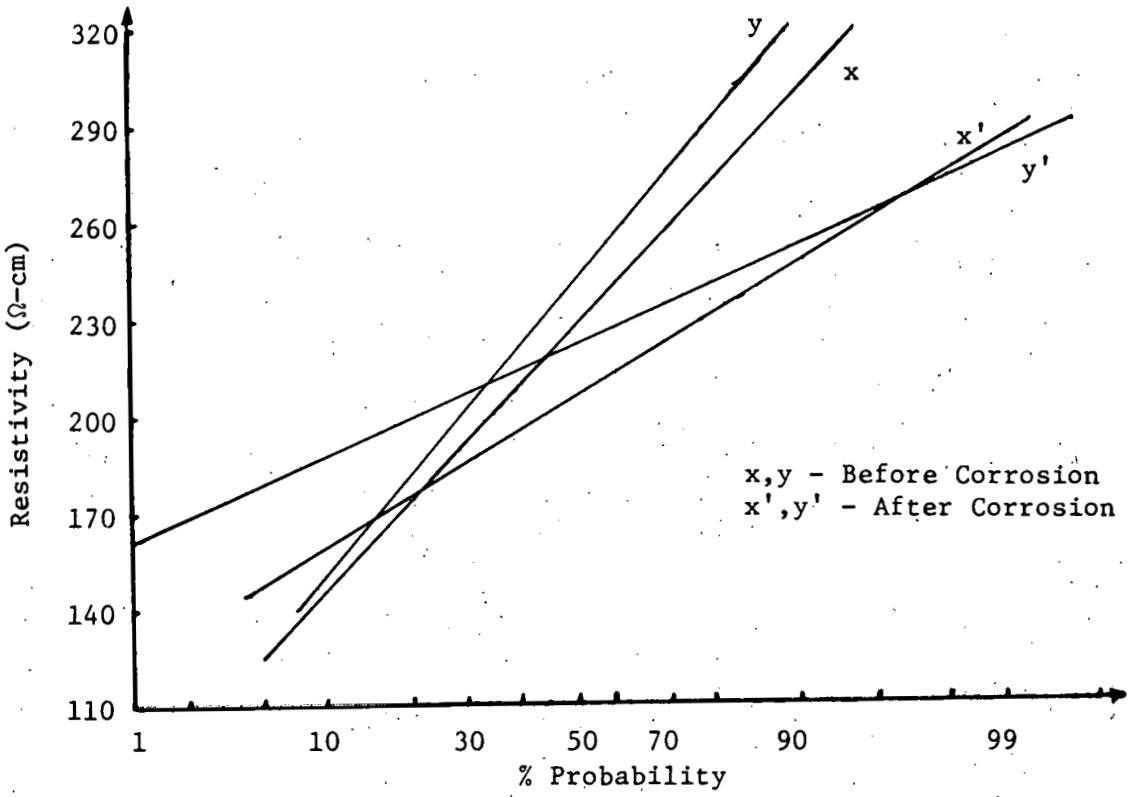


Figure 4. Soil Resistivity vs % Probability, Copperweld^{R} Rod.

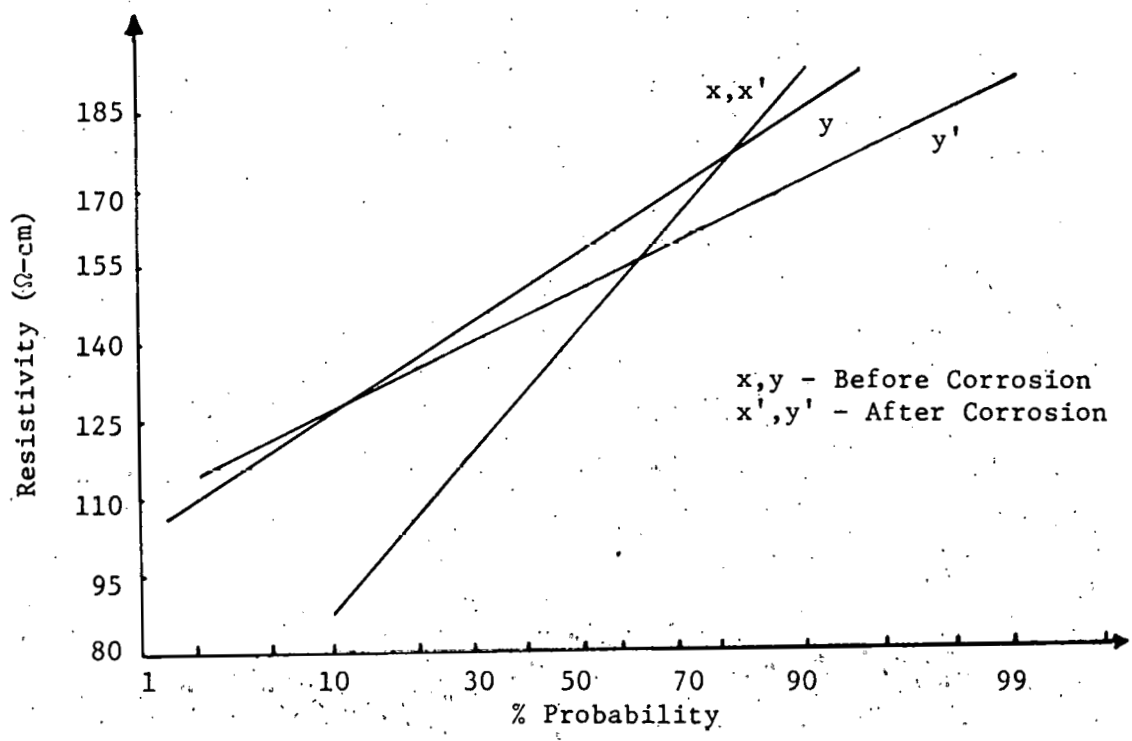


Figure 5. Soil Resistivity vs % Probability, Rebar Rod

found at 50% probability) following corrosion (one exception being the slight difference in X-axis data for the rebar rod).

Current Integration

According to Faraday there is a direct relationship between current and metal loss. Theoretically, under ideal conditions, this law is correct. Due to the presence of side reactions, metal alloys, and erosion, Faraday's law, as applied to metal loss, is somewhat in error. Monitoring charge (corrosion current integration) is a good indicator as to the rate of metal dissolution, however. For pure metals theoretical calculations are very close to actual measured weight loss. With other electrodes, measured corrosion rates serve as guides for weight loss determination.

Integration of dc current for long time periods proves to be a difficult task. Integration of ac current is purposely neglected; unless there exists a dc offset, the definite integral of an ac signal over an even number of periods will equal zero. Several methods exist for dc current integration. Of the available integration methods, several were employed for use in the corrosion experiments.

Instrumentation

A dual-trace Gould 110 chart recorder and a Simpson Multicorder were used to provide continuous current recordings of all corrosion experiments. Numerical integration was carried out using an HP-97 programmable calculator with a standard numerical integration program. This method proved to be very laborious and time-consuming. The Gould 110 recorder was also equipped with the latest in long-term integration techniques, i.e. an alternating input polarity integrator. This device uses the standard analog integration method. When a predetermined maximum and minimum voltage is reached, however, the input polarity is reversed, thus continuous charging and discharging of the integration capacitor takes place. Integration output becomes a series of pen traverses contained on a small recording strip along the chart paper edge. After calibration one simply counts the number of traverses and multiplies by the appropiate scale factor. In order to physically count traverses without severe eyestrain the chart speed must be increased, yielding enormous amounts of paper. Aside from its use in checking numerical integration accuracy, this means of integration proves too awkward for use.

An electronic counter was built to sum a frequency proportional to the current. Current is converted to a voltage using a precision shunt; this voltage is amplified, filtered, and presented to a voltage controlled oscillator (VCO) where it is changed into a proportional frequency. Digital decade counters are used to sum the VCO cycles over some period of time. The least significant bit (LSB) of the binary counters is incremented every period (cycle) of the VCO. Time periods used in corrosion experiments lasted from days to weeks.

Ultrasonics

If the corrosion process were uniform, current integration would be very useful. Unfortunately, current integration yields an average weight loss regardless of the uniformity of corrosion (13). If corrosion tends to be localized the threat of rod separation becomes the main concern. The problem is to find a way to detect localized corrosion non-invasively. Once again an analogous situation from industry provides a possible solution.

For a long time plant engineers had no accurate way to detect corrosive weakening in exposed pipelines. Previously sample holes were drilled and pipe thickness recorded by use of a micrometer. This sampling technique was far from accurate and very invasive, necessitating periodic pipeline shut-downs and repairs (14). Today ultrasonics is used to measure metal thickness and thus corrosion progress.

Background (15)

Ultrasonics represents any frequency of sound higher than human hearing. High frequency vibrations are introduced to the exposed metal surface using a piezoelectric device and an appropiate sound-coupling media e.g. (water, grease). Depending on the properties of the material under study, the sound wave will propagate until a second interface is encountered; at this point some sound waves will be reflected back toward the transducer.

Applications and Limitations (16)

In applying ultrasonic measuring techniques to ground beds four very important requirements must be met: (1) the ground bed must consist entirely of ground rods, capable of propagating sound, (2) tops of ground rods must be exposed above ground, accessible for measurements, (3) tops of ground rods must be perpendicular to the rod body, and (4) the ground rod must be solid and continuous (welds and threaded joints are unacceptable).

Prior to this ultrasonics has been used extensively to measure material thickness. Non-invasive measurement of ground rod length is proposed here. If ultrasonic laws hold true, placement of the transducer on top of the rod would send a signal to the rod base which would reflect back. Using the techniques described, the length could be read directly. Corrosion or rod separation anywhere would alter the length reading. In the case of a severed ground rod, length readings would be less than expected. For general rod corrosion, multiple as well as base reflections would be expected (17).

Using available equipment, a preliminary feasability study was made of ultrasonic length measurements for ground rod corrosion. The pulse-echo method was chosen to measure various length ground rods in differing states of corrosion. Equipment employed was the Branson, model 303, Ultrasonic flaw detector. Initially the instrument was

calibrated to a standard, out-of-soil, eight foot ground rod; showing that the instrument was capable of measuring a rod of such length. Next a one foot section of rod, out-of-soil, was measured without recalibration. As expected, the instrument recorded a one foot length. In order to determine the effect of corrosion on ultrasonic readings the instrument was calibrated to an uncorroded, out-of-soil, one foot rod (see Figure 6). Subsequent testing of out-of-soil, one foot corroded rods revealed multiple reflections, the first being located at the point of corrosion (see Figure 7). The effect of soil on ultrasonic length measurement was observed using the simulated ground beds previously described. No difference was noted between observations made out-of-soil and those within the soil environment. An experimental ground bed nearby provided measurements of a driven, eight foot ground rod. The top of the ground rod was prepared using a metal file and the instrument was calibrated with an eight foot, out-of-soil ground rod. Measurements of the in-soil rod revealed markable signal attenuation, possibly due to an over-abundance of earth moisture. However, with an amplifier gain of 80 dB, a signal at the eight foot mark was clearly visible. An additional signal appeared approximately 1/3 of the way down the rod. This signal could be attributed to corrosion, rod bending, or a large water pool.

These preliminary tests show that ultrasonic length measurement provides a reliable method of detecting ground rod corrosion. Instrumentation procedures are easily learned and rugged industrial equipment is available. Problems of attenuation can easily be overcome by using more powerful transducers, signal generators, and receiving amplifiers.

CONCLUSIONS

Since it was impractical to design a ground bed totally immune to corrosion, non-invasive monitoring of corrosion rate was investigated. Five monitoring schemes were suggested; soil resistivity, current integration, rod-soil potential, rod-soil resistance, and ultrasonics. Results show that current integration teamed with ultrasonic length measurement provide the most thorough monitor. Integration of current injected into the ground bed allows calculations of metal loss. Ultrasonic length measurements indicate the severity of localized corrosion. Because of the nature of the ultrasonic measurements, special ground bed designs are required. Ease of long-term dc current integration was demonstrated with the construction of an electronic current integrator. Use of electronic LSI memory units in the integrator allowed storage and recall of many hours of integration history.

Soil resistivity and rod-soil potential measurements are too uncertain to be considered as practical monitoring schemes. Determination of corrosion using soil resistivity and rod-soil potential is presently more of an art than a science. Results from measurements vary greatly allowing only the observation of possible trends. Such measurements have, in the past, been used to determine if an area is susceptible to corrosion (18). Continuous rod-soil resistance measurement provides the most practical of the five schemes; however, this method will not predict oncoming ground bed failure due to corrosion.

Current integration in this report relied on precision shunts as pick-up devices. MSHA regulations regarding shunts in the ground system require that all shunts be able to handle maximum fault current (19). As a shunt's current rating is increased, its generated voltage drop is proportionally decreased. Since ground currents are typically only a few amperes (0-10 A) the shunt's voltage drop is extremely small and difficult to measure. Non-invasive current transformers are useful for low ac currents; however, in the past no practical low current dc device was available. Recently the Hall effect has been employed to non-invasively measure both ac and dc currents (20). Equipped with such a device, non-invasive measurements of low dc ground currents is possible.

ACKNOWLEDGEMENT

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DOUBLE INSULATED DRILL TEST

by

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ABSTRACT

This report examines the double insulated system as an alternative to frame grounding and the hazards which could possibly occur when using the drills in a mining environment The purpose of this examination is to determine the integrity of the double insulated system when subjected to conditions found in coal mines.

INTRODUCTION

Periodically, the question comes up concerning the use of a separate grounding conductor for the exposed, noncurrent carrying metal parts of double insulated drills. Coal and metal and nonmetal mine health and safety regulations (75.702, 77.702, 55, 56, and 57.12-25, 30 CFR) allow the Secretary or his authorized representative to permit grounding methods other than frame grounding if such methods provide no less effective protection.

In order to determine the effectiveness of double insulation systems, the Mining Enforcement and Safety Administration in Coal Mine Health and Safety activity requested the Mine Electrical Systems Branch, Pittsburgh Technical Support Center, to evaluate the Underwriters Laboratories (UL) standards for double insulated drills as well as the adequacy of double insulation protection under the adverse conditions present in underground coal mines.

Using UL's UL45 standard for portable electric tools as a reference, a detailed set of four tests was arrived at for examining double insulated drills for mine use. These tests are more rigorous than UL's UL45 standard and require the drill to be tested under more adverse conditions.

Two sets of tests 1 through 4 were run. The first set was run on drills direct from the factory (unconditioned drills); the second set, on drills conditioned with a water and coal dust mixture (coal-conditioned drills).

The 3/8 inch drills tested were all Ingersoll-Rand's model 8020, 115-125 volts, 1,000 rpm no load, 3 amp.

DIELECTRIC WITHSTAND TEST

The drill was loaded by a 3-hp, dc motor until the current input of the drill equaled the rated current input of the tool (fig. 1). The drill was continuously operated under full load until the temperature of the drill became constant. At this time the following test was conducted.

The tool shall withstand for 1 minute, without breakdown, the application of a dc potential, as illustrated in figure 2, in accordance with the following:

1. Between accessible dead metal parts and live parts: 3,500 volts plus twice the rated voltage of the tool.

2. Between foil wrapped tightly around insulated enclosure and live metal parts: 3,500 volts plus twice the rated voltage of the tool.

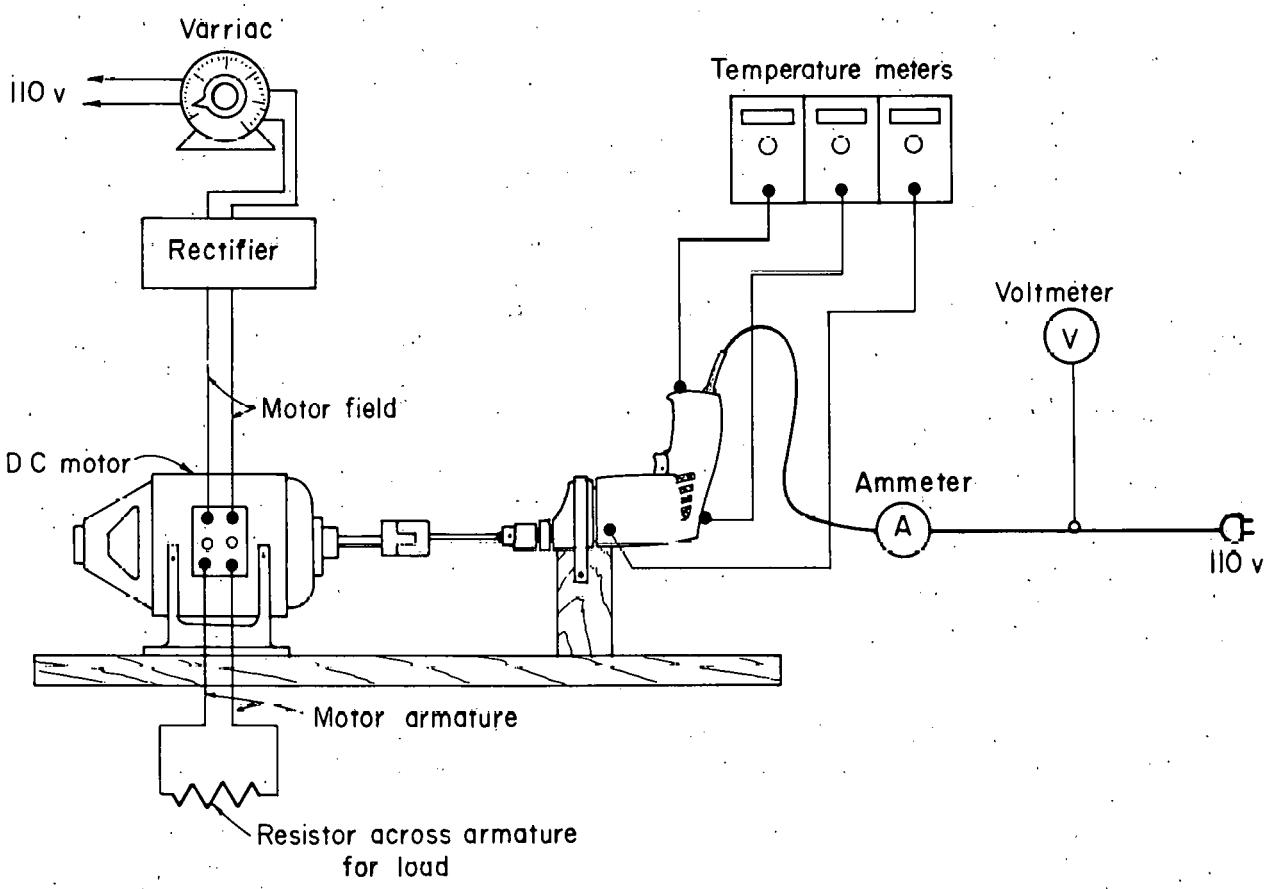


Fig. 1. Drill loading test

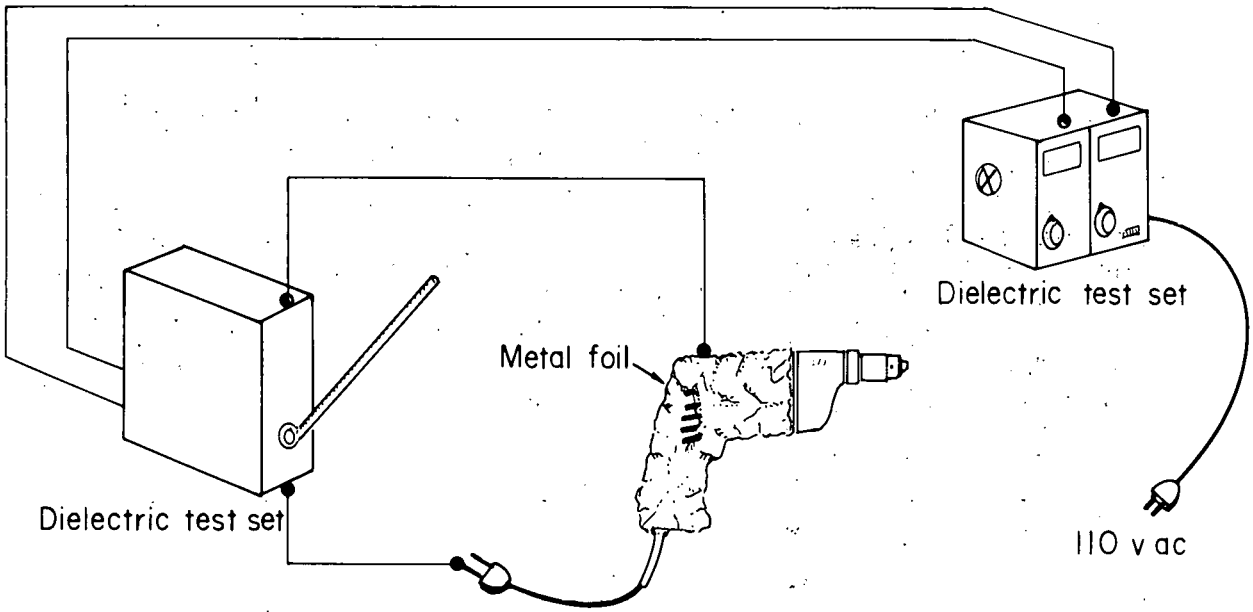


FIGURE 2. - Dielectric withstand test.

3. Between accessible dead metal parts and metal coil wrapped around the power supply cord, the inlet bushings cord guards, strain relief clamps, and the like: 3,500 volts plus twice the rated voltage of the tool.

LEAKAGE CURRENT TEST

Prior to measurement of the leakage current, the tool is to be operated at no load, but with normal airflow through the tool, for 100 or more hours, or until the brushes wear out if the latter condition occurs in less than 100 or more than 25 hours of operation. If the brushes wear out in less than 25 hours of operation, they are to be replaced and operation is to be continued until the tool has operated for a total of 25 hours.

Leakage current refers to all currents, including capacitively coupled currents, that may be conveyed between dead metal parts of a tool and ground or other dead metal parts of a tool. All dead metal parts are to be tested for leakage currents. The leakage currents from these parts are to be measured to the grounded supply conductor individually as well as collectively and from one part to another.

The leakage current is to be measured using a metal foil wrapped tightly around the nonmetal part of the tool. The metal foil is not to remain in place long enough to affect the temperature of the tool.

The measurement circuit for leakage current is to be as shown in figure 3. Unless the meter is being used to measure leakage from one part of a tool to another, the meter is to be connected between the dead metal parts and the ground supply conductor (fig. 4).

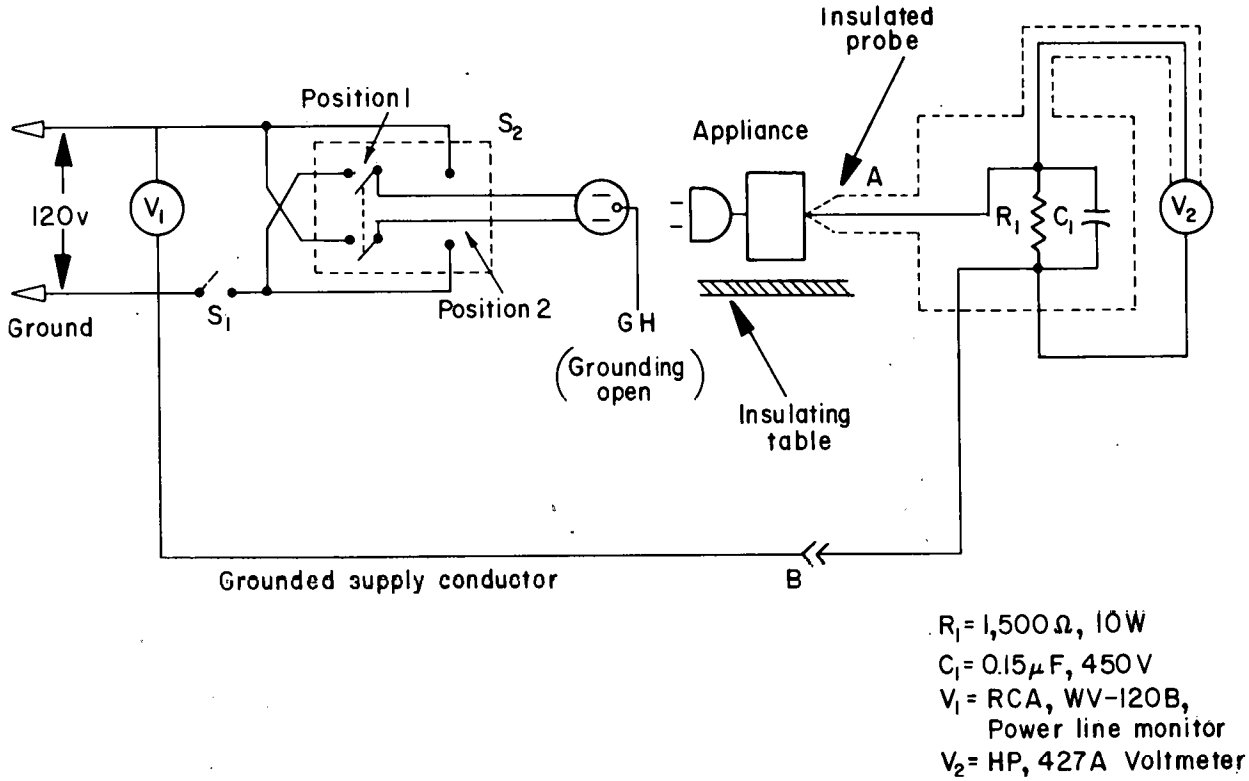


Fig. 3. Schematic of leakage current test

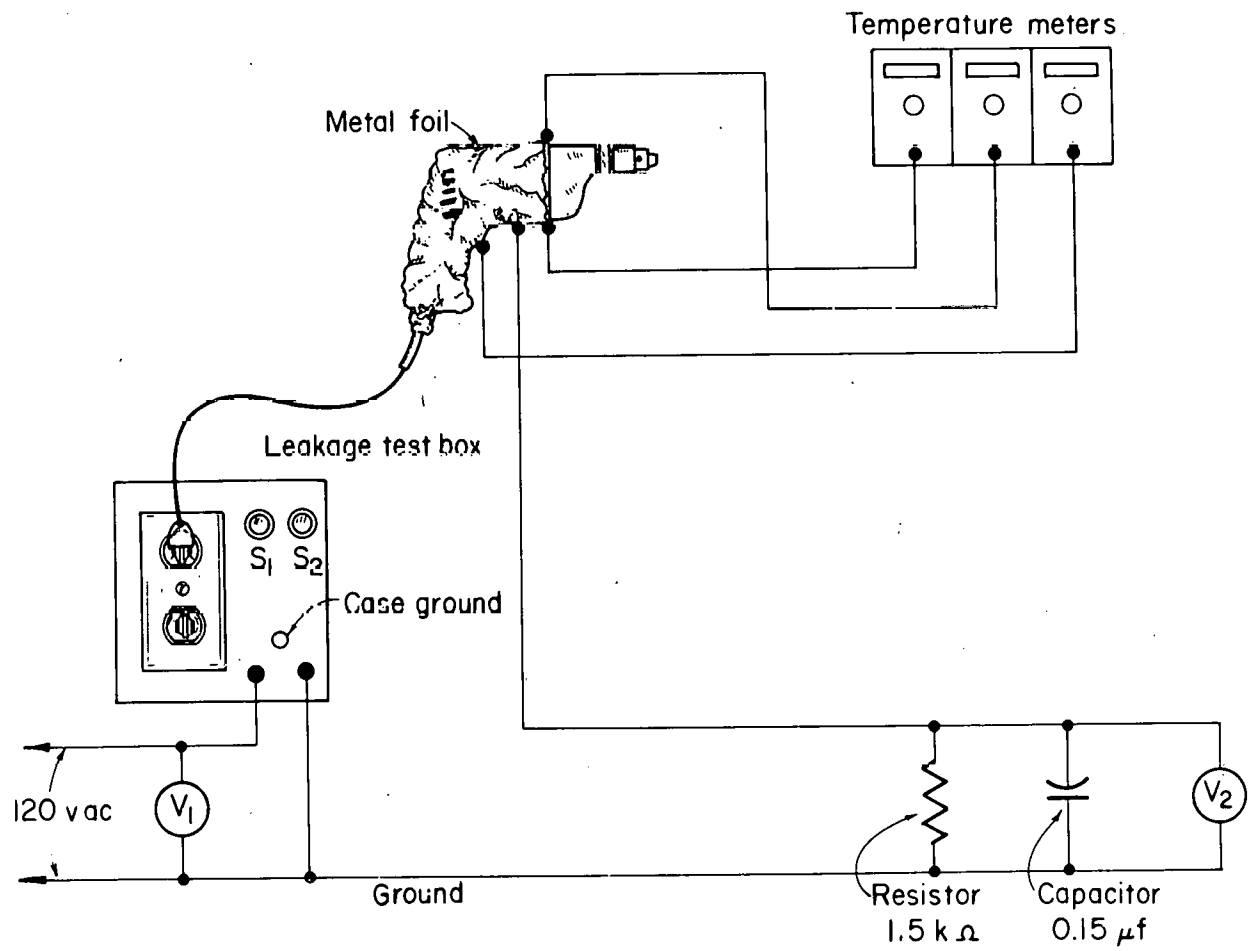


Fig. 4. Leakage current test

A sample of a tool is to be tested for leakage current after the conditioning described previously. The supply voltage is to be adjusted to 120 volts. The test sequence, with reference to the measurement circuit (fig. 3), is to be as follows:

1. With switch S1 open, the tool is to be connected to the measuring circuit. Leakage current is to be measured using both positions of switch S2, and with the tool switching devices in all of their normal operating positions.

2. Switch S1 is then closed energizing the tool, and with a period of 5 seconds, the leakage current is to be measured using both positions of switch S2, and with the tool switching devices in all their normal operating posi-positions.

3. The leakage current is to be monitored until thermal stabilization. Both positions of switch S2 are to be used in determining this measurement. Thermal stabilization is considered to be obtained, when the temperature of the drill becomes constant.

An individual measurement is to be made of the leakage current to each dead metal part that is insulated from other dead metal parts. Leakage current, as measured in this test, includes current resulting from any distributed capacitance as well as current through leakage resistance.

The leakage currents of a tool when tested shall not be more than 0.5 milliampere.

INSULATION RESISTANCE TEST

The insulation resistance test is to be conducted on the sample that was used for the current leakage test unless it is necessary to modify the sample or introduce extraneous conditions in measuring the currents to inaccessible parts. If a different sample must be used, it is to be conditioned in the same manner as for the current leakage test. In preparation for the test, the tool is to be kept in an enclosure for 48 hours or more at room temperature and at a relative humidity of 90 to 95 percent. The measurements of insulation resistance are to be made immediately after the tool has been removed from the conditioning chamber (fig. 5).

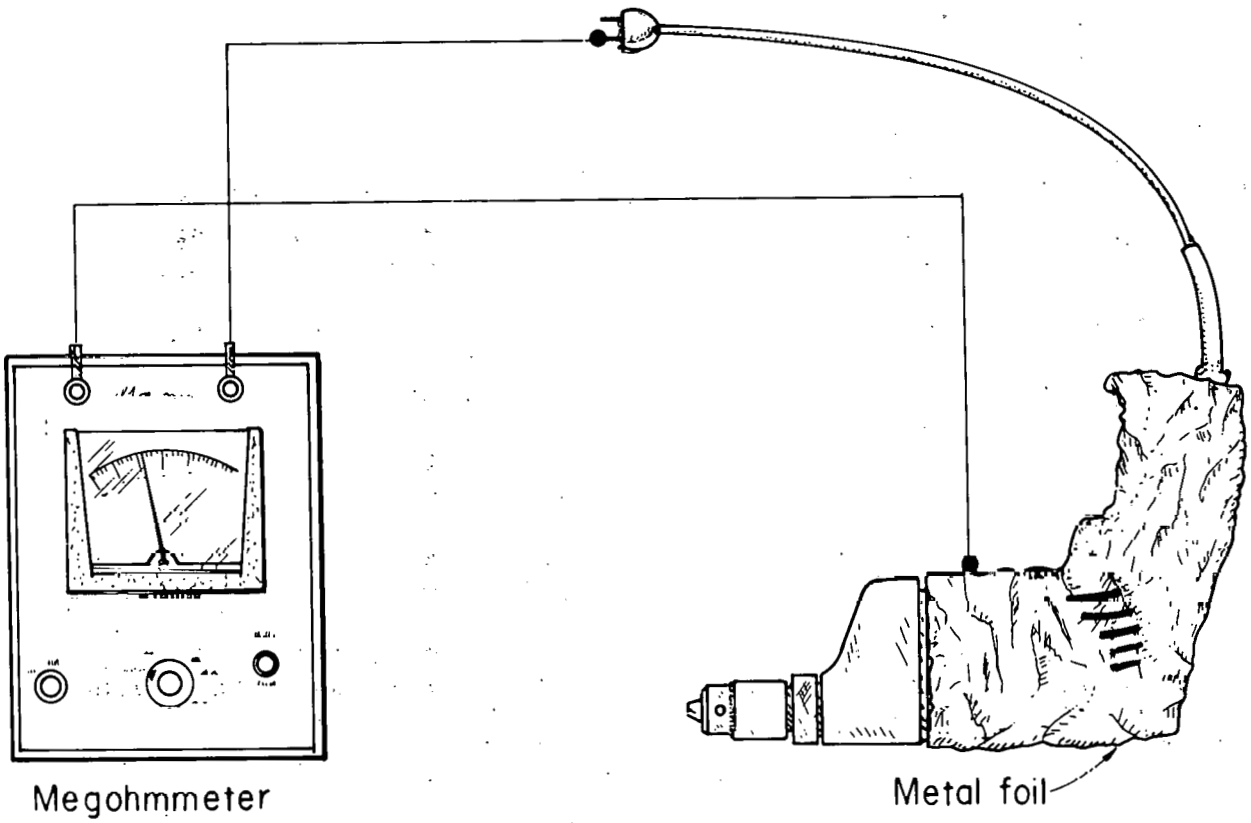


Fig. 5. Insulation resistance test

For a tool, the outer enclosure of which consists wholly or partly of insulating material, the term dead metal parts signifies metal foil wrapped around the exterior of the enclosure.

In the determinations of insulating resistance, a dc potential of 1,000 volts is to be employed, and the value of the insulation resistance is to be determined 10 seconds after the application of the test potential. An appropriate megohmmeter may be used for conducting the insulation resistance test, or other suitable means may be employed. The tool is not to be running during this test. Between live parts and dead metal parts the insulation resistance shall be at least 2 megohms.

OVERLOAD TEST

If the tool can be subjected to a running overload, each of three samples of the complete tool is to be subjected to operation at no load for 15 minutes, immediately followed by operation at full load (rated load for a general use tool) for 15 minutes. Directly following this, the load is to be increased in steps of 10 percent of the rated current for each of four successive 30-minute periods, followed by nine 10-minute periods, then followed by additional periods of 5 minutes each, as necessary, to produce breakdown of the functional insulation.

Breakdown of the functional insulation is considered to have occurred when (1) flame appears, (2) the winding burns open, or (3) the tool stalls under no load condition.

During the running overload operation, any protective device provided with the tool is to be short-circuited, and the branch circuit protection is to be of high enough capacity to withstand the test currents without opening the circuit. The objective of the test is to determine the ingenuity of the tool insulation and not the effectiveness of a protective device.

Operation of the tool under conditions of extreme overload shall not affect the insulation to the extent that live parts are exposed. The tool must also withstand for 1 minute the application between live parts and dead metal parts (or the foil), a dc potential of 1,000 volts plus twice the rated voltage without breakdown.

UNCONDITIONED DRILL TESTS

The following four tests were made on drills taken from factory sealed boxes. These drills had no prior conditioning. For identification purposes in this section, the drills will be referred to as drill A and drill B.

Test 1--Dielectric Withstand Test

The purpose of this test was to determine if the drill's insulation system could withstand a high voltage after the drill had reached full load operating temperature. In order to obtain the full load operating temperature, the drill was loaded to full load and its temperature monitored until it had stabilized.

Before the dielectric withstand test was run, drill A was brought up to operating temperature by being mounted in the drill test jig and loaded to full load current.

The temperature probes were mounted as follows:

T1--Chuck end of drill on metal case.

T2--Right side of drill as viewed from commutator end in the exhaust ports of the commutator.

T3--Left side of drill as viewed from the commutator and in the exhaust ports of the commutator.

The following is a list of the results of the dielectric withstand test:

	Volts
Between metal case of drill and prong 1 male power connector	3,750
Between metal case of drill A and prong 2 of male power connector	3,750
Between metal case of drill A and foil wrapped tightly around the insulated case	3,750
Between foil wrapped tightly around the insulated case and prong 1 of male power connector	3,750
Between foil wrapped tightly around the insulated case and prong 2 of male power connector	3,750

The above readings indicate that the insulation system on the new drills is very good. At the above voltages, the insulation system did not break down. The breakdown voltage was not determined since this would have destroyed the insulation of the drills.

Test 2--Leakage Current Test

The leakage current test was performed on drill A to determine the leakage current a man could be subjected to while using this type of drill. The drill was conditioned for the test by running it with no load for approximately 200 hours. After conditioning drill A, the insulated case was tightly wrapped with metal foil to provide a means for measuring leakage current from the nonmetal handle.

The temperature probes for the following test were located as follows:

T1--Chuck end of drill on metal case.

T2--Right side of drill as viewed from commutator end in the exhaust ports of the commutator.

T3--Left side of drill as viewed from the commutator end in the exhaust ports of the commutator.

A 0.15-microfarad capacitor and a 1,500-ohm resistor connected in parallel were used to simulate a human body. The voltage across the resistor capacitor combination should not exceed 0.6 volt to keep the current the human body would be subjected to under 0.5 milliampere.

The following is a list of the time, temperature, and voltage readings taken:

Time	Temperature, ° F			Voltage to metal case, polarity switch S_2		Voltage to foil, polarity switch S_2	
	T1	T2	T3	Position 1	Position 2	Position 1	Position 2
3:35 p.m....	82.0	82.6	82.7	0.006	0.0055	0.004	0.003
3:45 p.m....	83.2	77.9	78.3	.0059	.0054	.0037	.0025
3:55 p.m....	86.7	82.4	82.0	.0059	.0052	.0037	.0025
4:05 p.m....	89.0	84.3	83.9	.0062	.0056	.0032	.0026

It can be calculated from the above data that the maximum current a man would be subjected to while operating a typical drill is 0.0052 milliampere. This amount of current cannot be felt by a man and is a safe exposure level.

Test 3--Insulation Resistance Test

The purpose of this test is to determine the effects of storing a double insulated tool in a humid environment. The value of the insulation resistance over a time period of 10 seconds is measured for this determination. As insulation resistance is adversely affected by heat and humidity, the drill was placed in a humidity chamber at 95 percent humidity and a temperature of 70 F. This allowed the drill's insulation system to thoroughly absorb moisture. By conditioning the drill in this manner, insulation resistance values are obtained which represent a drill which has been stored in a humid environment. Drill A was used in this test since it had been run for an extended period of time which allowed heat, the other factor that deteriorates insulation, to have its effect.

A 1,000-volt megohmeter was used to measure the resistance of the drill's insulation system.

The following is a list of the results of the insulation resistance test.

Insulation resistance:

	Results
Between metal foil wrapped tightly around insulated case and metal case.	Infinity
Between metal foil wrapped tightly around insulated case and prong 1 of power cord.	Infinity
Between metal foil wrapped tightly around insulated case and prong 2 of power cord.	Infinity

As can be seen from the above test results, the drill's insulation system integrity was not adversely affected by heat and hunidity.

Test 4--Overload Test

After extended usage and constant overloading of a drill, the motor will eventually fail. The failure of the motor, especially under overload conditions, usually results in smoking heating, and burning of the drill. The purpose of this test is to determine the dielectric strength of the drill's insulation system after an overload failure of the drill has occurred. In this test, a drill was constantly overloaded until it burst into flame and was no longer able to operate electrically.

A new factory boxed drill, drill B, was placed in the drill loading test jig as shown in figure G-1. The drill was loaded in the following manner at the times indicated:

Time	Drill loading
8:05 a.m.	Started running, no load
8:20 a.m.	Full load, 3 amps.
8:35 a.m.	Loaded drill to 3.25 amps.
9:05 a.m.	Loaded drill to 3.6 amps.
9:35 a.m.	Loaded drill to 3.9 amps.
9:40 a.m.	Drill B burned out and burst into flame.

After drill B had time to cool, a dielectric withstand test was performed. The following is a list of the results of the dielectric withstand test:

	Volts
From the metal case to each prong of the male power connector	1,000
From metal foil wrapped tightly around the insulated case and each prong of the male connector	1,000
From metal foil wrapped tightly around the cord strain relief and each prong of the male power connector	1,000
From metal foil wrapped tightly around the insulated case and the metal case	1,000

Drill B was disassembled. The field coil and armature were then given a dielectric withstand test. The test results are given below:

	Volts
Dielectric strength from each coil lead and the frame of the coil	1,000
Dielectric strength between the commutator and the shaft	3,500

The above results indicate that the drill's insulation system has good dielectric propoerties even though the drill was overloaded until it burst into flames.

COAL-CONDITIONED DRILL TESTS

The following four tests were conducted on drills that were subjected to conditions as might be found in the coal mines. A mixture of 650 milliliters-200 mesh coal dust and 2-1/2 gallons of water was prepared. Two new factory boxed drills were placed in the solution and allowed to soak for 3 hours. The drills were then allowed to air-dry at room temperature for 48 hours. These drills are referred to in the following four tests as coal-conditioned drills C and D.

Test 1--Dielectric Withstand Test

The purpose of this test was to determine the dielectric properties of drill C, which had been subjected to an adverse environment. The coal-conditioned drill was loaded to full load and its temperature nomitored until it had stabilized.

The drill was mounted in the drill test jig and loaded to full load at rated current.

The temperature probes were placed on the drill as described in the previous dielectric withstand test.

The following is a list of the results of the dielectric withstand test:

	Volts
Between metal case of drill and prong 1 of male power connector	3,750
Between metal case of drill and prong 2 of male power connector	3,750
Between metal case of drill and foil wrapped tightly around insulated case	3,750
Between foil wrapped tightly around the insulated case and prong 1 of male power connector	3,750
Between foil wrapped tightly around the insulated case and prong 2 of male power connector	3,750

Test 2--Leakage Current Test

The purpose of this test was to determine the current a man could be subjected to when using a drill which had been conditioned with coal dust and water. Insulation systems are adversely affected by heat, dirt, and humidity. Drill C, which was previously conditioned with water and coal dust, was then run at no load for approximately 500 hours. By conditioning the drill in this manner, it gave these three factors time to deteriorate the insulation systems. The insulated case of the drill was then tightly wrapped with metal foil to provide a means for measuring leakage current to the insulated case.

A 0.15-microfarad capacitor and a 1,500-ohm resistor connected in parallel were used to simulate a human body. The voltage across this resistor capacitor combination should not exceed 0.6 volt to keep the current the human body would be subjected to under 0.5 milliampere.

Listed below are the time, temperature, and voltage readings taken:

Time	Temperature, ° F			Voltage to metal case, polarity switch S_2		Voltage to foil, polarity switch S_2	
	T1	T2	T3	Position 1	Position 2	Position 1	Position 2
1:00 p.m...	106.9	78.9	79.2	0.004	0.0044	0.0026	0.0038
1:10 p.m...	107.0	78.7	83.1	.0042	.0044	.0027	.0039
1:20 p.m...	106.3	77.9	78.9	.004	.0043	.0027	.004
1:30 p.m...	105.8	78.0	77.8	.0038	.0044	.0027	.004
1:40 p.m...	106.0	77.7	77.1	.0038	.0042	.0027	.004
1:50 p.m...	105.4	76.9	76.8	.0042	.0044	.0028	.004
2:00 p.m...	104.8	76.4	76.4	.0042	.0045	.0027	.004

It can be calculated from the above data that the maximum current a man would be subjected to while operating a typical drill is 0.00375 milliampere. This amount of current cannot be felt by a man and is considered a safe exposure level.

Test 3--Insulation Resistance Test

The purpose of this test is to determine the insulation resistance over a period of 10 seconds. Since the drill was conditioned with coal dust and water, and ran for approximately 500 hours, the value of insulation resistance determined by this test should represent a drill which had been used for a long period of time. Prior to the insulation resistance test, the drill was placed in a humidity chamber at 95 percent humidity and a temperature of 70°C for six days. This allowed the insulation system of the drill to absorb moisture.

A 1,000-volt megohmmeter was used to measure the resistance of the drill's insulation system.

The following is a list of the results of the insulation resistance test:

	Results
Insulation resistance:	
Between metal foil wrapped tightly around insulated case and metal case.	Infinity
Between metal foil wrapped tightly around insulated case and prong 1 of power cord.	Infinity
Between metal foil wrapped tightly around insulated case and prong 2 of power cord.	Infinity

The temperature probes for the leakage current test were located as follows:

T1--Chuck end of drill on metal case.

T2--Right side of drill as viewed from the commutator end in the commutator exhaust ports.

T3--Left side of drill as viewed from the commutator end in the commutator exhaust ports.

As can be seen from the above test results, the drill's insulation system was not adversely affected by coal dust, heat, and humidity.

Test 4--Overload Test

After extended usage and constant overloading of the drill, the motor will eventually fail. The failure of the motor, especially under overload conditions, usually results in smoking, heating, and burning of the drill. Since the drill used in this test had been conditioned with coal dust and water, the overloading and burnout of the drill in the test should also result in some tracking on the insulation system of the drill. The purpose of this test was to determine the dielectric strength of the drill's insulation system after an overload failure of the drill had occurred. The drill was constantly overloaded until it burst into flame and was no longer able to operate electrically.

Time	Drill Loading
9:24 a.m.	Started running, no load.
9:50 a.m.	Full load, 3 amps.
10:05 a.m.	Loaded drill to 3.3 amps.
10:35 a.m.	Loaded drill to 3.6 amps.
11:05 a.m.	Loaded drill to 3.9 amps.
11:13 a.m.	Drill D burned out and burst into flame.

After the drill had cooled, a dielectric withstand test was performed on the drill. Listed below are the results of the dielectric withstand test:

	Volts
From the metal case to each prong of the male power connector	1,500
From metal foil wrapped tightly around the insulated case and each prong of the male power connector	1,500
From metal foil wrapped tightly around the insulated case and the metal case	1,500
From metal foil wrapped tightly around the cord strain relief and each prong of the male power connector	1,500

Drill D was disassembled. The field coil and armature were then given a dielectric withstand test. The test results are given below:

Dielectric strength from coil lead one and the frame of the coil broke down at 600 volts.

Dielectric strength from coil lead two and the frame of the coil broke down at 800 volts; and

Dielectric strength between the commutator and the shaft was taken to 3,000 volts, and showed no breakdown.

The above results indicate that the drill's insulation system has good dielectric properties even though the drill was overloaded and burst into flame. The insulation of the coil leads and frame of the coil did break down at 600 volts and 800 volts. For the same test on the unconditioned drill, the leads did not break down and the voltage was taken to 1,000 volts. The breakdown of the insulation at 600 and 800 volts is still acceptable since the maximum voltage the drill's insulation system will have on it will be approximately 200 volts.

CONDITIONED WET DRILL TESTS

The two leakage current tests explained in this section were made on two separate drills that were conditioned with coal dust and water and tested wet, simulating very severe conditions that are present in the mines. The two drills are referred to as (1) drill E, a new drill that was taken from a factory sealed box, and (2) drill C, a coal-conditioned drill previously used in tests 1, 2, and 3. After soaking in the coal dust/water mixture for a period of time, the two wet drills were taken out of the solution and each given the leakage current test.

Leakage Current Test--New, Wet Drill

The purpose of this test was to determine the leakage current a man would be subjected to when using a wet, coal-conditioned drill. Testing a drill in this manner simulates the effects of a man dropping a dirty drill into water, then drying the outside of it off and using it again. Drill E was soaked for 7 hours in a solution of 650 milliliters of -200 mesh coal dust and 2-1/2 gallons of water.

After allowing the water to run out of the drill, the outside of the drill was wiped with a rag. The insulated case of drill E was the tightly wrapped with metal foil.

The temperature probes for this test were located as follows:

T1--Chuck end of drill between metal case and insulated case.

T2--Right side of drill as viewed from the commutator end in the exhaust ports of the commutator.

Readings were taken across a 0.15-microfarad capacitor in parallel with a 1,500-ohm resistor. The voltage across the resistor capacitor combination should not exceed 0.6 volt to keep the current the human body would be subjected to under 0.5 milliampere. A current above 0.5 milliampere is considered to be unsafe.

Listed below are the time, temperature, and voltage readings taken:

Time	Temperature, ° F		Voltage to metal case, polarity switch S_2		Voltage to foil, polarity switch S_2	
	T1	T2	Position 1	Position 2	Position 1	Position 2
3:20 p.m.....	75.4	88.4	0.22	0.18	0.032	0.063
3:30 p.m.....	76.1	89.0	.205	.185	.033	.060
3:40 p.m.....	76.0	89.1	.18	.17	.055	.048
3:50 p.m.....	76.1	89.8	.51	.05	.033	.04
4:00 p.m.....	75.6	87.9	.15	.13	.028	.038

It can be calculated from the above data that the current a man would be subjected to when holding the metal case of the drill after it was pulled out of the water and coal dust solution is approximately 0.43 milliampere. This is considered to be a safe amount of current and would not hurt the drill operator.

Leakage Current Test--Coal-Conditioned Wet Drill

The purpose of this test is to determine the leakage current available from a drill which had been used for a long period of time and subjected to coal dust and water. Testing the wet drill simulates the condition where a man drops a drill into water then dries the outside and uses it.

Drill C was soaked for 6-1/2 hours in a solution of 650 milliliters of -200 mesh coal dust and 2-1/2 gallons of water. The water was allowed to run out of the drill and the outside was wiped with a rag. The insulated case of the drill was tightly wrapped with metal foil.

Temperature probes were placed on drill C at the following locations:

T1--Chuck end of drill between the metal case and the insulated case on the left side of the drill as viewed from the commutator end.

T2--Chuck end of drill between the metal case and the insulated case on the right side of the drill as viewed from the commutator end.

Voltage readings were taken across a 0.15-microfarad capacitor in parallel with a 1,500-ohm resistor. The voltage across this resistor capacitor combination should not exceed 0.6 volt to keep the current the human body would be subjected to under 0.5 milliampere. A current above 0.5 milliampere is considered ot be unsafe.

Listed below are the time, temperature, and voltage readings taken:

Time	Temperature, ° F		Voltage to metal case, polarity switch S_2		Voltage to foil, polarity switch S_2	
	T1	T2	Position 1	Position 2	Position 1	Position 2
3:30 p.m.....	88.2	82.1	1.5	0.45	8.8	2.2
3:40 p.m.....	112.1	93.8	.68	.78	.0053	.0045
3:50 p.m.....	109.0	97.9	.06	.055	.0047	.0035
4:00 p.m.....	105.2	98.5	.055	.05	.0045	.0034
4:10 p.m.....	104.5	98.9	.04	.045	.0045	.0033
4:20 p.m.....	107.5	96.8	.04	.04	.0045	.0033

It can be calculated from the above data that the current a man would be subjected to when holding the insulated case after the drill is first pulled out of the water is approximately 75 milliamperes. This is in the range of currents that could be fatal to the drill operator.

CONCLUSIONS

As can be seen from the tests which were conducted on the drills, the double insulation system held up under very adverse testing. Of the leakage current tests made on the different drills, the coal-conditioned wet drill test of drill C on page was the only failure. This failure was expected as the drill was dripping wet when it was energized. The hazardous leakage current levels of this test decreased to a safe level after 10 minutes of running time.

The tests proved double insulation protection to be very effective in providing electrical shock protection for hand-held tools. The protection performed very well even under adverse wet conditions. However, as with the standard frame grounded handtool, the double insulated handtool should not be used while wet or in situations where a man is in contact with wet surfaces.

We recommend the double insulated system be accepted under Sections 75.702 and 77.702 and in nonmetallic mines under Sections 55, 56, and 57.12-25, when it has been listed by Underwriters Laboratories. On those few occasions when double insulated or standard handtools must be used in wet areas, it is highly recommended that the tool be kept dry and a ground fault interrupter be used with it.

REMOTE GROUNDING OF LARGE SURFACE ELECTRICAL EQUIPMENT

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INTRODUCTION

The most important function of a grounding system is to protect the mine personnel from electric shock. The grounding system should also limit the energy available at a fault location, provide for the control of over-voltages, and aid in the isolation of defective areas by selective relaying. The present system may be described as a "resistance-grounded-neutral system", wherein a low-resistance ground bed is connected through a current-limiting resistor to the neutral point of the transformer secondary winding. One or more metallic ground conductors extend from the ground bed to the frames of all portable and mobile equipment. It may be possible to improve certain aspects of ground system performance by eliminating the safety ground bed and relying instead upon the machine/soil contact surfaces to provide a connection to the earth.

STEP AND TOUCH POTENTIALS

Mine personnel are exposed to a shock hazard whenever the potential of a machine frame is raised above that of the surrounding earth. A field trip was made to a large surface-mining complex in the Midwest to measure voltage gradients around various types of mining equipment. Figure 1 shows a potential survey which was mapped around a Bucyrus-Erie 280-B loading shovel. The machine was disconnected from the substation and a small test current was then circulated between the machine frame and a driven rod located 200 feet away. Fall-of-potential measurements were made from several points on the shovel, with the line of electrodes oriented in several directions. Points of equal potential were then plotted and connected together to form voltage contour lines. The voltage on the shovel frame (and the bucket) was normalized to 1.0, and the numbers on the contour lines represent the potential between the machine frame and that line. Because the shovel's counterweight overhangs the treads by several feet, a person standing on the ground while touching the shovel could be exposed to 80% of the frame voltage. The potential gradients are particularly steep near the corners of the treads and around the bucket. It is possible that a person who touched the machine while standing on the ground would be badly shocked.

A potential map around a vertical-mast drill rig is shown in Figure 2. Several of the equi-potential contour lines seem to pass beneath the crawler tread, which indicates that portions of the tread were not

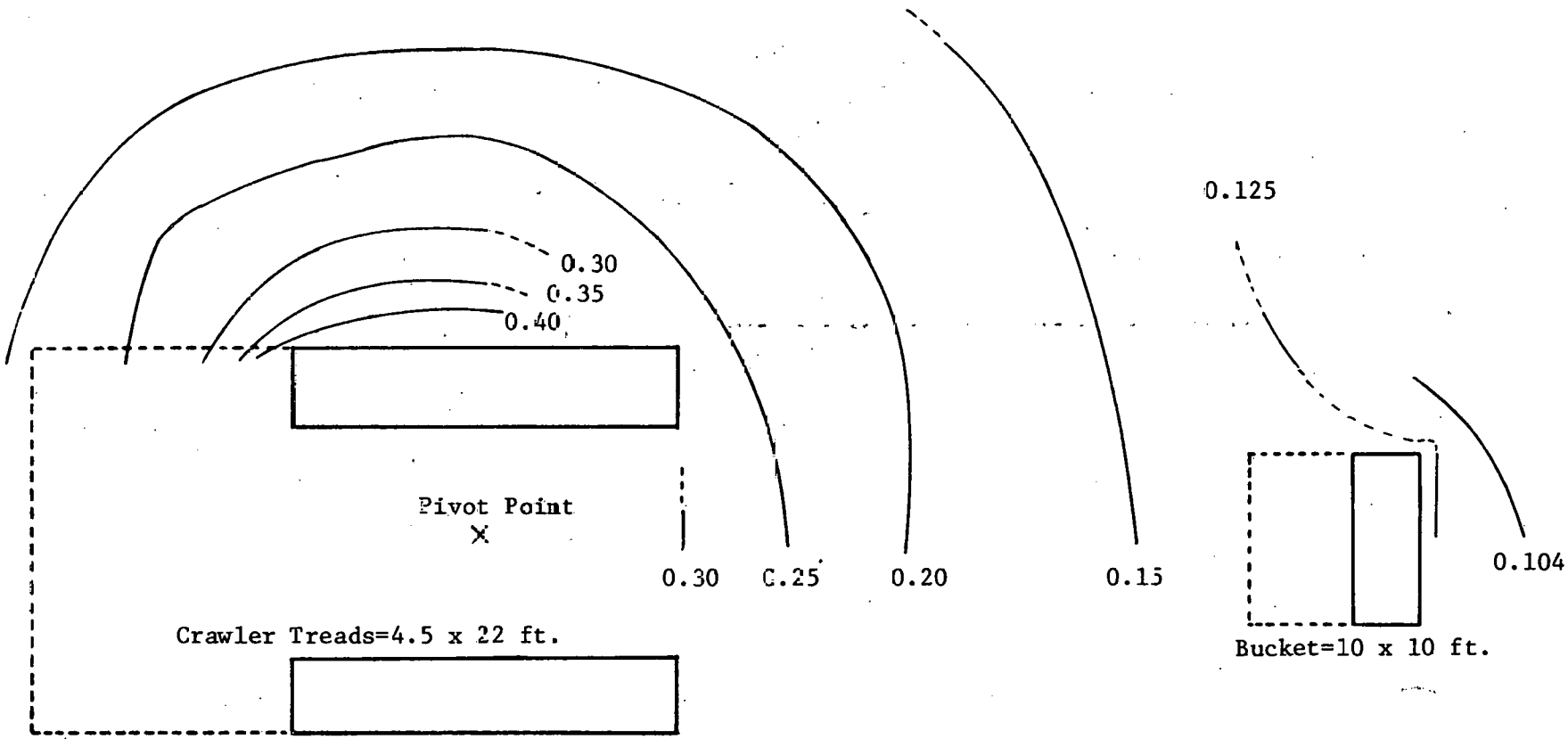


Figure 1. Potential gradient survey around a loading shovel, with voltages given in per unit of frame potential.

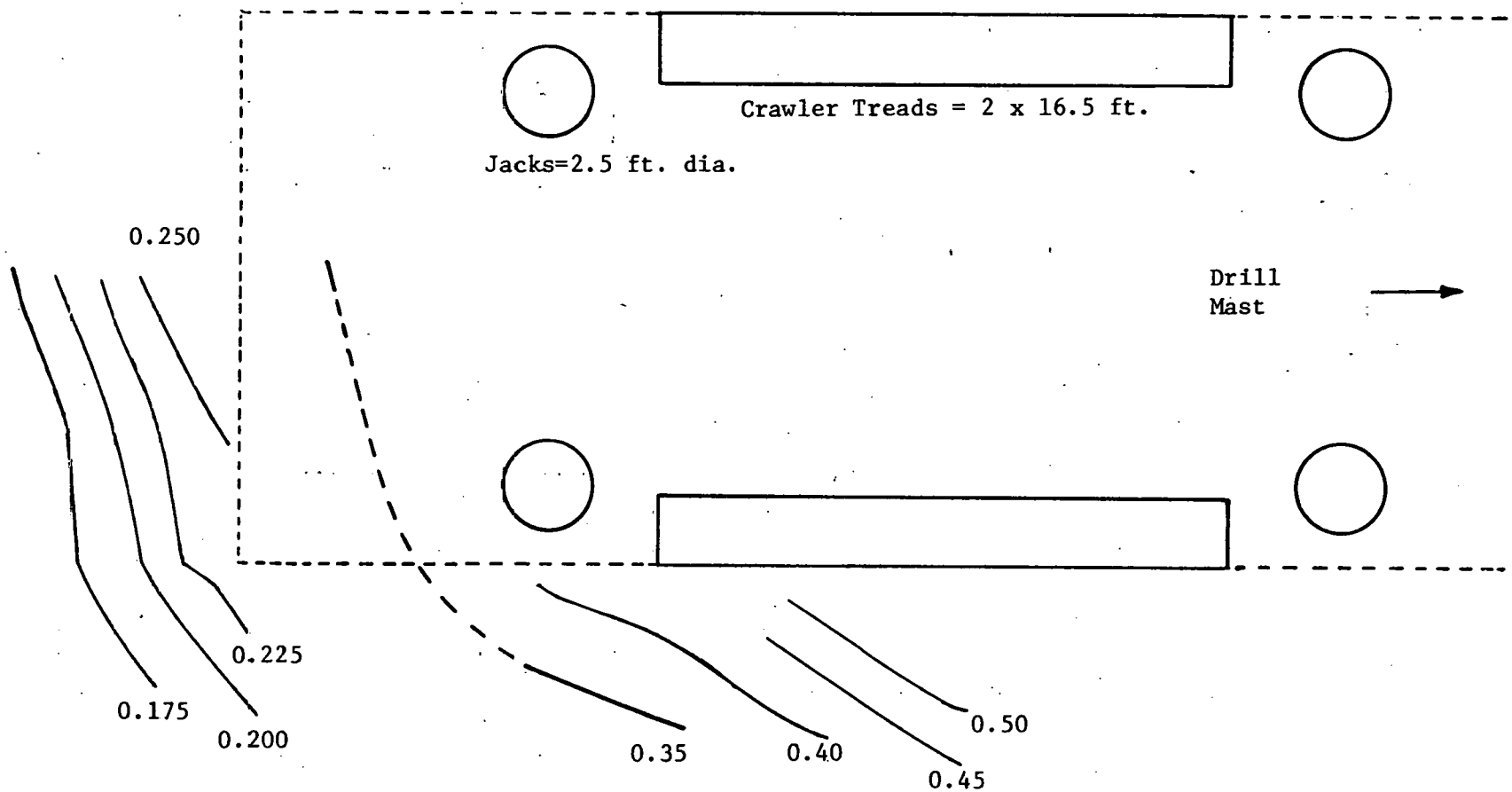


Figure 2. Potential gradient survey around a blast-hole drill, with voltages in per unit of frame potential.

actually in contact with the mine floor. It is probable that the four circular jacks are supporting a large portion of the drill's weight. As in the previous example, the possibility exists that an unlucky individual could be exposed to severe step-or touch-potentials if the machine frame becomes "hot" with respect to ground. In some cases, malfunctions can lead to elevated frame potentials, but several steps can be taken to minimize the likelihood of electrocution. Persons in contact with the ground should not sit, touch, or lean against any piece of equipment, even though it may appear to be well grounded. To facilitate getting on and off the machine, hand rails and steps should be constructed of non-conducting material, or should be well-insulated from the machine frame if made of metal.

PERFORMANCE COMPARISON OF REMOTE GROUNDING AND FRAME GROUNDING

The behavior of a standard remote grounding system will be compared with that of the proposed frame grounding system when the two are subjected to a variety of situations which can occur in mining environments.

TRANSFER OF POTENTIAL

The transfer of potential from the substation ground bed to the safety ground bed can endanger mine personnel in the same manner as was discussed earlier, ie. by elevating equipment frames to some lethal voltage above ground. The substation ground bed can normally be expected to handle primary faults as well as lightning and other surges coming into the substation via the overhead lines. Common practice dictates that the safety ground bed be physically separated from the substation ground bed by a distance of 25 or perhaps 50 feet. The following example will show that this may not be the best solution.

For the sake of simplicity it will be assumed that the substation ground bed consists of a single driven rod, 20 feet long and 2 inches in diameter. When immersed in a soil whose resistivity is 100Ω-ft, this rod will show a steady-state resistance of just under 5Ω. If the substation ground bed carries a fault current of 2000A, then the resulting potential on the bed is 10kV with respect to "true ground." Figures 3 and 4 show the voltage profile on the surface of the earth surrounding the electrode. The shape of the potential profile depends on the length-to-radius ratio of the rod electrode. Neutral-resistor grounding systems are designed so that, under fault conditions, the allowable potential in the circuit external to the resistor is 100V or less. If it is also decided to limit the induced potential on the safety ground bed to 100V, which is only 1% of the 10kV on the substation ground rod, then Figure 4 shows that the required bed separation must be about 300 feet.

If the substation ground bed was subjected to a lightning surge, then a nearby safety ground bed would be raised to a very high potential for a short length of time. However, it is possible that very little of this transferred potential would propagate through the trailing cable to the frames of equipment connected to the safety ground bed.[2]

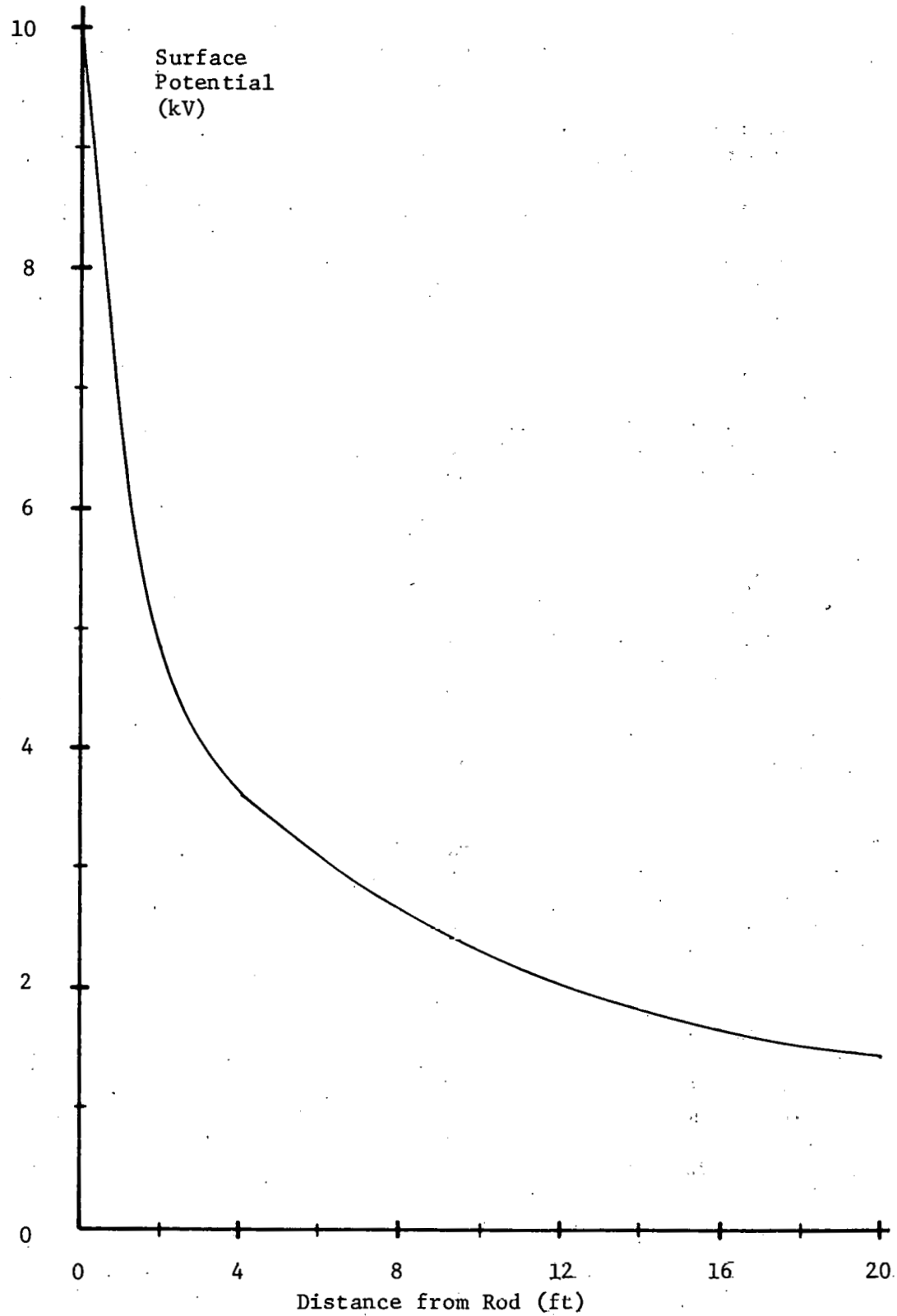


Figure 3. Surface Potential Profile Near the Substation Ground Rod.

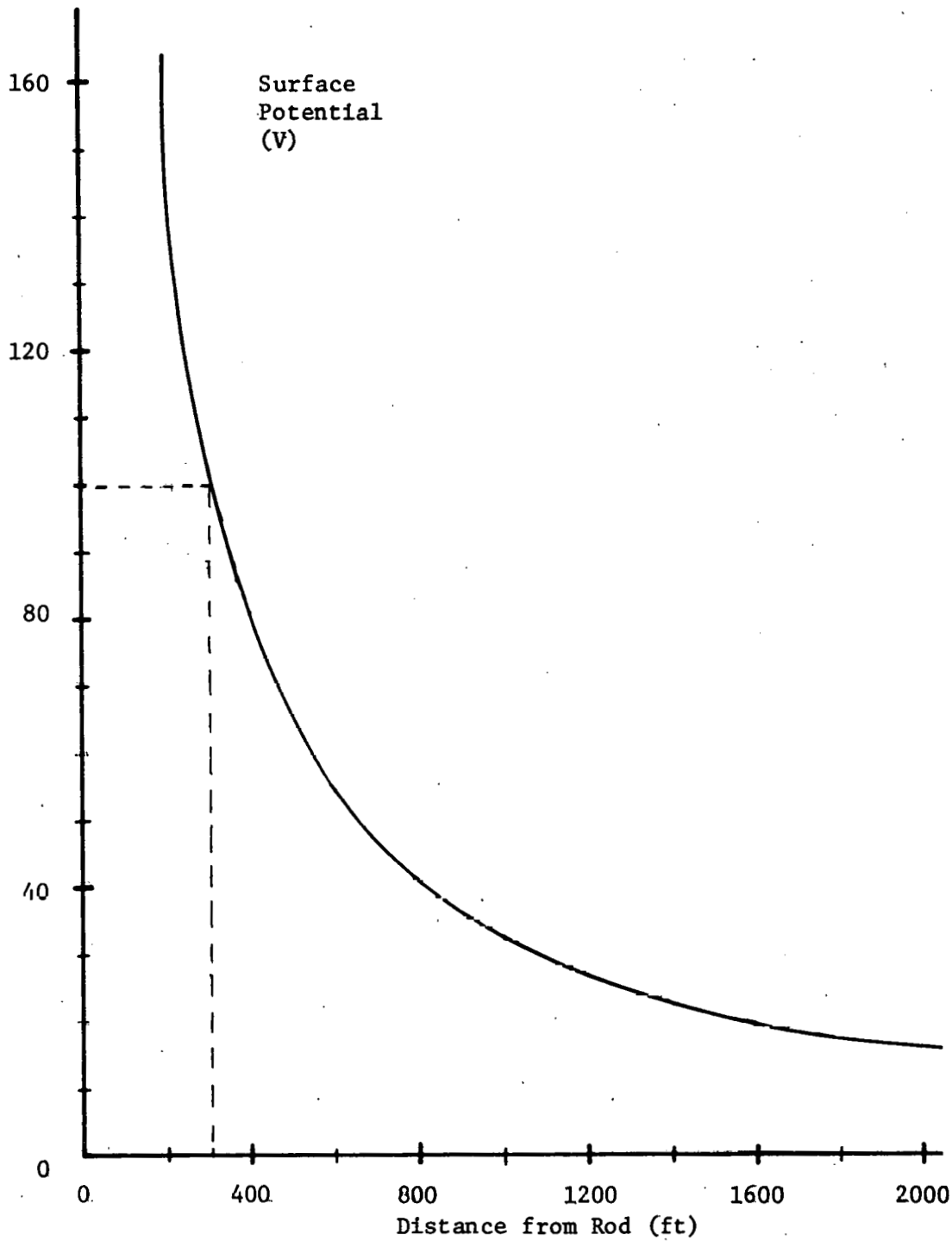


Figure 4. Expanded Potential Profile Around Substation Ground Rod.

Thus it appears that the remote grounding arrangement presently in use is inferior to frame grounding in the case of transfer of potential during a primary fault at the substation. The separation between a substation and the mining equipment to which it supplies power is usually on the order of several thousand feet, which should greatly reduce the magnitude of the transferred potential which is now possible with traditional 25 to 50 foot ground bed separation distances. The two grounding configurations appear to be evenly matched in performance in the event of a lightning stroke to the substation ground bed.

PHASE-TO-EARTH FAULT ON SECONDARY DISTRIBUTION

Many surface coal mines use shielded trailing cable to span the entire distance from substation to machine, and phase-to-earth faults would therefore be unlikely to occur. However, open-pit metal mines often use overhead lines to transmit power around the mine perimeter and down the benches. At strategic locations the overhead lines then terminate at switch-houses, from which the power is carried to machinery via trailing cables. These overhead lines may run for a mile or more, and phase-to-earth faults are possible.

If a frame-ground system was used, the fault current would flow through the machine frame on its way back to the transformer netural, thereby exposing mine personnel to a serious shock hazard. A remote grounding system provides a low-resistance earth connection through which the fault current can flow, but alternate paths through the machine frames still exist, and these parallel paths may carry a significant fraction of the fault current if their resistance is low. It appears that, again, the two grounding systems may be roughly equal in their ability to protect miners from electric shock. Protective circuitry should be used to detect the faults and deactivate the power system.

PHASE-TO-GROUND FAULT WITH OPEN GROUND CONDUCTOR

If the ground conductor(s) leading from the machine frame back to the safety ground bed at the substation should open, then the equipment could become elevated to a dangerous potential in the event of a phase-to-ground fault. Fault current would flow through the machine frame into the earth, and return to the transformer netural via the safety ground bed. This situation is unlikely to occur in mines which use trailing cable for the entire span between substation and machine, but is quite possible if overhead distribution lines are used to connect substations to switch-houses or junction boxes. With an open ground conductor, the neutral grounding resistor is placed in series with the safety ground bed and the contact resistance of the machine. The frame potential is determined by the resulting voltage-divider network, and is proportional to the ground resistance of the machine. Federal records show that in 1977 a man was electrocuted in a limestone quarry when he contacted a 2300V shovel. There was a line-to-ground fault on the machine, and the safety ground system was disrupted at the time.

Should a phase-to-ground fault occur on a frame-grounded system while the ground conductors were open, then a very small fault current might flow back to the transformer neutral via capacitive coupling. Fault current could also travel through the earth to the frame of another machine which was fed from the same substation and return to neutral. The machine frame would be elevated to a high potential, presenting a serious hazard to personnel.

Both the remote grounding system and the grame-grounding system are ineffective if the ground circuit is not continuous. Mines using overhead lines for a portion of their secondary distribution are especially susceptible to breaks in the ground wires, and reliable high-voltage ground-check monitors would be helpful.

LIGHTNING STRIKING A MACHINE

One does not normally think of a dragline or shovel as an earth electrode, but it is likely that, in the event of a lightning stroke to the boom of a dragline, the machine would temporarily act as a fairly effective ground rod. A study of the literature indicates that the impedance of an earth electrode under surge conditions will be equal to or less than its steady-state resistance.[3,5] The major cause of this phenomenon is the breakdown of the soil surrounding the electrode under the stress of intense electric fields.[4] Electrode resistance may also be reduced due to the formation of streamers radiating horizontally along the surface of the ground.[3]

If lightning should strike a large piece of surface equipment, there are several paths which the surge current can follow to earth. The shortest path would be directly through the machine frame/soil interface, while the other path would be back along the ground conductors to the safety ground bed. Table 1 shows the results of resistance measurements made on large units of electrically-powered surface mining equipment. This data was taken during the past four years and includes machinery operating in both surface coal mines and open-pit metal mines.[1] The resistance values range from 1Ω to nearly 90Ω, although all the resistance values found in surface coal mines are less than 15Ω. From the previous analysis one could assume that, under surge conditions, the impulse impedance of any unit of equipment would be less than the resistance figure given in the table. Since the approximate surge impedance of a trailing cable is 50Ω and overhead lines are about 400Ω, it is likely that current from a lightning stroke would flow to ground via the machine/earth contact surface.

If lightning should strike a large piece of surface-mining equipment, the outcome will probably be the same whether a remote ground system or a frame ground system is used. For the short duration of the stroke, hazardous step-and touch-potentials will exist on and around the machine. An examination of all the available MSHA reports on surface mine electrical fatalities over the last 7 years failed to produce any direct evidence of electrocution caused by the mechanism just reviewed. However, several accidents were described where the cause of death was undetermined, and lightning was suggested as one of

TABLE 1

MEASURED RESISTANCE VALUES FOR VARIOUS

TYPES OF MINING EQUIPMENT

Mine Type	Machine Type	Weight (lbs)	Contact Area (ft^2)	Resistance (Ω)
Open-Pit Metal	Drill 1	75,000	48	86
	Shovel 1	380,000	90	18.2
	Shovel 2	380,000	90	60.4
	Shovel 3	405,000	90	39
	Shovel 4	500,000	107	4.25
	Shovel 5	950,000	140	21
	Shovel 6	379,000	147	4.5
	Shovel 7	379,000	147	10.2
	Shovel 8	379,000	147	8.1
	Shovel 9	379,000	147	5.3
	Shovel 10	580,000	155	13.5
	Shovel 11	580,000	155	7.5
	Shovel 12	975,000	158	8
	Shovel 13	972,000	163	29
	Shovel 14	915,000	166	5
	Shovel 15	915,000	166	34.8
	Dragline 1	1,600,000	1018	0.9
Surface Coal	Drill 2	-	152	4.05
	Drill 3	-	152	4.8
	Drill 4	-	86	13.23
	Shovel 16	-	96	5.75
	Shovel 17	-	152	7.9
	Shovel 18	-	115	4.78
	Shovel 19	960,000	198	4.72
	Dragline 2	-	1520	1.4
	Dragline 3	1,271,000	804	1.05

a number of possible explanations.

CONCLUSION

It has been proposed that the frames of large mining equipment may be used to provide an earth connection in the ground circuit on high-voltage power systems, thereby eliminating the need for a safety ground bed. Comparisons with the standard safety-grounded system seem to show that an equal or improved performance may be achieved with frame-grounds without the expense of installing a ground bed. Adequate protective circuitry and ground-check monitors should be applied to such systems to safeguard mine personnel and to detect fault conditions.

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EXAMINATION OF MEDIUM AND LOW VOLTAGE TRAILING CABLE SPLICES

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ABSTRACT

Medium and low voltage, shielded and unshielded trailing cable splices were examined to determine possible defects due to workmanship and materials. This paper discusses the defects discovered, their possible causes, and recommendations for improving splices made on medium and low voltage trailing cable in the repair shop and in the mine. This work was performed under a Bureau of Mines contract titled "High and Medium Voltage Trailing Cable Splices and Connectors," Contract #H0166069.

INTRODUCTION

In the examination our prime concern was the system of reinsulating, reshielding, and rejacketing the trailing cable power and ground conductors which have been rejoined with some type of compression or welded connection. In making our recommendations the splicing process was divided into splice inspection, splice materials, and personnel training. Our approach throughout, in studying splicing problems, was broad and qualitative. It was specific and quantitative wherever possible.

During the examination a survey was performed of the users and suppliers of splices to determine the factors which affect low and medium voltage trailing cable splices. Low voltage was included since very little medium voltage is in service. The survey was done during the examination to gather opinions regarding the present performance and needed improvements of splice kits and repair shop splices on trailing cable for underground coal mining equipment. Low voltage was defined as 0-660 volts a.c. or d.c., and medium was defined as 661-1000 volts a.c. Electrical engineers, superintendents, chief mechanics, and section foremen were interviewed at underground coal mines. At repair shops we interviewed splicers, shop foremen, and shop managers. We spoke to product managers and field salesmen from splice kit manufacturing companies. Lastly, we maintained contact with various researchers and enforcement people in the area of trailing cable splices.

The physical examination of splices was done by dissecting splices on flat and round cables used on shuttle cars and mining machines in underground coal mining. The main purpose was to gather information about splicing problems by examining splices on all types of trailing cables. A total of 60 splices which were made underground and in shops were collected from repair shops and coal operators over a six-month period. Each splice was collected by inspecting a scrap pile of splices and selecting samples with an emphasis on selecting shielded cable samples. Of all the samples collected, all of the shielded cables were examined.

SURVEY RESULTS

In our survey we gathered information mostly on low voltage reeled or spooled cables which are damaged most often. The types of information gathered were considered applicable to medium voltage dragged cable. In general, most of the splice kit problems which were brought to our attention involved installation and premature failures. The long-term performance was difficult for most coal mine operators to determine.

Splice Kit Selection

The reasons often given for selecting a given splice kit were:

1) Mine mechanics preferred a particular kit after many had been tried. The prime reason for selection was ease of application of the jacket.

2) The use of a certain kit minimizes violations. (The jacket remains in place with proper identification.)

3) The kit jacket can be easily repaired to bring about compliance.

4) The excellent training program given by the supplier of a kit.

5) Low or medium priced kit. (The highest priced kit was often used as a reason for selecting the medium priced kit.)

6) The type of splice kit selected for use on low voltage cable is most often used on medium voltage cable if the kit supplier has an approved kit.

There was little concern about the specifics of flex life, moisture resistance, and abrasion resistance of the splice kit. The mine operator's emphasis is on compliance of the jacket and he often lacks the engineering manpower to get involved in the details of splicing. Also, the jacket is easily inspected compared to other parts of the splice.

Types of Damage to Cable

In the low voltage range, the cable type most often damaged is the flat type cable. The flat cable is typically reeled or spooled from a machine which is usually a shuttle car. The shuttle car cable has more splicing done on it than other cables. According to the information we gathered, this cable is generally sent to a repair shop when it had 10-12 splices in it, but we have been told of cables having 20-25 splices in them. Common types of damage to spooled or reeled cable include:

1) Pin holes due to various kinds of abuse and continuity checks using a sharp instrument which punctures the jacket and insulation.

2) Over-tensioning damage which causes the conductor to shrink in size and break splices already in the cable.

3) Crushing, cutting, and abrasion which most often occur during back-spooling and when the machine drives over its own cable, or by the cable being dragged over surfaces of the machine.

4) Poorly made splices which cause conductor strand damage.

Other types of low and medium voltage cable connected to mining machines are typically of a round construction and are dragged behind the machine rather than being spooled. The medium voltage cables usually have metallic shields. Dragged cables are damaged far less often than the flat-reeled cable, and are sometimes changed-out after 3-4 splices have been made. The damage is usually concentrated in the first 50-100 feet of cable trailing behind the machine. Common types of damage to dragged cable include:

1) Crushing and cutting due to run-over or roof falls.

2) Abrasion from being dragged against the ribs.

3) Over-tensioning due to entanglement or failure to move the power center.

4) Poorly made splices in which conductor strand damage occurs or shielding which is improperly installed.

The proper splicing of dragged cable becomes very important when one considers that the cable is handled with bare hands more often than other cable. While the shuttle car cable can be operated with many pin-holes in the jacket and insulation, the dragged cable with pin-holes is much more hazardous due to the amount of handling.

Common Splice Kit Problems

The most serious problems appear to be adhesion of the splice jacket to the cable jacket and the connection of the power conductors on the shuttle car cable. Connection of the power conductors appears to be less of a problem on dragged cables since the cable is generally not as severely flexed.

The jacket is a multi-function component when compared with the other components of the splice kit. The multi-function performance and ease of application of the jacket are important criteria for selection of a specific splice kit by a coal mine operator. The combination of flexing, abrasion, crushing, and primarily poor workmanship practices contribute to poor moisture resistance and poor adhesion of the cable jacket.

The power and ground conductors in an underground splice are usually joined with a crimp connector. Our survey revealed that the primary problems with conductor rejoining was unequal tension in the conductors due to length differences. The length differences occurred during the crimping of the conductors. The problem was aggravated by using a crimping tool which requires spreading the conductors. It was reported that a crimp tool which minimizes the spreading of the conductors will generally minimize unequal tension problems. We also found that the use of connectors with flared ends was limited. The flared ends minimize damage to the conductor during cable flexing.

In addition to other preparation problems, cutting of the conductor strands while stripping the insulation was often mentioned during our survey.

Metallic Shielding

On medium voltage cable the metallic shielding most often applied over the spliced insulation is a tinned-copper braid type. The shielding is used on all 1000-volt non-reeled trailing cables. The metallic shielding represents another step in the splicing process, but according to our survey, evidently creates no additional problems other than the additional time required, and the fact that it is sometimes not replaced. The original cable shield and splice shield are most often joined by over-lapping with splice shield. Soldering was not performed in most cases.

Splice Repair Shops

Every coal operator we spoke with desired to have some standards of workmanship or testing to which they could uniformly require conformance. Most of the repair shops were in agreement feeling that their work was superior to a splice applied underground, but there was no uniform way to judge the quality.

PHYSICAL EXAMINATION RESULTS

The physical examination started with an insulation resistance test to determine if the splice had failed and then proceeded to the dissection. The dissection included inspection of the splice jacket, filler, shielding (if present), splice insulation, conductors, and connectors. The inspection was qualitative rather than quantitative. Based on the condition of the splice, an evaluation of the possible personnel hazard caused by the splice was also made.

The splices examined have not been proven to be representative of the entire coal industry, but the results expose a few of the fundamental problems.

Based on the observations made during the physical splice examinations, this is a summary of the results.

1) Twenty-three percent of the forty splices examined had actually failed.

2) Sixty-eight percent of the forty splices examined were considered in poor condition (including the 23% failed samples) and represented a potential hazard due to poor jacket bonding and/or protruding conductor strands.

3) Fifty-seven percent of the forty splices examined had jackets which were not bonded to the cable jacket. This was due to improper cleaning, oversize slip-on jackets, and lack of bonding of filler tape put over the cable jacket. A small number were due to unknown causes.

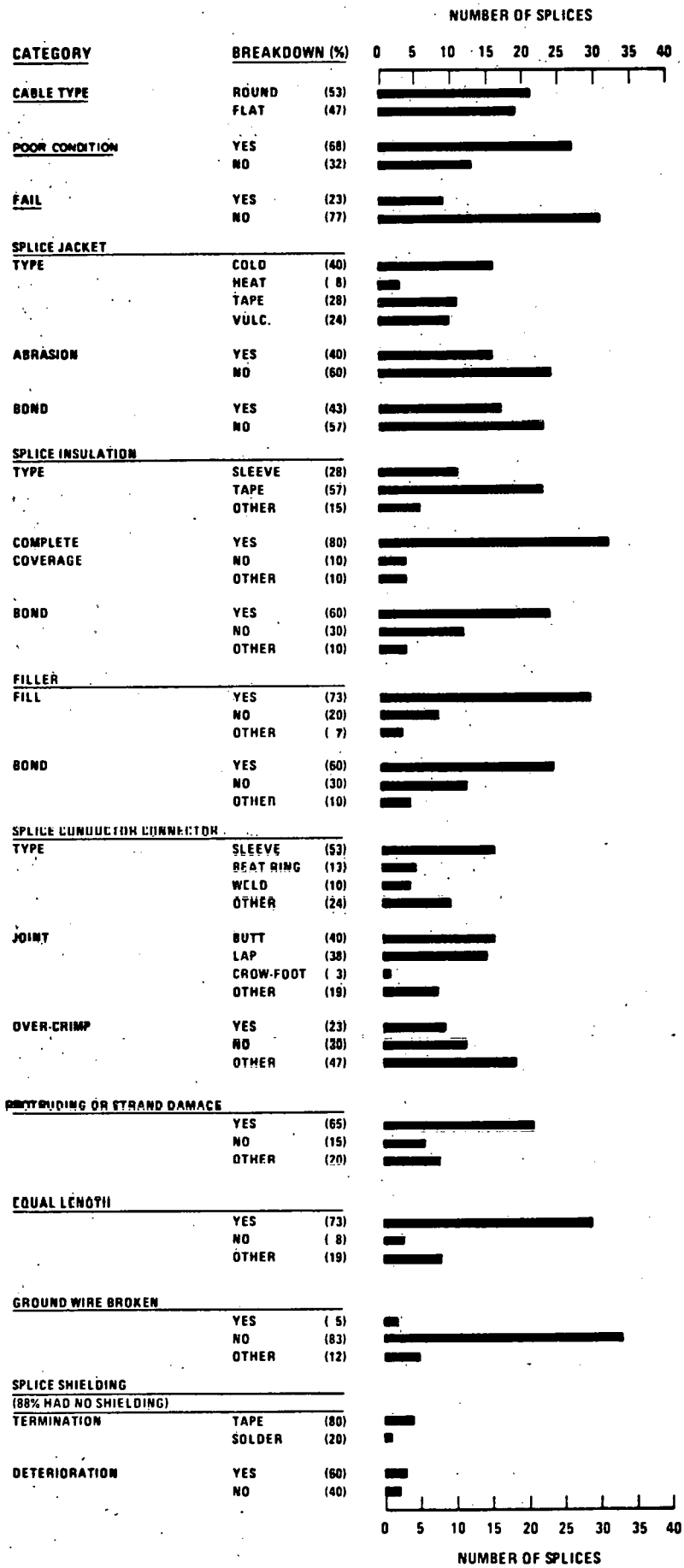
4) Sixty-five percent of the forty splices examined had protruding conductor strands or strand damage which could lead to insulation puncture. These were cut strands and lap type connections which have protruding strands.

5) The knitted type shielding used during splicing appears to deteriorate when the splice is flexed. The shielding be comes embedded in the insulation and breaks into small pieces.

6) The vulcanized splices often had wrinkled or folded insulation tape probably due to flexing of the entire splice. This doesn't appear to cause problems at the low and medium voltage levels, but could be a serious problem at high voltages.

Figure 1 shows the details of the examination results. A description of each category is in the next section.

Figure 1 SPLICE EXAMINATION RESULTS



Splice Examination Categories

The splices were examined according to 12 different categories of characteristics. A description of each follows.

Cable Type

Fifty-three (53) percent of the splices examined were round while the remainder were flat. During the collection of splices, an attempt was made to get as many round cables as possible since these would more likely be dragged instead of reeled cables. The flat splices were also collected since the most splicing occurs on flat cables.

Poor Condition

Splices were primarily determined to be in poor condition if a potential hazard existed due to deterioration of the jacket due to abrasion or poor bonding. Secondly, conductor strands penetrating into the splice insulation were considered to be a poor condition since they could eventually penetrate the jacket. Thirdly, incomplete coverage or binding of the conductor insulation was considered to be a poor condition if the jacket bond was questionable. Figure 2 shows a splice which was in poor condition due to poor jacket bond and numerous other problems.

Fail

All of the splices were insulation resistance tested power conductor to ground. Any readings below 1/2 meg-ohm were considered failed splices. Only twenty-three (23) percent of the splices had failed. Since the ends of our splices were not carefully terminated, any higher insulation resistance reading was considered to be an indication of a splice that had not failed. Figure 3 shows a splice that had failed when contact occurred between the power conductor and ground wire due to a split insulating sleeve which was undersized.

Splice Jacket

1) Type

Four types of splice jackets were examined: a cold, slip-on sleeve where the jacket is slipped over the splice area and held in place by an adhesive; a heat shrink sleeve where the jacket is heated and shrunk over the splice area; a tape which was either a flame resistant vinyl or self-vulcanizing rubber; and a vulcanized jacket which is made in the shop. During this examination, the jacket was considered to be a device for mechanical protection of the splice area and was considered as a component separate from the splice insulation.

2) Abrasion

All the splice jackets were visually judged for any observable deterioration or material removal caused by abrasion. We did not observe any failures directly attributable to abrasion.

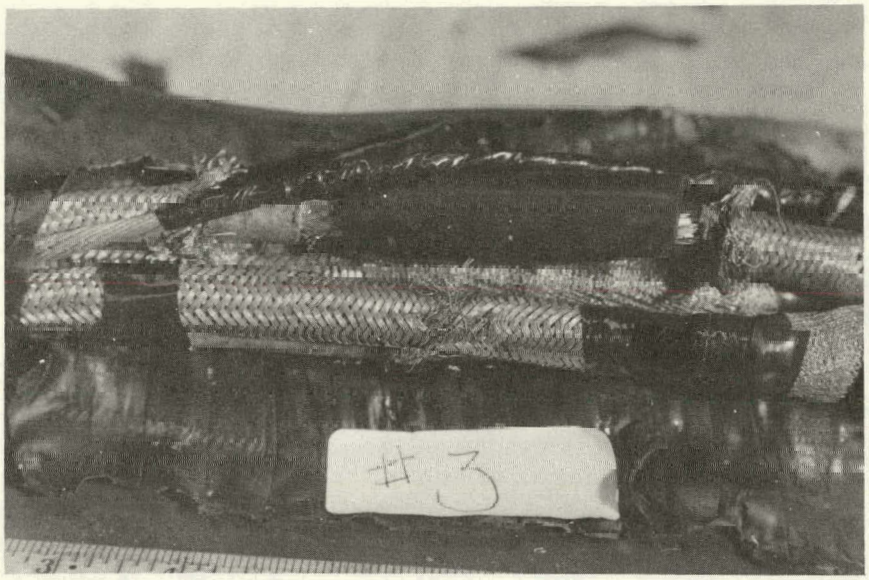


Figure 2 Shielded cable splice in poor condition due to poor jacket adhesion, protruding conductor strands, and insufficient insulation sleeve length.

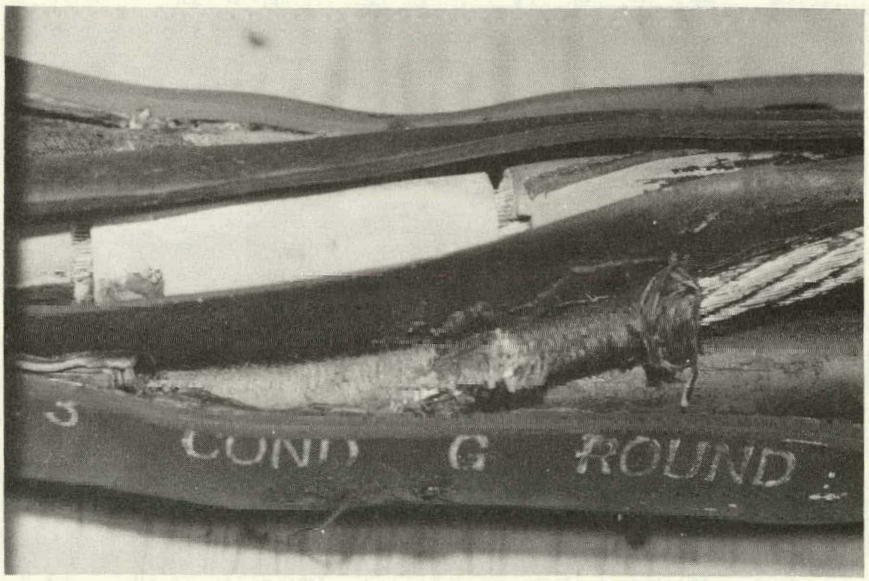


Figure 3 Splice failure due to inadequate closure of the insulating sleeve which allowed contact to occur between the power conductor and ground.

3) Bond

When the jacket was cut open the bonding of the adhesive was determined. The jacket where the adhesive bond between the splice and cable jacket had to be torn or broken to remove the jacket was considered to have a good bond. The actual strength of the bond was not measured. Figure 4 shows a common problem where the filler tape was not cemented to the cable jacket.

Splice Insulation

1) Type

Sleeve and tape type insulations were most commonly examined. The tape type of insulation included the vulcanized rubber tape. During this examination, the insulation was considered separately from the jacket. Its main function was to provide electrical insulation by completely covering the connected area and excluding the entrance of water by a good bond to the cable insulation.

2) Complete Coverage

Each splice was examined for complete insulation coverage of the re-joined conductors. For example, a sleeve may have been too short or the amount of tape used inadequate. On some splices, this was not applicable. Figure 2 shows an insulating sleeve of inadequate length.

3) Bond

Each splice was examined for bonding between the splice insulation and the factory cable insulation. The split sleeve type insulation was most often not bonded to the cable insulation.

Filler

1) Fill

Each splice was examined to determine if a filler was used. Filler is most often used on round cables where the cable jacket has been removed to prepare the cable for splicing.

2) Bond

The filler was examined for bonding to the cable jacket since the splice jacket was often bonded to filler.

Splice Conductor Connector

1) Type

The splices examined had sleeve, beat ring, and welded connectors.



Figure 4 Only the splice jacket and filler tape are pictured. The filler tape on the inside of the splice jacket was not cemented to the cable jacket.

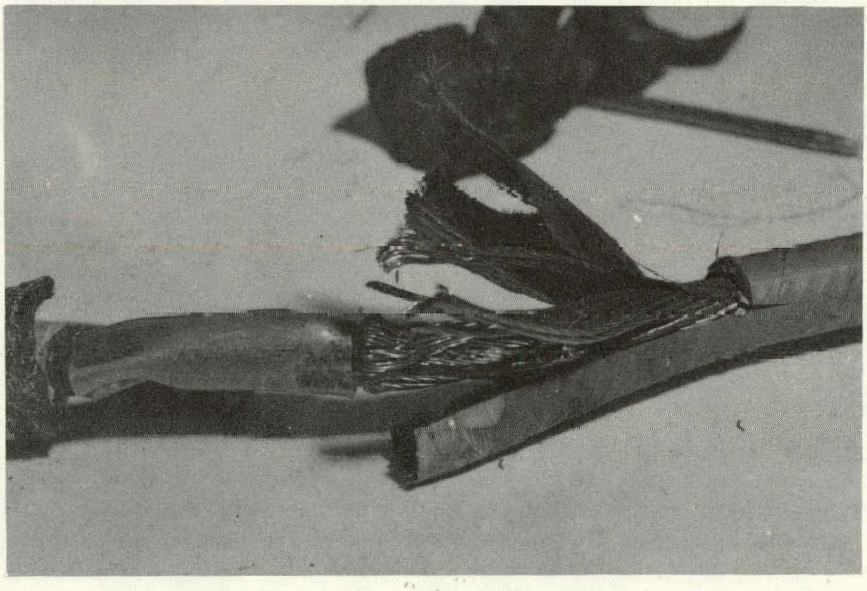


Figure 5 The photograph shows a splice where power conductor strands had been cut to fit the connector.

2) Joint

The conductors were joined by butt, lap, or crow-foot type joints.

3) Over-Crimp

The connectors were examined for obvious over-crimping and other problems such as oversizing or undersizing. Figure 5 shows a splice where strands had been cut to fit the connector.

Protruding or Strand Damage

Each splice was examined for cut or broken strands which are likely to occur during splicing or flexing of the cable. Also, the protrusion of any strands for a lap joint splice was checked. Figure 6 shows a splice that has protruding strands due to a lap type connection.

Equal Length

The splices were checked for equal length of the rejoined conductors. This was a visual check with emphasis on detecting conductor damage due to unequal tensions. This was not a quantitative measurement.

Ground Wire Broken

The ground wire continuity was determined by visual inspection. Figures 2 and 6 show a practice of taping uninsulated ground wires which could become a hazard.

Splice Shielding

1) Termination

The manner in which the knitted shielding tape was terminated by either vinyl tape or soldering was determined.

2) Deterioration

Any shield deterioration due to flexing where the knitted shield wires broke and began to penetrate the insulation was noted during the examination. Figure 7 shows where the knitted shield tape became embedded in the insulation tape.

CONCLUSIONS

Based on our survey of various personnel, our physical splice examinations, mine visits, and further study we conclude the following:

1) The primary problem with applying splice kits in the mine is poor workmanship due firstly to lack of training and secondly lack of concern for safety. Any training programs must present the proper procedures and build concern for safety.

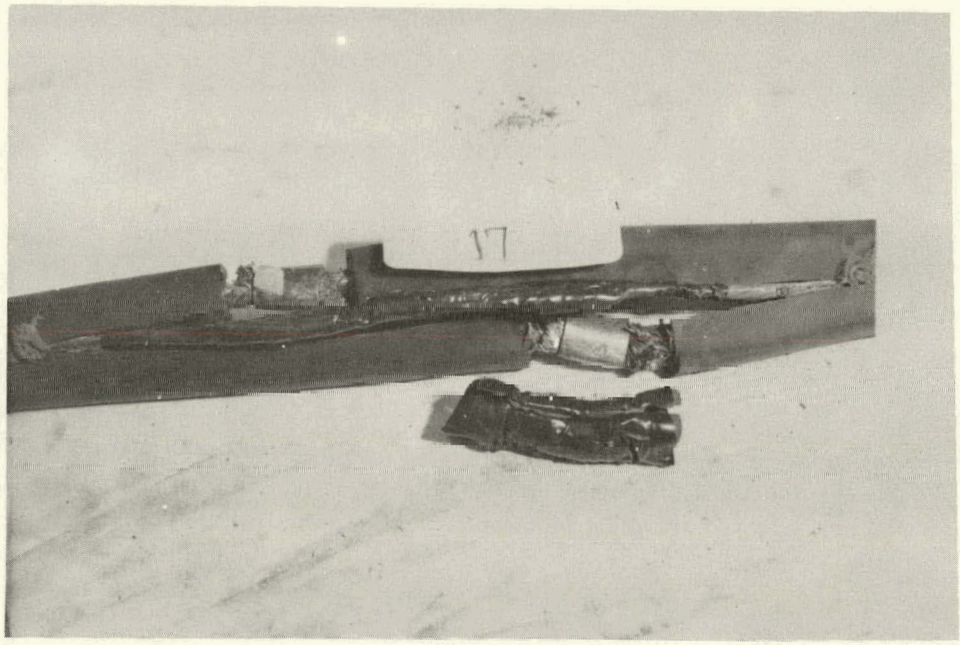


Figure 6 Flat cable splice showing protruding strands due to lap type rejoining of the conductors.



Figure 7 Photograph shows embedment of the splice shield in the insulation tapes.

2) The present situation on splicing is poor. A majority of low and medium voltage splices examined were in poor condition. Even though they had not failed they represented a potential hazard with continued use. Failures were due to poor workmanship.

3) To achieve improvements in the current splicing situation, splices applied to trailing cables must be (1) properly inspected to detect the flaws in assembly, (2) made of materials capable of performing according to a defined set of specifications, and (3) assembled by properly trained personnel.

4) Types of splices used on trailing cable should be designated as emergency, temporary, and permanent, where an emergency splice is any insulated device used to prevent a fault until a hazardous situation has been alleviated, a temporary splice is a splice made from an approved splice kit, and a permanent splice is a shop applied vulcanized or molded splice. The terminology used to describe splices should reflect reality.

5) The prime reasons for selection of a specific splice kit are ease of application of the jackets and the training program offered with the kit.

RECOMMENDATIONS

We would like to make the following general recommendations:

1) An instrument or procedures be developed for inspecting jacket bonds using ultra-sonics or some other non-destructive technique. This would be primarily intended for the coal mine operator.

2) Programs be undertaken in the areas of instructions and training of mine personnel for the application of splice kits. Training represents the greatest possibility for improving the splice kit safety and reliability.

3) More specific requirements be developed for trailing cable splicing procedures and materials. This should be a coordinated approach which takes into account requirements for splice kits, repair kits, shop splices, and cable maintenance. This should be done in the following subject areas listed in order of importance.

   a) Training of mine mechanics and electricians

   b) Splice testing for both mechanical and electrical performance

   c) Splice insulation for water exclusion

d) Jacket adhesion

e) Splice kit instruction sheet and training manual

f) Jacket flame resistance and for jacket smoke, toxicity, and corrosive products

g) Penetration resistant insulation

h) Connector - crimping systems for components and flexing

i) Binding of conductor strand

j) Cable cleaning materials for hand cleaning, solvents, and abrading

k) Shielding coverage and termination

l) Shield strand protection

m) Minimum application temperature and operating temperature

n) Packaging of splice kits for non-breakable containers and package strength

o) Cable preparation for penciling and tools

p) Shelf life

q) Filler type and quantity

r) Jacket identification

A PORTABLE TRANSIENT DETECTOR FOR USE IN UNDERGROUND COAL MINES

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ABSTRACT

Excessive voltage transients in mine power systems can deteriorate the dielectric material in transformers and cables. A knowledge of the transients can be obtained by measuring them with available recorders. However, commercially available transient recorders have several disadvantages including cost, form of output, portability and not being suited for use in mine environments.

The transient detector described in this report can measure the rise time, pulse width, and peak value. These parameters were chosen because they correlate quite well with the degree of insulation damage. The output is recorded on a digital thermal printer. The digital logic uses CMOS, and analog circuits are designed to minimize power drain.

The detector is used with a voltage divider to connect to the mine power system. This paper concludes with recommendations on the in-mine application of the portable transient detector.

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INTRODUCTION

Voltage transients are a fact of life on almost every power system. Careful design, within the technical and economic constraints of a system, will result in a reduction of transient-related component failures. However, the typical problem facing the mine power engineer is not one of new system design, but rather the economic and safe operation of an existing system. The problem then, is to assess the nature of transients on the power system, and to select an appropriate method to eliminate them or reduce their damage.

The sources of transients, as well as methods for minimizing transient-related failures, have been documented in the literature: for example references 1 and 2. Consequently, these topics are not discussed in this paper, although a few general comments are in order.

The major source of abnormal transients in the mine power system is from the switching operations of air, oil, or vacuum circuit breakers with the effect being most pronounced in mine power systems which use vacuum circuit breakers. The first type, known as a chopping transient occurs when a high-efficiency circuit breaker interrupts a small current such as a transformer's exciting current. In this case the current is forced to zero before the natural current zero, thereby trapping magnetic energy in the power system that is spontaneously released as a large voltage disturbance (3). The other transient source during switching is due to prestrike and restrike. During initial energization of system capacitance substantial overvoltages (three to seven times line voltage) may occur if the initial inrush current is momentarily stopped and then followed by a subsequent prestrike or contact closure (4).

Since in the usual mining operation high-voltage circuit breakers are called upon most often to function as switches rather than as protective devices, the potential for transient-induced failure is great. However many failures are unjustly attributed to high-voltage transients. The systematic use of the detector described in this paper can provide the necessary information for isolating and reducing the number of transient-related failures.

Transient Measurement

Several manufacturers market transient measuring devices for use in the laboratory. The better ones provide an almost perfect digital record of any voltage disturbances. One of the best available detectors was utilized for in-mine investigations. Although it performed admirably in the lab it would not function correctly in the mine environment, and was not portable. A smaller and less expensive detector was also tried. It functioned correctly in the mine environment and was somewhat portable, but it did not have sufficient resolution. After additional and fruitless attempts to locate a suitable detector a decision was made to design and build one.

The detector was designed to have the following general characteristics:

1. portability - the detector must not require additional equipment, such as an oscilloscope or tape recorder to function; the unit's volume must be less than one cubic foot and weigh less than twenty-five pounds; it must have its own power source,

2. cost - the detector's cost should be less than $200,

3. construction - the detector's circuits should be constructed from readily available components, and the construction should require only elementary skills in circuit board assembly, and

4. measurement parameters - the parameters recorded by the detector must be relatable to power system component specifications.

These characteristics have been incorporated into the final design.

Measurement Parameters

The choice for meeting the fourth general characteristic, listed previously, is to relate the detectors measured quantities to dielectric withstand properties. Specifically the detector was designed to measure the same quantities which are used in the BIL impulse wave test. These parameters, which are diagramatically illustrated in Figure 1, are peak value, rise time, and pulse width, which is the time to decline to 50% of peak value (5). These values can be used to evaluate the severity of a transient, and to select transient suppression devices.

These parameters are limited by the detector to the following values:

0 μS $\leq$ rise time $\leq$ 99 μS,
0 μS $\leq$ pulse width $\leq$ 99 μS, and
$\sim$1.5 V $\leq$ pulse height $\leq$ 10V.

The selected limits for rise time and pulse width are based on the type of transients which are most common in the mine power system. The range of the detector will allow recording transients up to 10 to 15 times line voltage, depending upon the construction of the voltage divider.

Circuit Description

The block diagram of the peak detector is given in Figure 2. The analog input circuit accepts the analog signal and generates control signals to the input control logic. The input control logic contains control logic, counter and analog-to-digital converter to compute the parameters. The output control logic gives the appropriate control signals to serialize and control the flow of information to the calculator. The data output logic contains shift registers, decoders, and analog multiplexers to condition the data for the calculator input. The clocks are generated with three inverters each. The various blocks are now described in detail.

Analog Input Circuit. The analog input circuitry can accept voltages in the range from -10 to +10 volts and the circuit is given in Figure 3. The 318 op-amps are used to rectify the incoming pulse and the rectified voltage is scaled before it reaches the inputs of the 306 comparator. This comparator works well only in the $\pm$ 5 volt range. The 306 comparator and the 3140 bifet op-amp are used for a high speed analog peak detector. The output of the peak detector is fed to a non inverting 3140 op-amp circuit to boost the voltage into the 0 to +10 voltage range for the analog-to-digital converter.

The three 311 comparators are used to drive the digital logic. The typical timing diagrams are given in Figure 4 for these signals. The $\overline{\text{start}}$ = $\overline{\text{ST}}$ signal changes state when the threshold level is exceeded. The $\overline{\text{width}}$ = $\overline{\text{WD}}$ produces a 1 → 0 transition on the falling edge of the pulse at 50% of initial peak value. The rise $\overline{\text{time}}$ = $\overline{\text{RT}}$

produces negative pulses everytime the 306 comparator dumps more charge onto the holding capacitor during the period the analog pulse is rising. This signal can be used to measure rise time. The PEAK signal is proportional to the peak of the analog pulse.

Input Control Logic. The peak detector is designed for functioning under battery power. Therefore, all digital logic uses CMOS circuits. All integrated circuits (IC's) are the 4000 series unless specified otherwise (6). Most IC's in the Figures list only the last two or three digits. The number within the circle represents the location of the IC. The input control circuit is given in Figure 5.

The inverters (4049), the D flip-flops (4013) and the AND gates (4073 and 4081) are the logic IC's which accept signals from the start (ST), rise time (RT), and width (WD) input signals and generate control signals. The enable signal (EN) insures the 14518 dual decade counter counts during the appropriate range of the analog pulse. The master reset (MR) signal resets the entire system. The load rise time (LRT) loads the count from the dual decade counter into a shift register in the data output circuit everytime a rise time (RT) pulse is generated. The start of conversion (SOC) signal is a pulse which will activate the analog-to-digital converter (MM5357). The end of conversion (EOC) signal enables the output control logic. The typical timing diagram for these signals is given Figure 4.

Data Output Logic. The data output logic is given in Figure 6. The data output logic contains shift registers (4021's & 4013) which will serialize the data into 4 bits. The decoder (4028) controls the analog multiplexer switches (3-4016's) which will simulate contact closures on the TI 5050 calculator keyboard.

Output Control Logic. The output control logic is given in Figure 7 and the associated timing diagram in Figure 8. The output control logic provides the appropriate control signals to the data output logic (Figure 6) to serialize and print each character.

The D flip-flops (4013) are used to generate the load (LD) signal and enable the counter (4024) after the data conversion is complete in response to a $0 \rightarrow 1$ transition on end of conversion (EOC). The states of the counter are decoded to provide shift pulses (SH), enable pulses (EN), identifying pulses (#), and total pulses (TOT) in the proper time sequence.

The system reset pulse (SR) resets a D flip-flop in the input control logic which in turn resets the entire systems via the master reset (MR). This insures no false resetting will occur. The peak detector will also ignore any new analog pulse until the peak detector is reset.

Calculator (TI 5050). An inexpensive digital printing device with low power requirements was needed and a thermal printer meets these requirements. Most thermal printers print each character one at a time

with a 5 x 7 dot matrix. This requires a significant amount of logic to sequence all of the signals which are required to control a thermal printer. Therefore, it was decided to use the logic which already existed in the calculator (TI 5050) to perform the printing task.

A general interface to a thermal printer would allow the printing of special headings and characters. Interfacing directly to the keyboard of the calculator restricted our data format to the form which is indicated in Figure 9.

Clocks. Each clock (Figure 10) was built from three inverters plus resistors and capacitors. This design was chosen because it is relatively immuned to power supply variations. A variable resistor was used to provide frequency adjustment on the 1 MHz clock.

Mine Application of the Detector.

This detector can be assembled and packaged by any technician. The other assembly required for use of this detector is a voltage divider. There is an abundance of information reported in the literature on dividers, including reference 3. When constructed the divider can be packaged in a small wooden box. After testing the detector is ready for use.

The transient detector can be used on both a-c and d-c power systems. Because of its portability it can be placed and left inside a switchhouse, load center, or wherever required. At some later time the detector can be recovered along with the paper tape listing the parameters of all detected transients. By replacing the threshold level adjustment resistor with an external variable resistor the detector can be quickly adjusted to record only transients which are greater than a preselected value.

As stated earlier the utility of this detector is in collecting sufficient data to enable the selection of an appropriate transient-suppression device. Accordingly the accuracy of the measured parameters is within 20% for the worst case of an almost square pulse, and less than 10% for the typical case. Field experience with this device has demonstrated its ability to collect useful data with minimum effort.

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1. Morley, L. A., Kohler, J. L., and Chubb, W. R., "Coal Mine Electrical System Evaluation, Volume II - Transient Analysis," Annual Report on USBM Grant NO. G0155003, The Pennsylvania State University, May 15, 1977.

2. Kunjara, N. A., Russell, G. P. and Stamek, E. K., "Prediction and Suppression of Electrical Transients in Mine Electrical Systems," paper presented at the 1977 Annual Meeting of the IEEE Industry Applications Society, San Francisco, California, Oct. 1977.

3. Greenwood, A. N., Electrical Transients in Power Systems, John Wiley and Sons, Inc, New York, 1971.

4. Pflantz H. M., "Analysis of Multiple Prestrike and Interruption Phenomena in Capacitive Circuits," paper presented at the IEEE Summer Meeting of the Power Engineering Society, San Francisco, July, 1972.

5. IEEE Standard 141-1976, p. 148.

6. RCA Integrated Circuits, RCA Solid State, Box 3200, Somerville, N. J. USA.

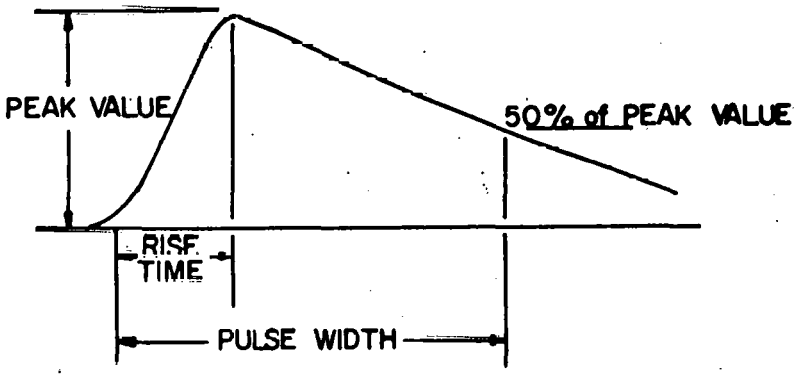


Figure 1. BIL Impulse Wave Test

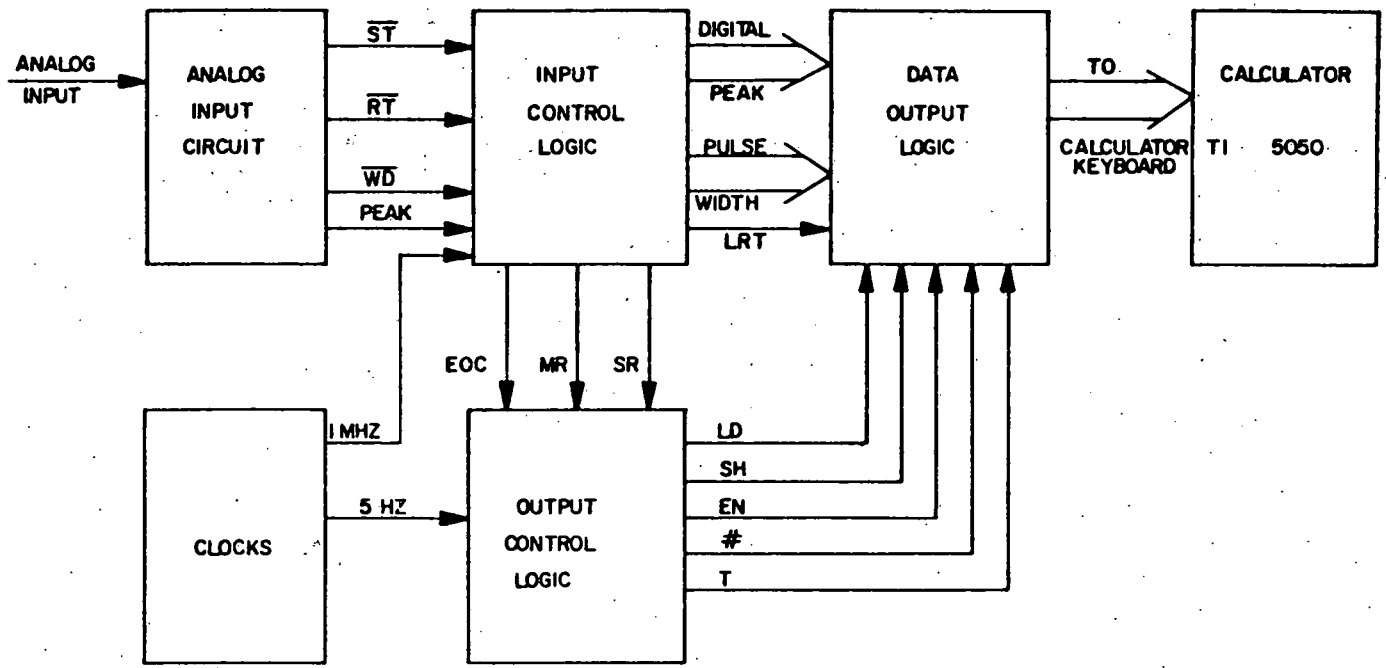


Figure 2. Block Diagram of Peak Detector

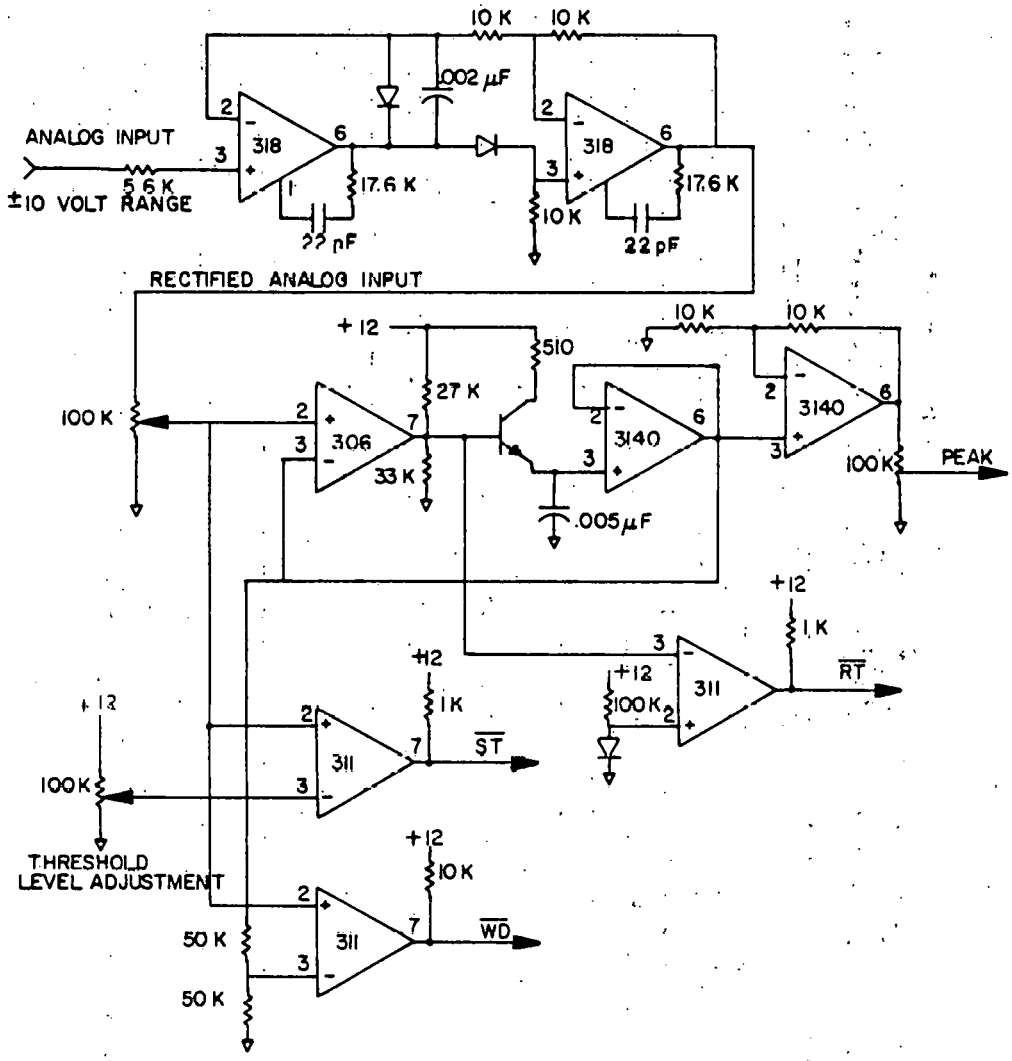


Figure 3. Analog Input Circuitry

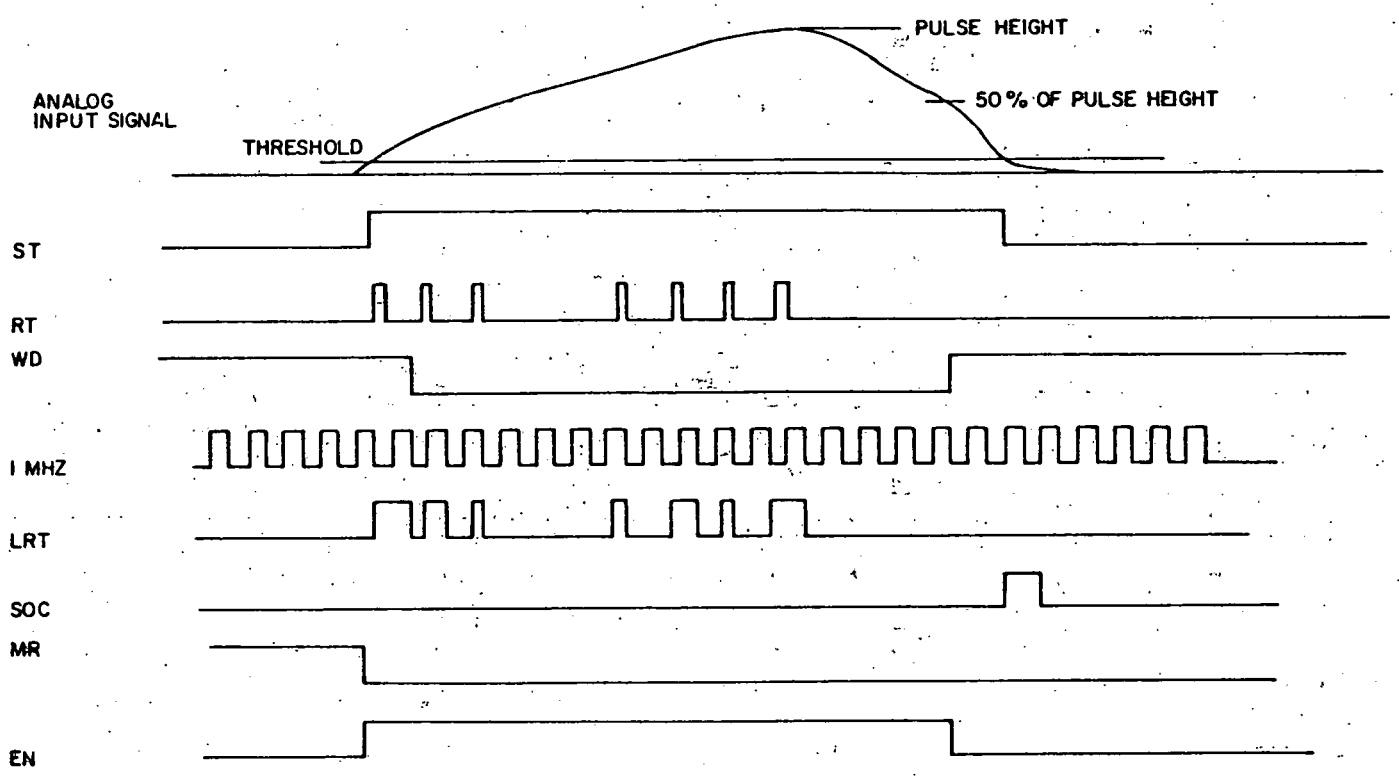


Figure 4. Timing Diagrams for Input Circuitry

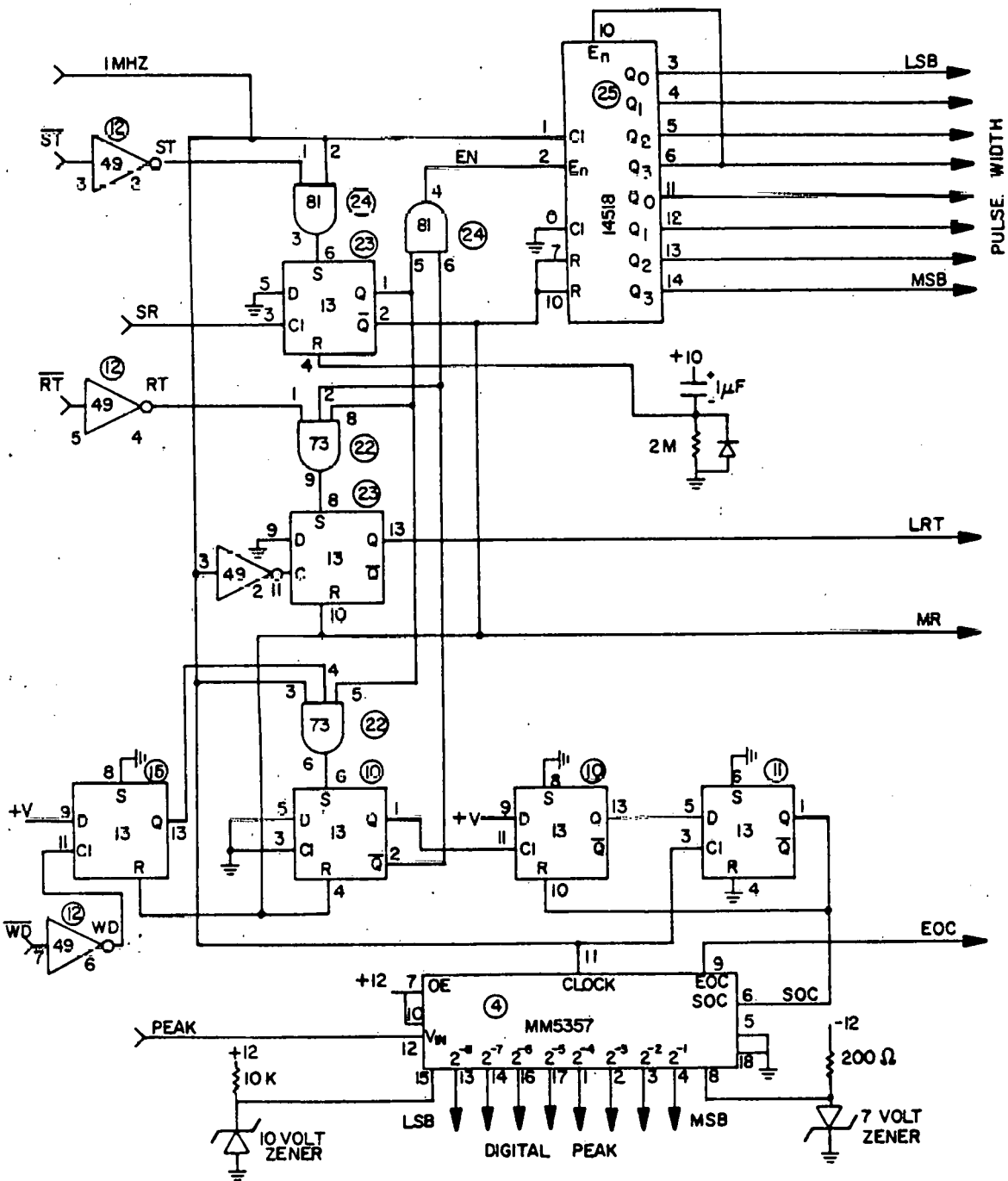


Figure 5. Input Control Logic

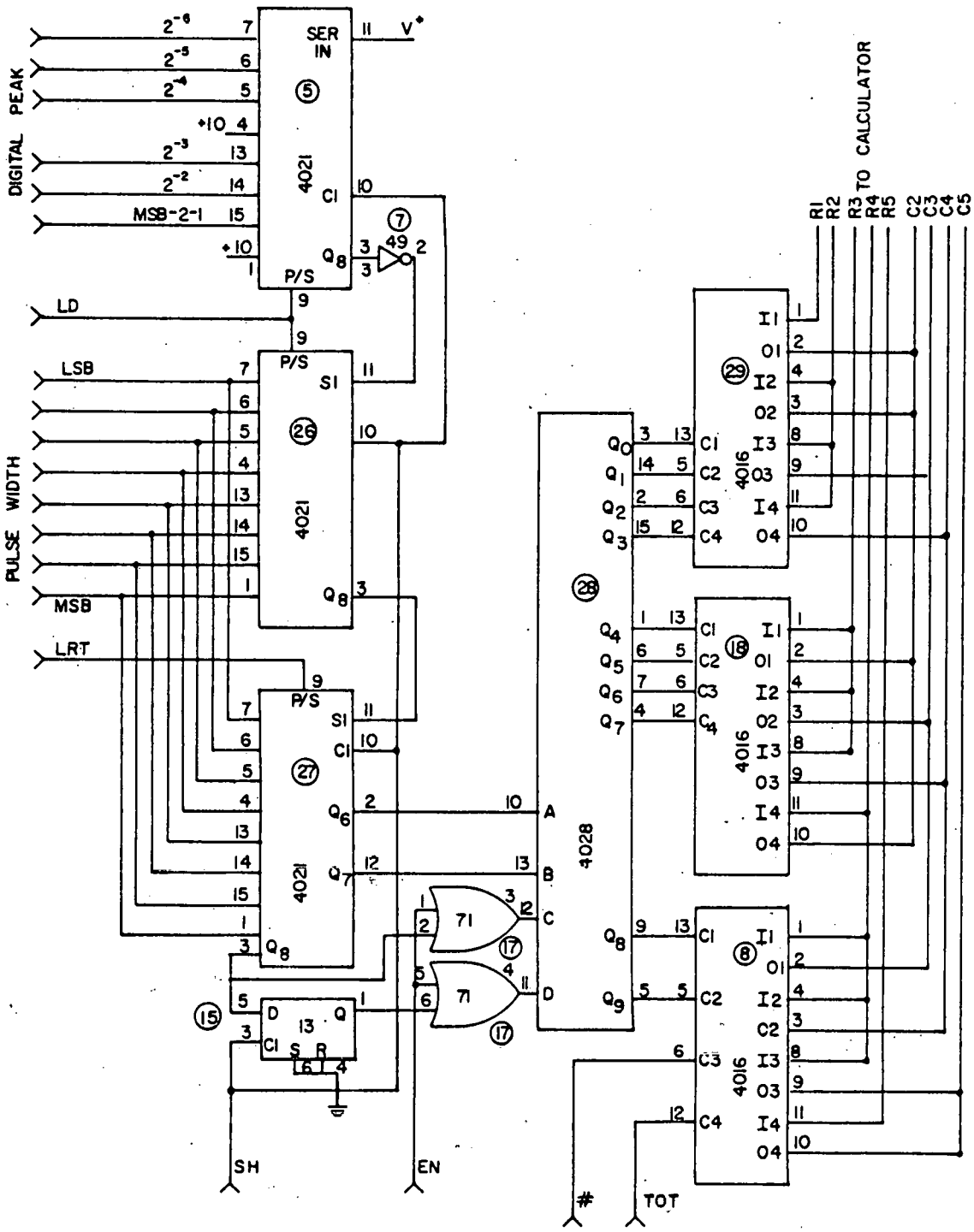


Figure 6. Data Output Logic

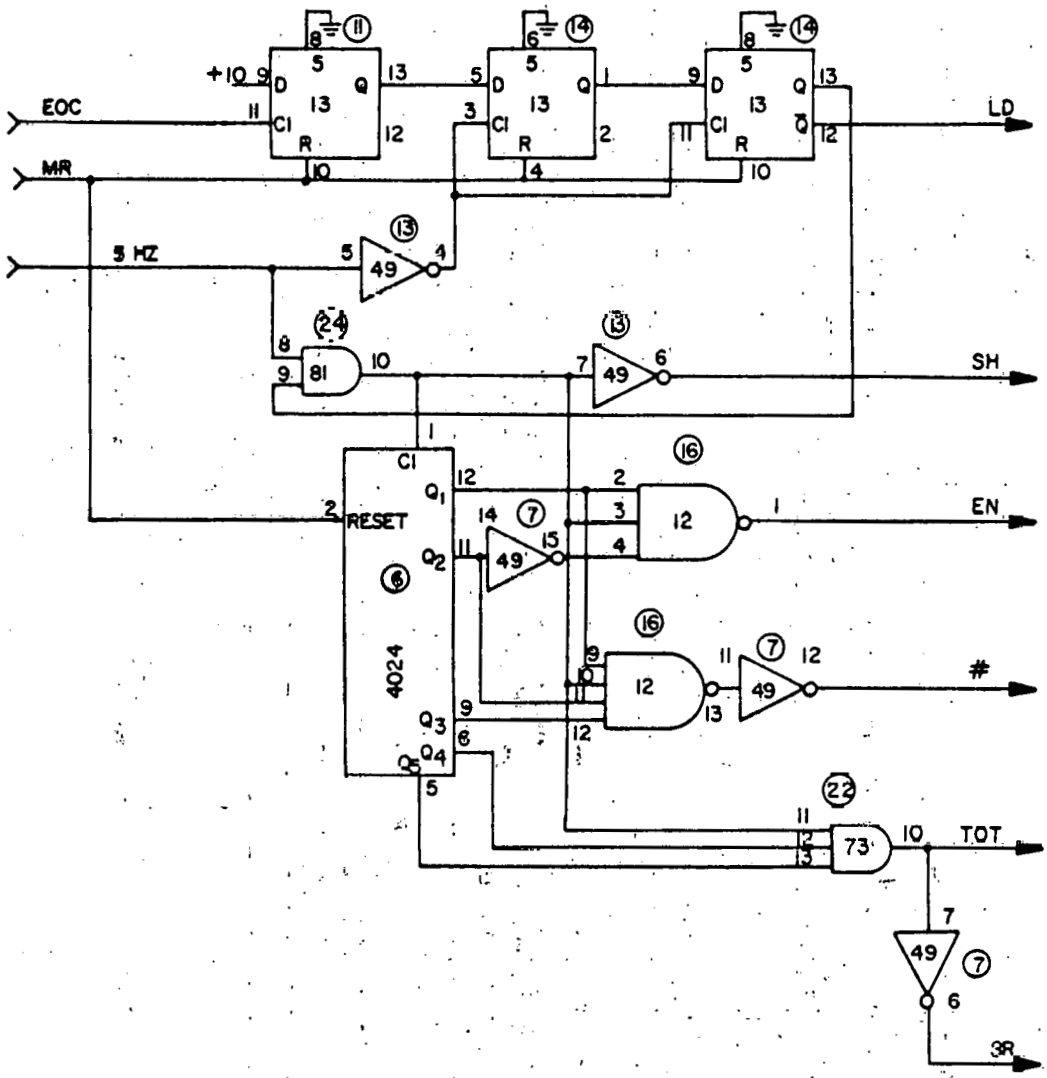


Figure 7. Output Control Logic

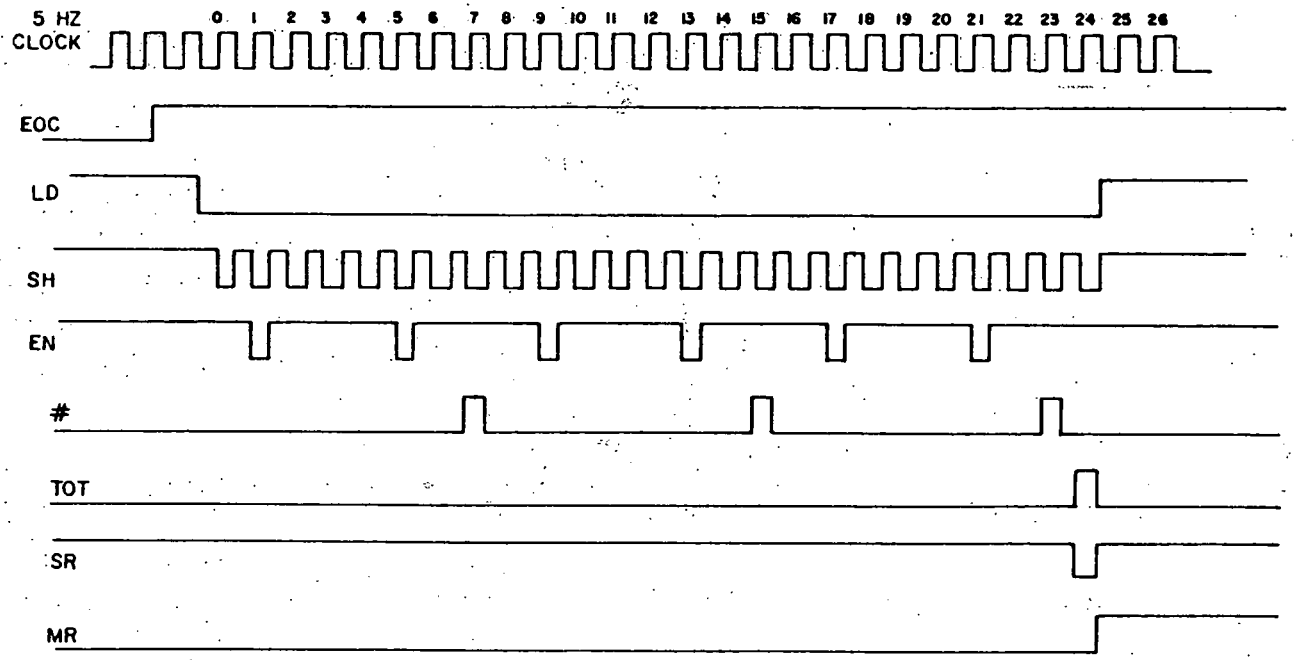


Figure 8. Timing for Output to Calculator

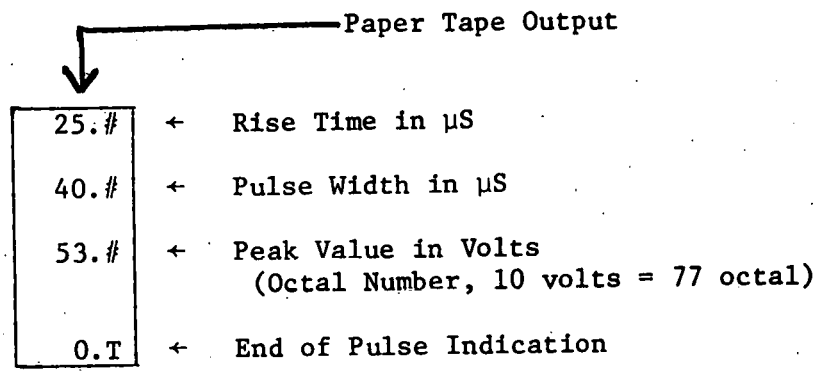


Figure 9. Data Format for Thermal Printer

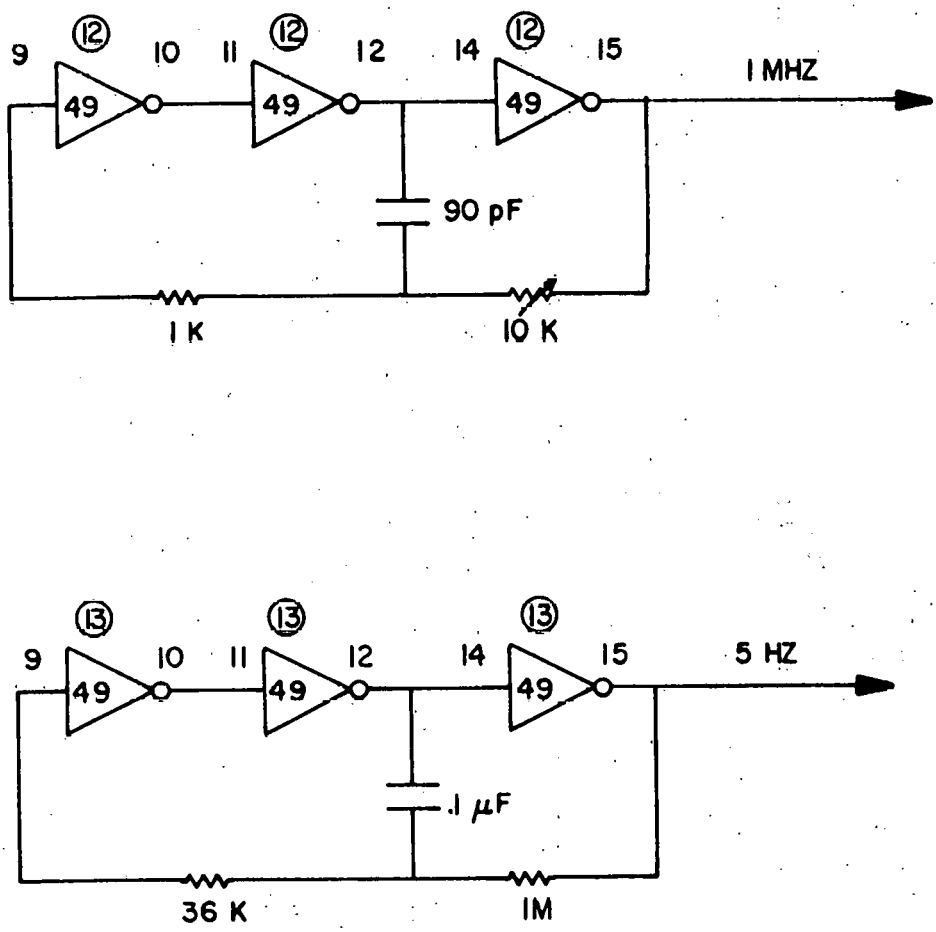


Figure 10. Clock Circuits

TEST CRITERIA FOR EVALUATION OF

TRAILING CABLE SPLICES

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ABSTRACT

The Mine Safety and Health Administration (MSHA) Technical Support, Beckley Electrical Testing Project (BETP) personnel at Beckley, West Virginia, are currently conducting field and in-house testing and evaluation for underground trailing cable splices. Pertinent parts of the test machine and methods of accomplishing the tests are discussed herein. At the present time, splice kits commercially available are being tested. Vulcanized splices are also being tested and evaluated for effectiveness in excluding moisture, withstanding severe bending, and resisting abrasion and tension.

INTRODUCTION

The Federal Mine Safety and Health Act of 1977, Section 306(e) establishes general criteria for permanent splicing of trailing cables. (FIGURE 1a) Section 75.604, Code of Federal Regulations, Title 30, contains the same information as Section 306(e) of the Act. The following electrical terminologies have not, as yet, been thoroughly defined by MSHA.

1. Mechanically strong splices
2. Adequate electrical conductivity
3. Adequate flexibility
4. Effectively insulated
5. Effectively sealed to exclude moisture
6. Vulcanized or otherwise treated
7. Good bonding to the outer jacket

In January 1977, the Beckley Electrical Testing Project was given the task of reviewing a draft of proposed regulations developed by the U. S. Bureau of Mines and Beckley Technical Support. It became readily apparent that additional testing and evaluation was needed to develop a meaningful regulation. Therefore, a testing program was established by BETP's Electromechanical Group.

FLEXIBILITY TEST

In mid-August 1977, the Electromechanical Group designed and constructed a flex machine to operate totally pneumatic systems. (FIGURE 2c) This machine was built to simulate a shuttle car's sheave wheels bending effect. Hopefully, this would simulate the underground performance.

A spliced cable 17 feet long is mounted on the machine. (FIGURE 2a) This cable is made to ride over and to wrap around the wheels approximately 360 degrees. (FIGURE 3b) Each end of the cable specimen is clamped with a hydraulic jack to hold the cable firmly while the test is being conducted. (FIGURE 4a)

The traverse plate (FIGURE 3c) is an integral part of the pneumatic reciprocating cylinder that imparts the fore and aft motion of the plate. The bidirectional stroke of the traverse plate is counted as one cycle. The number of cycles is registered on an electromechanical recorder. (FIGURE 5a) The stroke of the plate may be accomplished selectively by relocating the reverse valves. (FIGURE 6a) The cycle stroke of the traverse plate can be controlled from five to twenty cycles per minute. This is accomplished by setting the pressure on the valving system center which is located on the adjacent side of the machine. (FIGURE 7b)

During the test, the cable is subject to tension. To provide this repulsing action, a tension cylinder was installed on the left side of the machine. (FIGURE 4c) Specimen are tensioned to 10 percent of the specified cable pull-out strength. (FIGURE 4b) The space between the valleys of the wheels is adjusted to a predetermined distance or equal to twice the diameter of the cable. (FIGURES 8a & 9a)

The cable's ground wire, pilot wire, and phase conductors are individually monitored by a set of current sensing control boxes. (FIGURE 10a) The monitor equipment was designed to deenergize the solenoid valve (FIGURE 7a) should any of the above wires/conductors become damaged or open-circuited. To restore the cyclic operation of the test machine, a bridge flip switch is turned to the "on" position. (FIGURE 10b)

As the traverse plate moves in either direction, the cable is pulled in over the periphery of the wheel, thereby allowing the spliced section to travel in a figure "S". (FIGURE 3b) At the termination of the stroke, the center of the splice has traveled approximately 16 inches past the tangential point of the wheel. (FIGURE 11a) At the duration of each test, the cable is monitored with a full load current until it no longer passes the full load current test. (FIGURES 5b, 12b, &13a) The applied current is based on the National Electrical Code standards. From the machine we can determine the ability of the spliced cable to withstand a severely repeated reversal bending effect around the sheave wheels.

PULL-OUT TEST

During the backspooling of a shuttle car cable, the cable is subject to high tension. For the purpose of simulating that tension in the laboratory, we constructed a tensile machine designed to operate hydraulically. (FIGURE 12) The system principally consists of a pneumatically-actuated pump, a set of flow meters, a regulatory valve, a bourdon pressure gauge, a hydraulic jack, and ohmmeters. The pull-out test is accomplished by mounting a completed splice of continuous loop configurations on both wheels. (FIGURE 12d) Ohmmeter test leads are connected before and after the spliced section in order to monitor the ground wire and the phase conductor continuity. (FIGURE 12a)

The hydraulic cylinder (FIGURE 12c) is then allowed to extend at a very slow rate by opening a control valve (FIGURE 12f) until the conductor or the ground wire is broken. The oil-filled pressure gauge will monitor the tensile strength offered by the wires. (FIGURE 12e)

ABRASION TEST

In January 1978, an abrasive device was installed on the traverse plate of the flex machine. (FIGURE 3c) The abrasive material used is made from a mild steel plate, ¼-inch thick and 2½-inches wide. The steel plate can be positioned at any desired attack angle (FIGURES 3a & 3b) on both flat sides of the spliced cable. The pressure against the insulator material is adjusted to a predetermined pressure before the abrasive test is conducted. The spliced section is moved across the abrasive material during the bidirectional stroke of the traverse plate. This allows the abrasive material to abraid the entire section or allows the conductor to be exposed to the environment. The test may be conducted in a dry manner (FIGURE 2c) or with liquid solution in a tank which is an integral part of the flex machine. (FIGURES 2b, 14a, & 17)

During the flexing and abrasion actions encountered by the slip-on rubber jackets, end lifting exists. (FIGURE 15a) Eventually, moisture will enter the phase conductor and provide an electrical conductive path which, during the actual operation creates imminent hazards to the peronnel handling the cable. The "lipping" of a jacket is due to the poor bonding of the insulator material and the jacket.

CURRENT LEAKAGE TEST

It is anticipated that by the end of 1978, a voltage withstanding test will be conducted in the BETP laboratory. The purpose of this type of test is to detect if there is any leakage between the cable insulator material and the ground wire and conductor. Electrical leakage causes hazards and thus poses a threat to the safety of the miners.

The test is conducted by taking a selected phase conductor as a common terminal end and testing it against any of the following- the ground wire, phase conductors, monitor check wire, pilot check, cable shield, ground check wire, and the water, or any combination of the above. The

presence of a leak is detected on the built-in meter of the equipment. (FIGURE 16d) The test equipment can be operated in either alternating current or direct current potential. (FIGURE 16a)

With the exception of the cable end, (FIGURE 16c) the cable is immersed in a water-filled 12 X 15 X 50- inch plastic tank for 24 hours before it is tested.

UNDERGROUND TEST

In April 1978, field tests were initiated for comparison with the results already obtained in the laboratory. These tests will relate the actual effects of abrasion, tension, and bending actions taking place during the constant paying out and rewinding of a shuttle car cable. This test is a good approach because it permits this multitude of factors to be tested at one time. This way it also enables us to evaluate the adaptability of a splice kit for underground use. In addition, underground tests will provide us an opportunity to give assistance in the proper installation techniques and the use of appropriate installation tools for the use of splice kits.

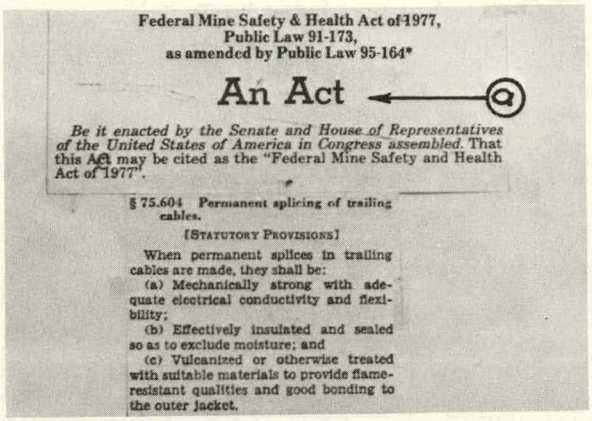
CONCLUSION

The Beckley Electrical Testing Project is presently observing cable splices in the field. Splices are installed with selected components from several splice kits. The application of this system has revealed a significant increase in the pull-out strength and flex life of the splices. Eventually, through our research and testing methods, an optimized splice kit will be available.

By applying proper techniques, using an appropriate installation tool and an optimized splice kit, machine down time will be at a minimum. This will also result in greater safety and reliability service to the miner.

Although extensive field testing is still required, it is hoped that by the end of 1978, the results obtained in our laboratory can be compared with input data from the field.

A seminar covering cable splicing will be initiated by BETP's Electromechanican Group for the miner in the near future.



Federal Mine Safety & Health Act of 1977,
Public Law 91-173,
as amended by Public Law 95-164*

An Act (a)

Be it enacted by the Senate and House of Representatives of the United States of America in Congress assembled. That this Act may be cited as the "Federal Mine Safety and Health Act of 1977".

§ 75.604 Permanent splicing of trailing cables.

[STATUTORY PROVISIONS]

When permanent splices in trailing cables are made, they shall be:

(a) Mechanically strong with adequate electrical conductivity and flexibility;

(b) Effectively insulated and sealed so as to exclude moisture; and

(c) Vulcanized or otherwise treated with suitable materials to provide flame-resistant qualities and good bonding to the outer jacket.

Figure 1

a. Act of 1977

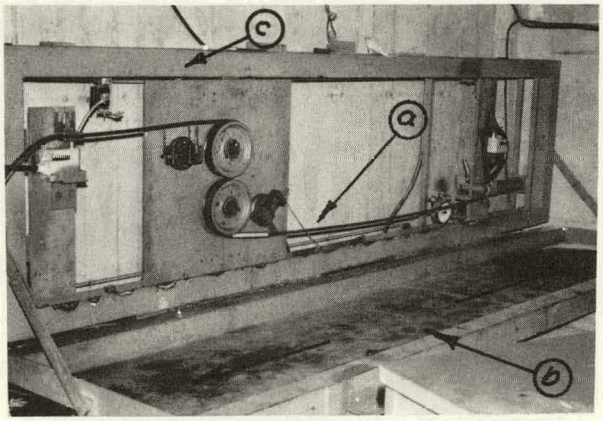


Figure 2

a. Seventeen-foot Long Cable
b. Water Tank
c. Flex Machine at a Vertical Position

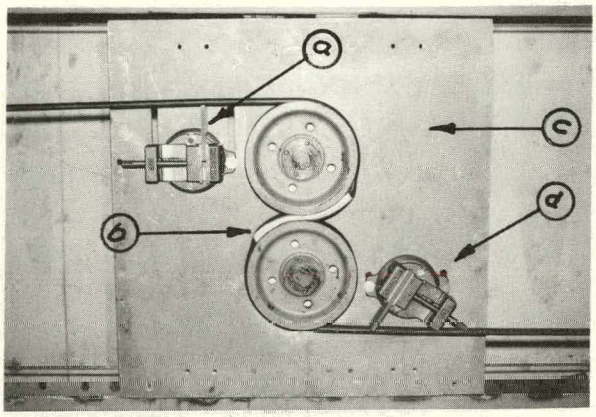


Figure 3

a. Abrasive Plate at a Perpendicular Position
b. Splice Section in Figure "S"
c. Traverse Plate
d. Abrasive Device at an Angle Position

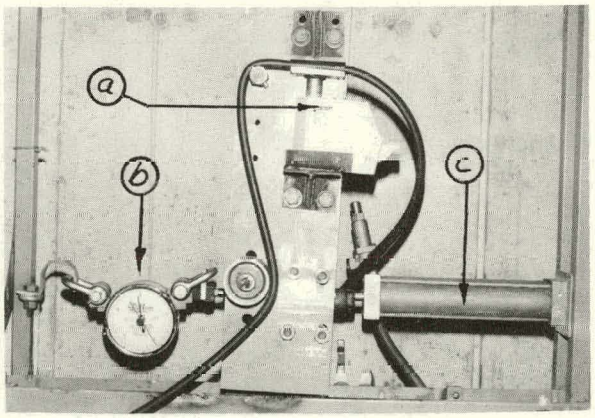


Figure 4

a. Hydraulic Jack Cable Clamp
b. Dynamometer
c. Pneumatic Tension Cylinder

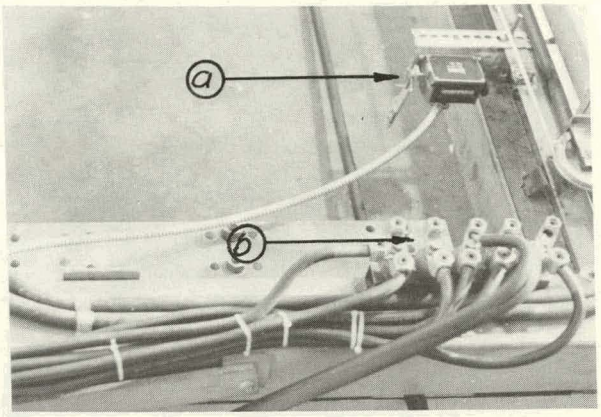


Figure 5

a. Cycle Counter/Recorder Device
b. Phase Conductor Full Load Current Terminals

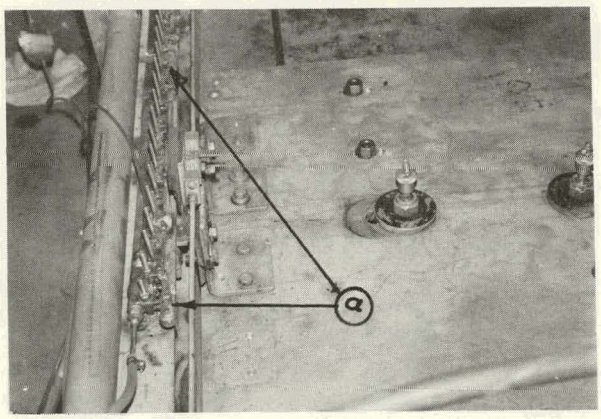


Figure 6

a. Pneumatic Reverse Valves

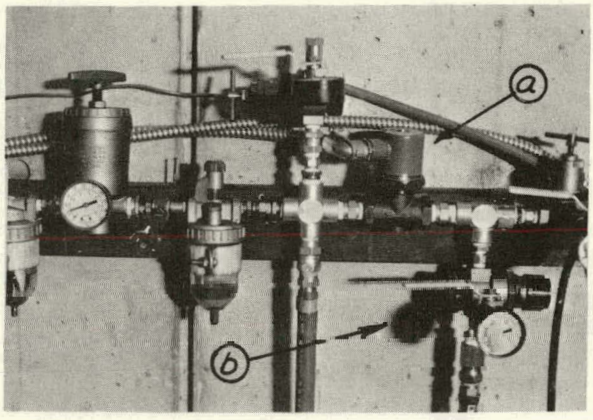


Figure 7

a. Solenoid Valve
b. Traverse Speed Regulator Valve

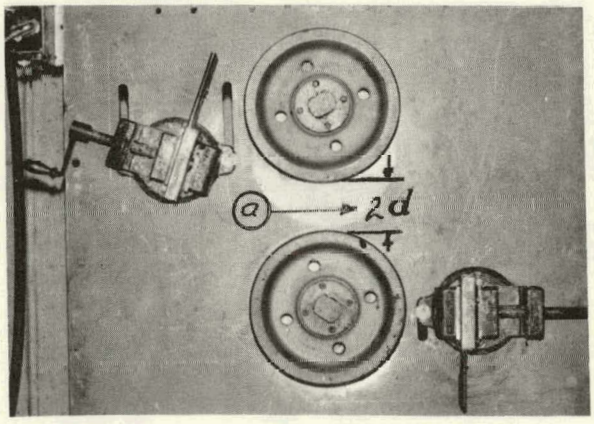


Figure 8

a. Sheave Wheels Valley at Maximum Distance

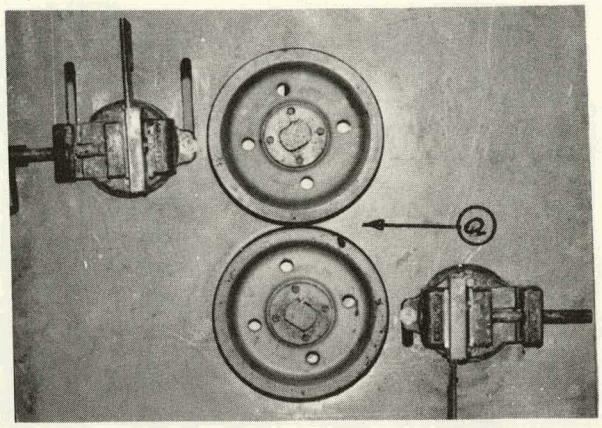


Figure 9

a. Sheave Wheels Valley at Minimum Distance

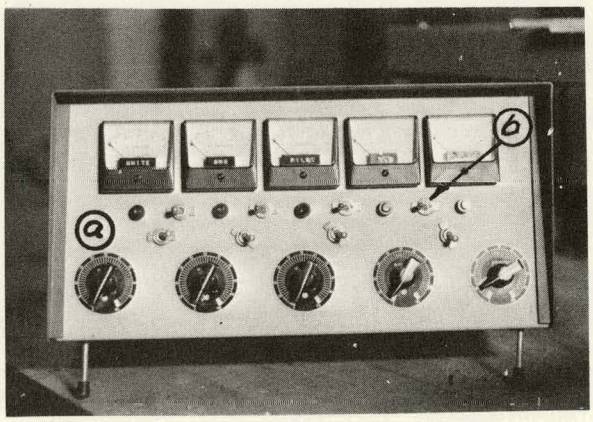


Figure 10

a. Ground Wire/Pilot Wire and Phase Conductor Current Sensing Box
b. Bridger Switches

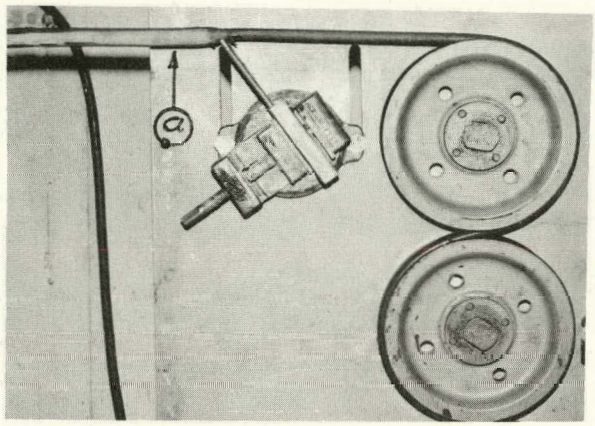


Figure 11

a. Splice Section Past the Abrasive Plate

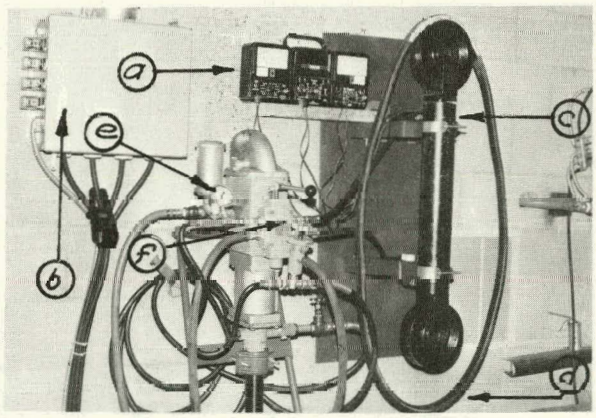


Figure 12

a. Monitoring Ohmmeters and Test Leads
b. Phase Conductor Full Load Monitoring Current Box
c. Hydraulic Cylinder
d. Continuous Loop Configuration of a Splice Cable
e. Bourdon Pressure Gauge
f. Hydraulic Control Valve

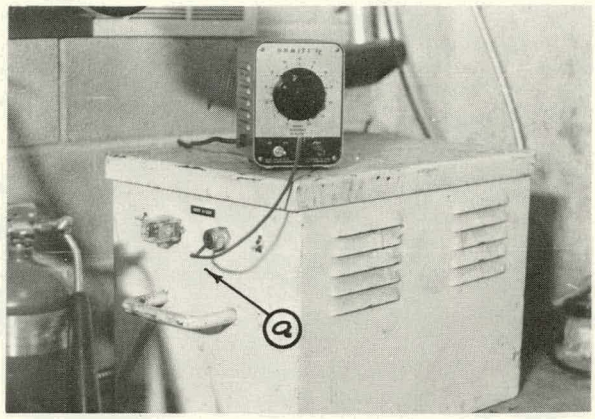


Figure 13

a. Ground Wire Full Load Current Monitoring Box

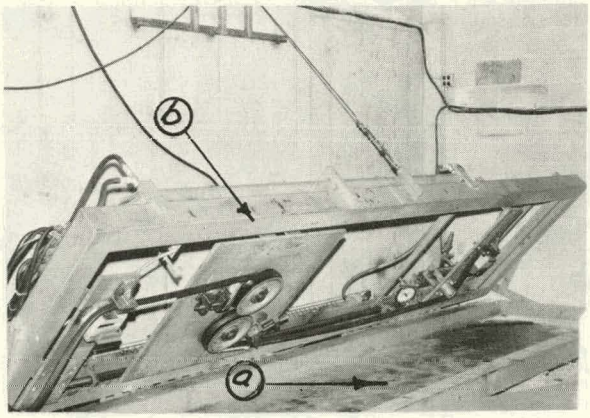


Figure 14

a. Water Tank
b. Flex Machine Held at 45 Degrees

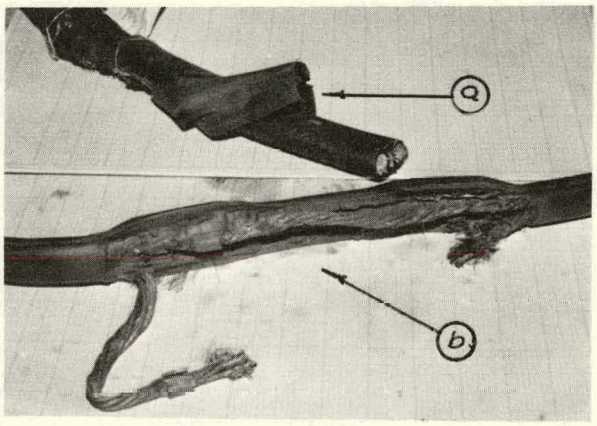


Figure 15

a. A Heat Shrink Jacket End Lifted
b. Broken Splice

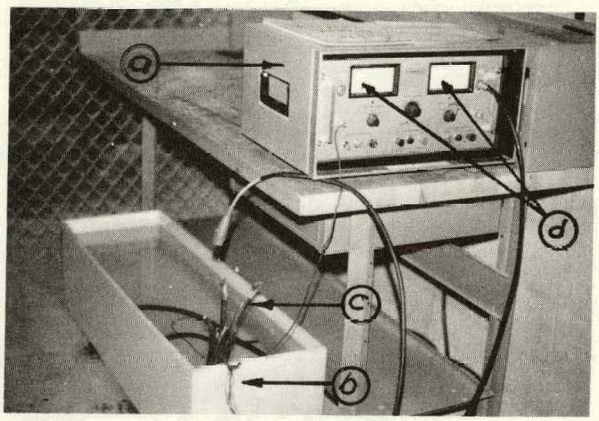


Figure 16

a. Leakage Test Equipment
b. Water Plastic Tank
c. Cable Ends Above the Water Line
d. Built-in Meters

A NEW CONCEPT IN MACHINE-MOUNTED METHANE MONITORING

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ABSTRACT

An improved, less complex Methane Monitoring System has been developed providing higher reliability, more fail-safe capability and simpler maintenance than present systems.

These improvements result from the use of a precalibrated sensing head module. This concept, a significant departure from conventional systems, is made practicable through the employment of a unique digital output signal over a three-wire line to a visual display module.

The reliability and ease of maintenance are further enhanced by making the modules of the system intrinsically safe and interconnecting them in a non-critical manner with convenient connectors.

Related topics such as battery backup, remote control signals, operation and calibration techniques are covered.

***

The functions of a Machine-Mounted Methane Monitoring System are to detect the presence of methane and give an appropriate warning of a potentially hazardous level (1.0-1.5%) for the operator to take corrective action plus an automatic shutdown signal should the level reach 2% CH_4. The system must to a high degree be fail-safe.

For about ten years, these systems have come to be standardized in basic concept.

The purpose of this paper is to offer a new system concept that is less complex yet offers greater reliability and simplified maintenance.

Let us look at a present-day system (Fig. 1).

Particularly vulnerable to problems is the detector cable system. One type of problem is inherent in the 3-wire system (Fig. 2); the power supply lines are part of the bridge and must be kept in balance. The cables must be hard-wired for stability, and use of change-out connectors is precluded. Any change in cable length (or conductor size) or contact resistance results in a signal change and recalibration is required.

A 4-wire system (Fig. 3), with the bulk of the bridge balancing resistance at the sensor head, obviates the line resistance variation problems since the detector elements are insensitive to the resulting minor bridge voltage changes. Thus, the use of change-out connectors is feasible as connector resistance variation is highly attenuated.

In any case, calibration on-site requires access to the sensor (for application of known calibration gas) plus access to the read-out box. This procedure poses a dilemma -- the use of personnel skilled in mining coal pressed into service as instrumentation technicians under a hostile environment is inefficient.

Observe Fig. 4 for a clue as to how to improve the situation. If the sensor was made independent of the read-out box and feed cable characteristics, it could be "PRE-CALIBRATED" and the use of cable connectors at each end would be feasible.

Added circuitry in the sensor package (Fig. 4) -
- regulates the supply voltage
- amplifies the sensor bridge output
- detects alarm points
- encodes the status into a digital signal fed back to the machine via the third (green) wire.

The set points are the same as before (1% and 2% CH_4) plus an additional negative-trip setting that senses a downscale signal and encodes it as a TROUBLE. The latter feature greatly expands the context of fail-safe coverage over present systems. Physically, the precalibrated sensor assembly is only slightly larger than a present MSA Sensor Assembly.

Perhaps the best feature lies in system simplification. Note: Fig. 5 - The sensor cable is now a non-critical, 3-wire cable of virtually any length and connector type as the sensor supply can vary from 4.5 VDC to 14 VDC. The signal wire, carrying a digital signal, is also independent of physical characteristics.

The digital signal is decoded electronically with a non-complex circuit in the POWER SUPPLY/MACHINE CONTROL case. Additionally, the same digital signal is visually decoded by an LED. The LED is mounted in the STATUS INDICATOR BOX which may be connected anywhere along the line or at a remote location. More than one may be incorporated. An incandescent ALARM lamp is also included. (The circuitry for it consists of a transistor and a resistor.) No problems with intrinsic safety are present as the system requires low voltage, low power and is essentially resistive.

To fully appreciate the visual decoding, refer to Fig. 6. In the NORMAL situation, the LED will flicker about once per second. This

signal is only for reference, as the enabled machine is a sufficient operator NORMAL indicator. An incandescent lamp will not respond to the short pulses. At the "WARNING" range of methane, the signal LED plus any auxiliary incandescent lamps will flash; and at 2%, all indicators will visually appear to be on continuously. Note that the latter visual signals are just the same as for present MSA systems.

TROUBLE signals, as generated by open or shorted wires or negative-trip, will either be full "ON" or "OFF".

The required automatic machine shut-down signals are derived from the decoder logic circuit. A response time delay is selected digitally without component change. Although this paper will not cover circuit details, the system will operate over an 8:1 range of basic clock frequency -- this has been shown both on a theoretical basis and with a working model.

The POWER SUPPLY/MACHINE CONTROL equipment is easily placed in current 2G housings or the chassis mounted in the machine control box. The status indicator box is very small and can be of any durable plastic or fiberglass. It should also be of a plug-in nature. Its loss or failure will not affect the automatic system features.

Remote indication may be made through the 2G housing (Fig. 5) where there is a contact for heavy tungsten lamp loads -- or, extending the status LED leads to a remote control indicator box.

Checking of system calibration must be done by the application of a known CH_4-in-air gas to the sensor and observing the proper response. Actual calibration is best done aboveground and a stock of precalibrated sensors may be stored as spare parts underground. Changing out a sensor, particularly with connectors, is a simple matter. The power cut-off and lamp circuit check is accomplished by a momentary switch on the STATUS INDICATOR box that shorts the + supply to the signal wire (Fig. 4) thus simulating a "TROUBLE" condition.

In summary, this new concept reduces the complexity of a Machine-Mounted Methane Monitoring System while improving its effectiveness.

FOOTNOTE:

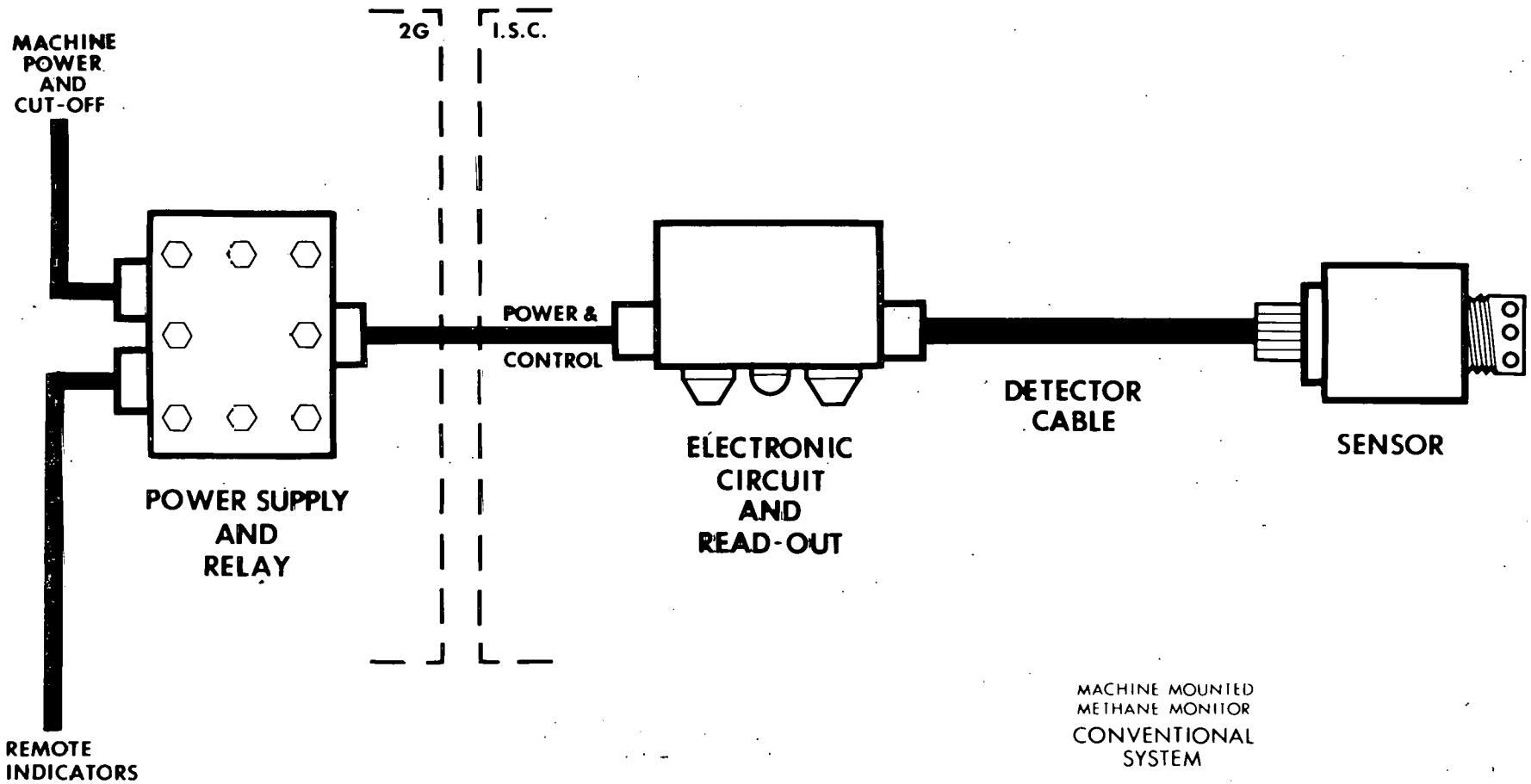
The question will be raised as to the fact that there is no meter (analog or digital) in this proposed system.

The purpose of a meter is primarily for calibration -- otherwise it is merely a vulnerable accessory. Within the context of "PRE-CALIBRATED" sensors as described, a meter is not needed.

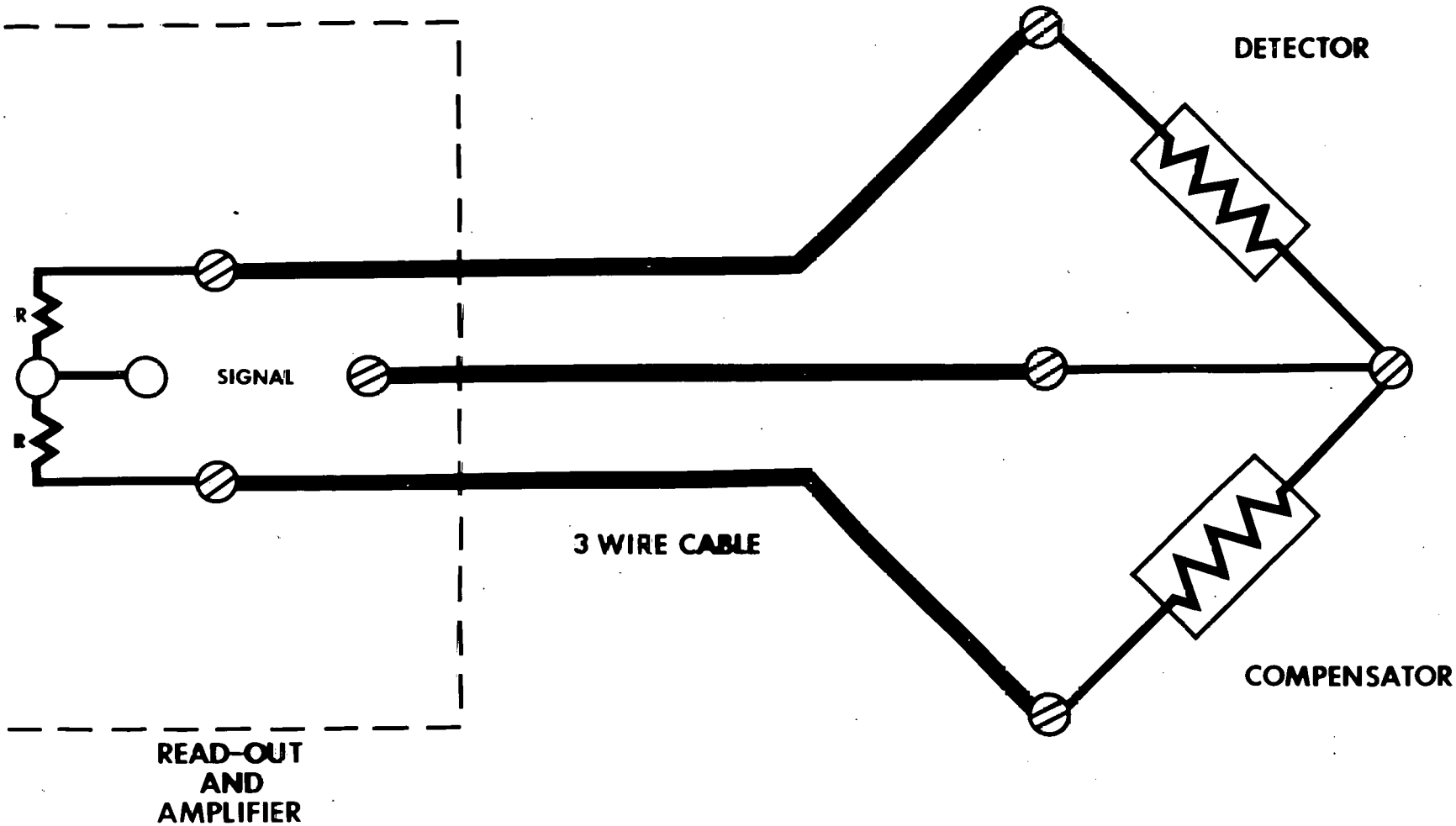
The function of a machine-mounted unit is satisfied by these automatic functions:

Range CH_4	Response
0% (downscale)	TROUBLE, SHUTDOWN
0-1% NORMAL	--
1%-2% WARNING	VISUAL
2% ALARM	VISUAL, SHUTDOWN

Accuracy and repeatability have not been compromised but enhanced as this system is reduced in complexity to a most efficient form.



MACHINE MOUNTED METHANE MONITOR
CONVENTIONAL SYSTEM



THREE-WIRE
SENSOR-BRIDGE
CONNECTION

FIGURE 2

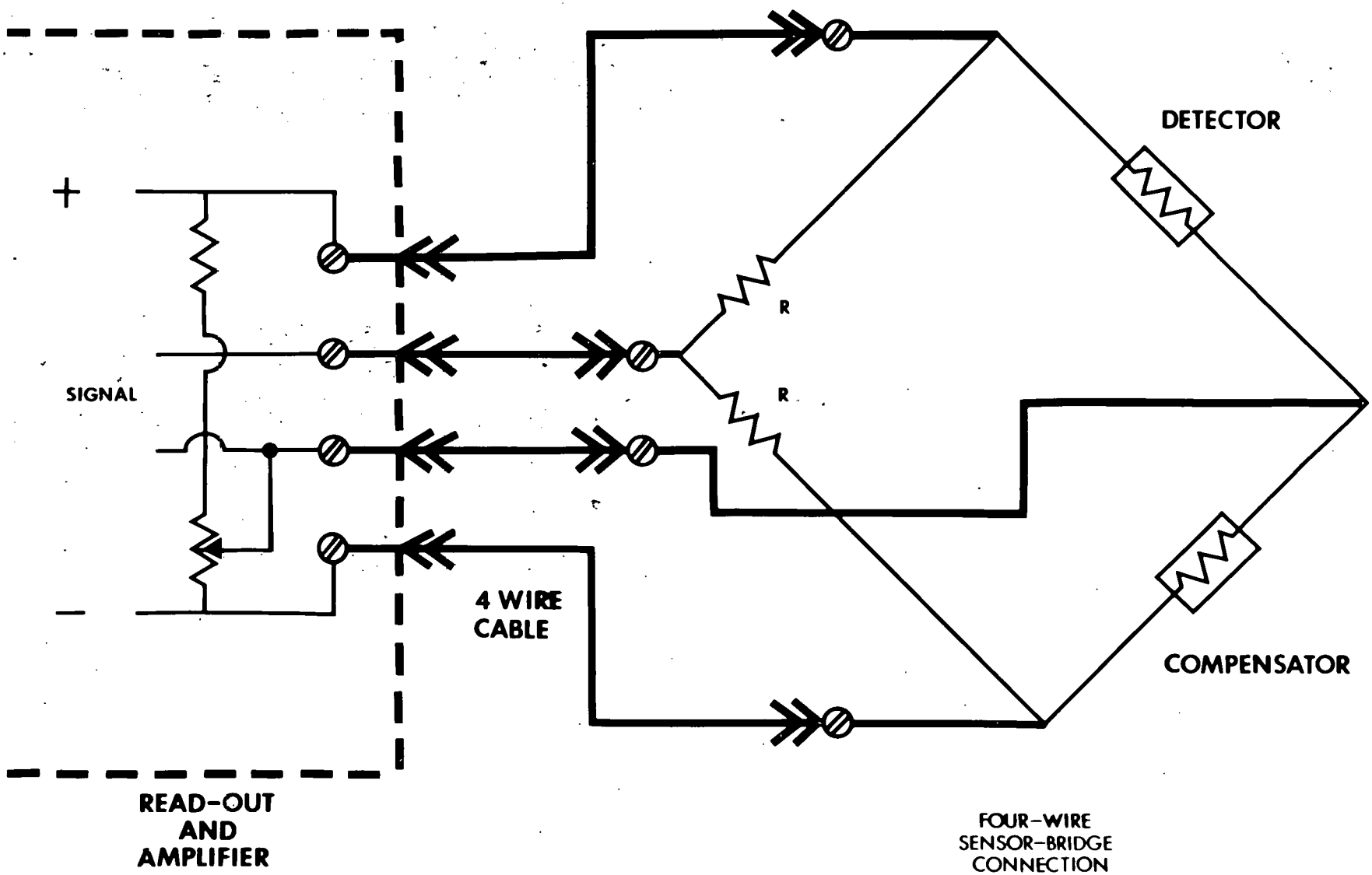
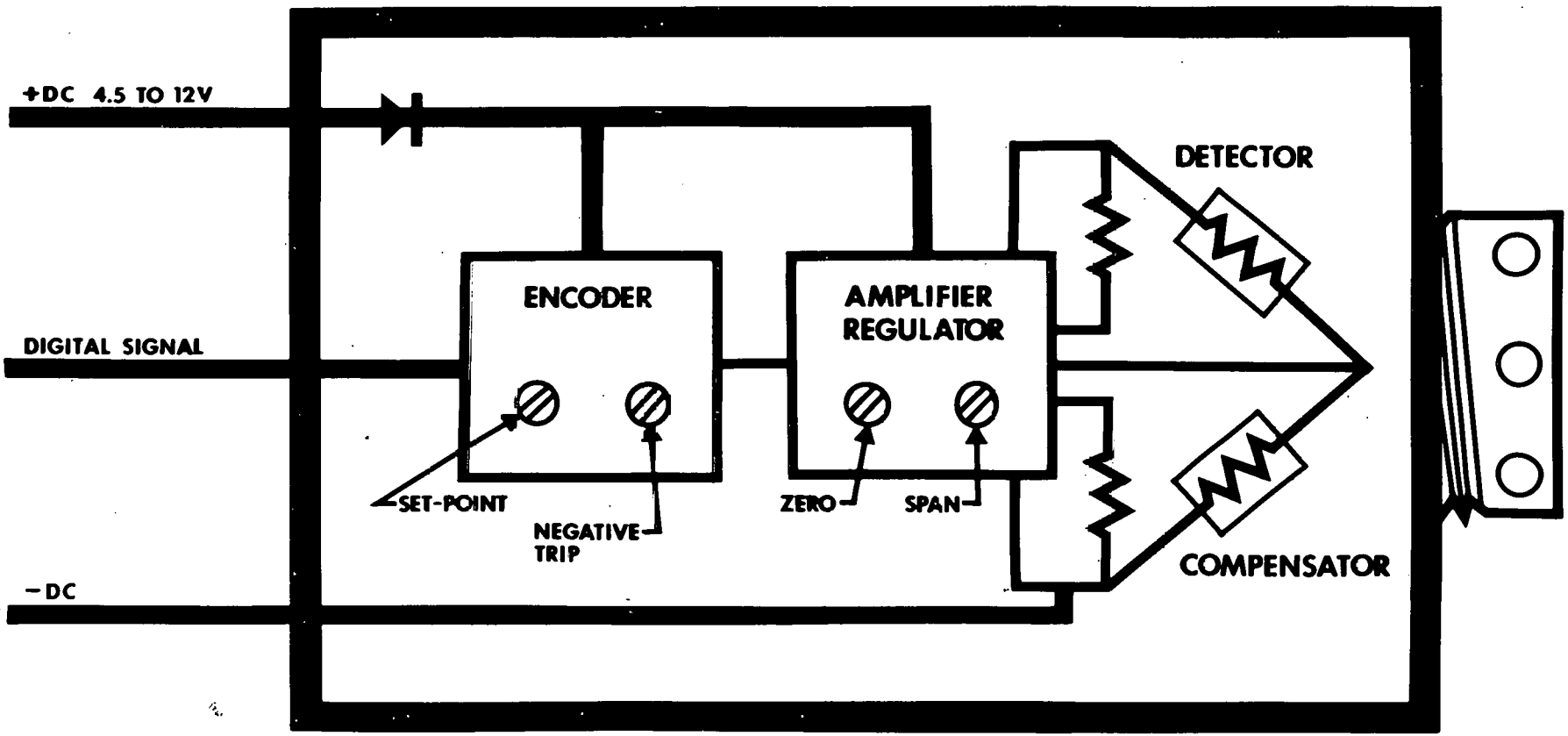

+
SIGNAL
−
READ-OUT
AND
AMPLIFIER
4 WIRE
CABLE
R
R
DETECTOR
COMPENSATOR
FOUR-WIRE
SENSOR-BRIDGE
CONNECTION

FIGURE 3



IMPROVED SENSOR

PRE-CALIBRATED SENSOR FUNCTIONAL

FIGURE 4

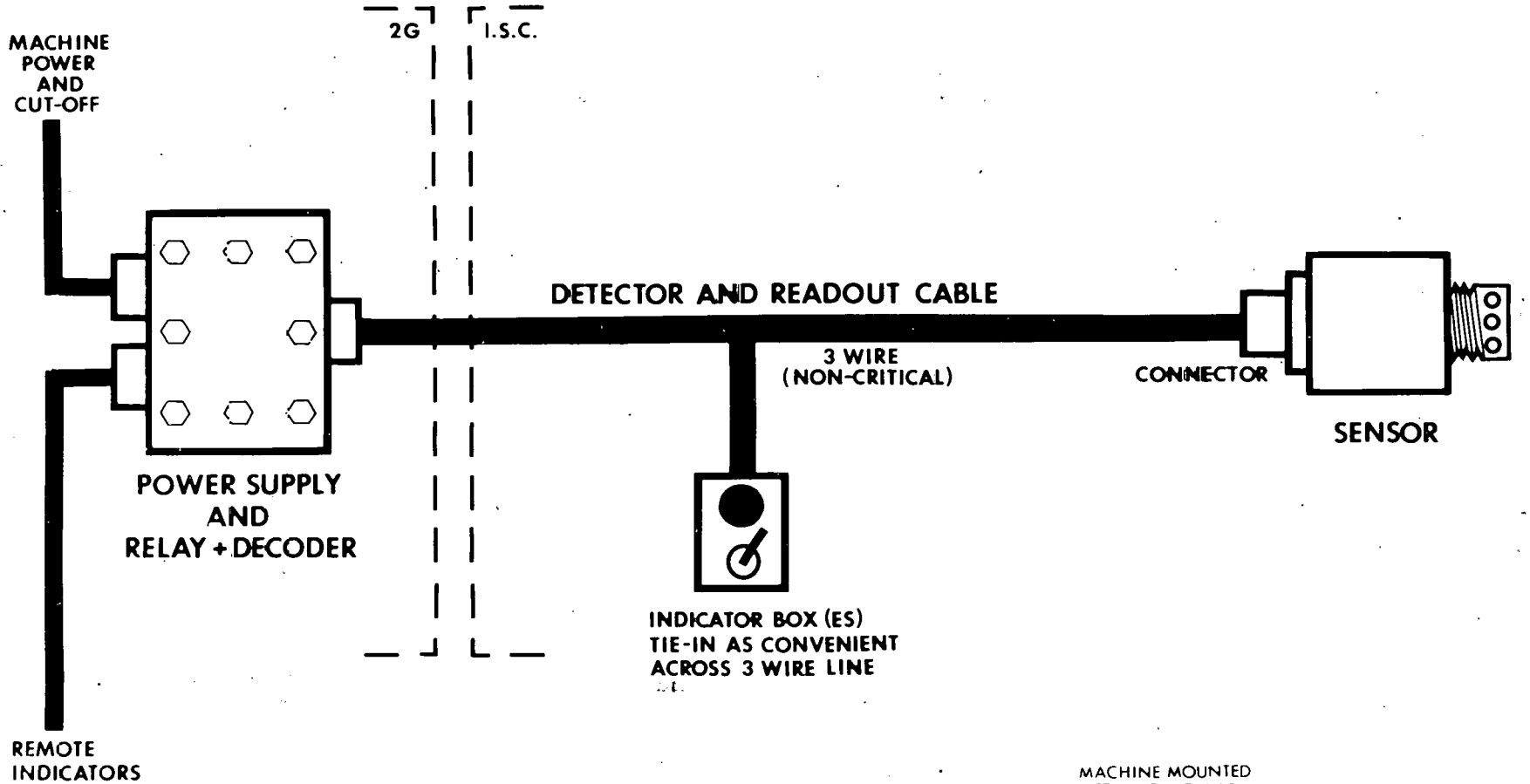
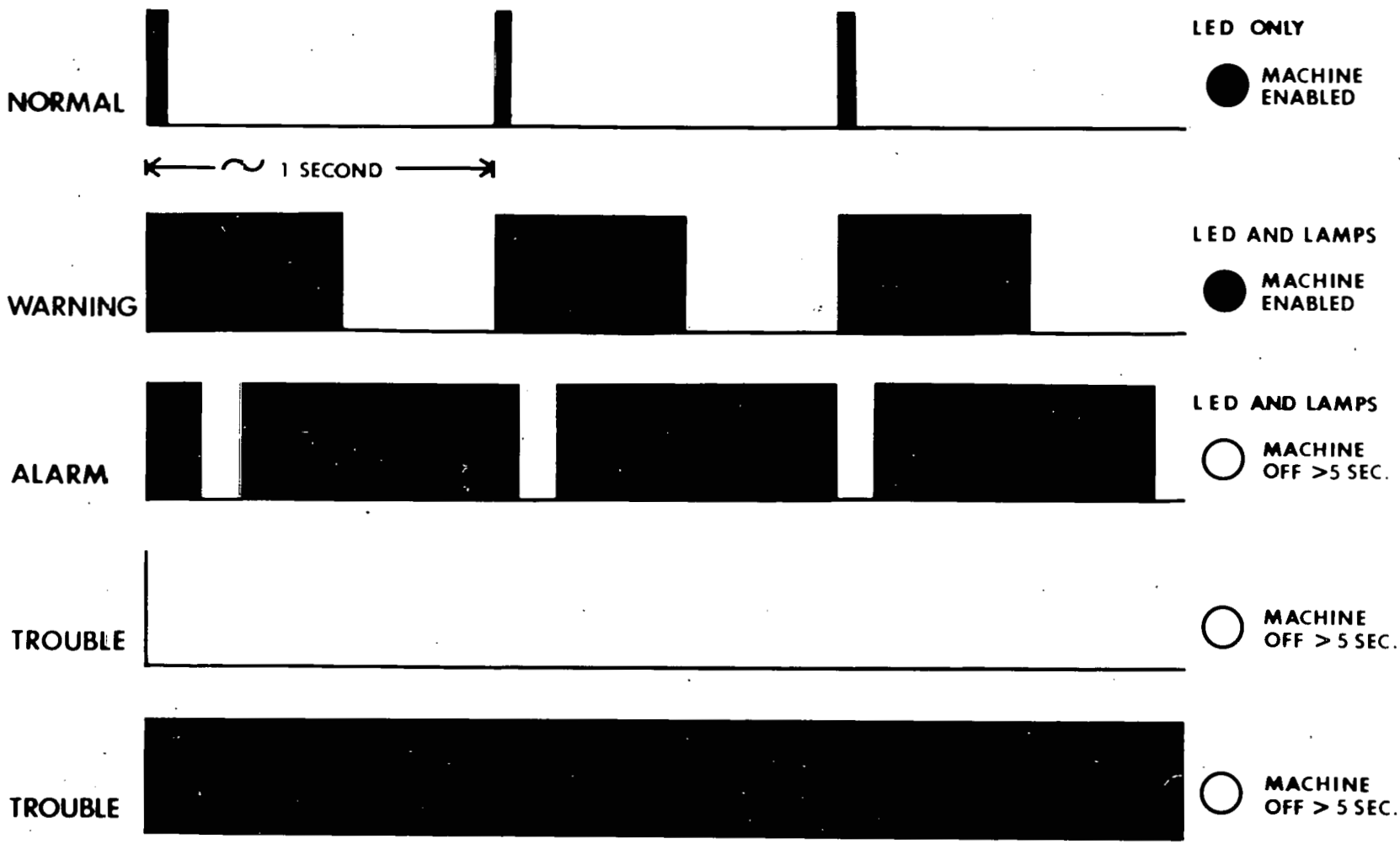

MACHINE POWER AND CUT-OFF
2G
I.S.C.
DETECTOR AND READOUT CABLE
3 WIRE (NON-CRITICAL)
CONNECTOR
SENSOR
POWER SUPPLY AND RELAY + DECODER
INDICATOR BOX (ES) TIE-IN AS CONVENIENT ACROSS 3 WIRE LINE
REMOTE INDICATORS
MACHINE MOUNTED METHANE MONITOR IMPROVED SYSTEM

FIGURE 5



IMPROVED MONITOR
VISUAL INDICATIONS

FIGURE 6

THEORY OF ELECTROMAGNETIC METHODS FOR NONDESTRUCTIVE TESTING OF WIRE ROPES*

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ABSTRACT

Past and current techniques for electromagnetic nondestructive testing of wire ropes are briefly reviewed. Recent theoretical work is also discussed. In particular, we mention prolate spheroidal void model for a broken strand or individual wire. Here we assume the wire rope is excited by an electric current loop. This primary field, in turn, induces both electric and magnetic dipole moments in the small void. The resulting external scattered field is then derived and numerical results are presented which suggest an effective configuration of sensing coils. The dual source of a magnetic current loop which is a model for a toroidal coil is also considered.

INTRODUCTION

Wire ropes are used extensively in many life sustaining situations. Elevator and mine hoist cables are two notable examples, but we might also mention the support cable for aerial tramways, ski chairlifts and gondolas, helicopter and suspension cables. In this paper, we will restrict attention to wire ropes used in mine hoists. There is an obvious need to perform tests of the integrity of such ropes without in any way impairing their function. Apart from careful visual examination and measurements of the external diameter, the nondestructive test methods available utilize electromagnetic fields, x-rays or mechanical waves. Here we consider only electromagnetic methods.

We begin with a brief review of the past and present techniques and principles of electromagnetic methods. Then we turn to some of the recent theoretical investigations starting with some idealized two-dimensional models. The three-dimensional theory of excitation of a homogeneous rope by either an encircling solenoid or toroid is then considered in more detail. The additional effect of idealized rope imperfections is treated by computing the induced electric and magnetic dipole moments. Finally, we present some numerical results for the anomalous external fields due to a rope imperfection or internal flaw. The properties of these anomalous fields are important in designing an effective configuration of sensing coils.

*This research was supported by U.S. Bureau of Mines Contract H0155008.

REVIEW OF PAST AND CURRENT TECHNIQUES

Since a thorough review of electromagnetic methods containing numerous references has recently been prepared [1], we present here only a relatively brief summary. In all past and current systems, the rope is excited with an axial magnetic field which can be either A.C. or D.C. The operating frequency for A.C. systems is generally somewhere between 10 Hz and 100 Hz. The search coil is typically a loop which completely encircles the rope.

Wall [2] gives an early account of the design and operation of electromagnetic rope testers. He is an early proponent of the A.C. method as opposed to D.C. magnetic testing favored in more recent times. He lists some of the advantages as follows: a) the rope need not continuously move through the magnetic system, b) the A.C. signal is easy to detect and c) any remnant magnetism due, say, to a previous test is wiped out. Wall was interested in testing colliery ropes of the locked coil type up to 2¼ inch (5.6 cm) diameter. He was able to ascertain that the flux penetration was almost complete at 20 Hz but only partially so at 50 Hz.

Semmelink [3] gives an extremely interesting and comprehensive account of the early history of electromagnetic testing of wire ropes in South Africa. He also tested about fifty hoist ropes with an A.C. device at 50 Hz. In his measurements on steel, Semmelink found, what is now generally known, that an increase of tensile stress causes a significant increase in magnetic permeability but not in D.C. resistance. Lang [4] gives an extensive description of a very similar A.C. device, and most of his data are shown for a frequency of 30 Hz.

Hilfbrunner [5] gives a good description of what is known now as the D.C. magnetic testing method. The method is based on imposing a strong axial magnetic field to the sample by either a solenoid coil or a permanent magnetic. The search coil is oriented in the radial direction and produces a voltage response only when there is some type of lateral non-uniformity. Stachurski [6] gives a very useful summary of the physical concepts employed in D.C. magnetic testing for flaws in wire ropes. A number of interpretative schemes are outlined in a qualitative manner, and he uses prescribed forms of flaws, breaks, and cracks to calibrate the device. Using three D.C. devices, Egen and Benson [7] performed some very useful tests on a special prepared rope with seven types of faults.

Hochschild [8] has reviewed the theoretical papers by Förster [9-12]. Wait [1] points out that electromagnetic methods of non-destructive testing of solid conductors are closely akin to techniques now used in geophysical prospecting [13]. Most of the theoretical papers on wire rope testing are carried out for the special case of azimuthal symmetry [14-16]. Burrows [17] treats asymmetrical "flaws", but assumes azimuthal symmetry of the sensing coils.

TWO-DIMENSIONAL ANALYSIS

Wait [18] has recently derived the internal and external fields of an infinite circular cylinder which is excited by either an encircling solenoid or toroid of infinite length. The solenoid excites the rope with an axial magnetic field and an azimuthal electric field as do the current rope testers. The toroid, which is the dual source, excites the rope with an axial electric field and an azimuthal magnetic field. Such a source would be useful in probing the axial conductivity of the rope. The impedance of the encircling solenoid or toroid is also derived and is shown to be independent of the rope position under the usual quasi-static conditions.

Wait and Gardner [19] have extended the two-dimensional model to allow for the presence of a small cylindrical flaw of arbitrary conductivity and permeability. They derive an expression for the scattered magnetic and electric fields and the resultant change in the impedance of the encircling solenoid. They also present numerical results for various anomaly parameters at frequencies from 1 Hz to 1000 Hz.

Past theoretical work has assumed that the conductivity and permeability are isotropic, but Wait [20] has recently presented the theory for excitation of a wire rope where the conductivity and permeability are uni-axial tensors. This anisotropic model should be more representative of stranded wire ropes.

SOLENOID AND TOROID EXCITATION

In order to model excitation of a wire rope by a finite source, we have analyzed the configuration in Fig. 1. The source is either an electric or magnetic current loop which encircles an infinite wire rope. The first step is to determine the internal and external fields for the homogeneous rope in the absence of the void.

An electric current loop is an idealized model of a thin solenoid [21]. Due to the azimuthal symmetry of the rope and the source, the primary fields are TE (Transverse Electric) and independent of ϕ. The electric field has only an azimuthal component E_ϕ^{pr}, and the magnetic field has both an axial component H_z^{pr} and a radial component H_ρ^{pr}. The explicit integral forms of the fields have been derived and evaluated previously [21]. Actually, E_ϕ^{pr} and H_ρ^{pr} are zero at the center of the rope ($\rho=0$). Also, H_z^{pr} is nearly independent of ρ inside the rope for sufficiently low frequencies.

A magnetic current loop is an idealized model for a thin toroidal coil [22]. This source is the dual of the electric current loop, and the primary fields are TM (Transverse Magnetic) and independent of ϕ. The magnetic field has only an azimuthal component H_ϕ^{pr}, and the electric field has both an axial component E_z^{pr} and a radial component E_ρ^{pr}. The explicit integral forms have been derived and evaluated previously [22]. Both H_ϕ^{pr} and E_ρ^{pr} are zero at the center of the rope ($\rho=0$), but E_z^{pr} is nearly independent of ρ inside the rope for sufficiently low frequencies. The additional source complication of finite width in the axial direction and arbitrary azimuthal extent has also been considered [23]. This

more general source results in more complicated fields without azimuthal symmetry.

ANOMALOUS FIELDS OF ROPE IMPERFECTIONS

In order to model an individual wire or broken strand within the rope, we select a thin prolate spheroid of conductivity σ_v and permeability μ_v. The prolate spheroidal shape is a convenient one because its electric and magnetic polarizabilities are known. However, we would not expect a significant difference for a thin cylinder of the same length and volume. To account for the winding geometry of the rope, we allow a rotation of the major axis of the spheroid about the ρ' axis by an angle α. Thus the major axis is oriented at an angle α to the unit vector $\hat{z}'$ and an angle $\pi/2 + \alpha$ to the unit vector $\hat{\phi}'$ as indicated in Fig. 1c.

Since the void has a contrast in both the electric and magnetic properties, both electric and magnetic dipole moments will be induced [24]. The electric polarizabilities for the incident electric field applied along the major axis, α^e_{maj}, or along the minor axis, α^e_{min}, are given by

$$\alpha^e_{maj} = - V(\sigma_w - \sigma_v) \tag{1}$$

and

$$\alpha^e_{min} = - 2V(\sigma_w - \sigma_v)/(1 + \sigma_v/\sigma_w) , \tag{2}$$

where V is the volume of the thin prolate spheroid and σ_w is the rope conductivity. Because of the low operating frequency, we have neglected displacement currents. To include them, σ_w would be replaced by $\sigma_w + i\omega\varepsilon_w$ and σ_v would be replaced by $\sigma_v + i\omega\varepsilon_v$ in (1) and (2) for $\exp(i\omega t)$ time dependence. The magnetic polarizabilities for the incident magnetic field applied along the major axis, α^m_{maj}, or along the minor axis, α^m_{min}, are similarly given by

$$\alpha^m_{maj} = - Vi\omega(\mu_w - \mu_v) \tag{3}$$

and

$$\alpha^m_{min} = -2Vi\omega(\mu_w - \mu_v)/1 + \mu_v/\mu_w) , \tag{4}$$

where μ_w is the rope permittivity.

In order to compute the induced dipole moments, it is first necessary to resolve the incident electric and magnetic field into components along the major and minor axes. The resultant induced dipole moments can then be resolved into the more convenient ρ, ϕ, and z components. When this is done, the induced electric and magnetic dipole moments for an electric loop source are [21]

$$(Ids)_z = E^{pr}_\phi(\alpha^e_{min} - \alpha^e_{maj})\sin\alpha\cos\alpha, \tag{5}$$

$$(Ids)_\phi = E^{pr}_\phi(\alpha^e_{min}\cos^2\alpha - \alpha^e_{maj}\sin^2\alpha) , \tag{6}$$

$$(Ids)_\rho = 0 . \tag{7}$$

Similarly the induced magnetic dipole moments are found to be

$$(Kd\ell)_z = H_z^{pr}(\alpha_{maj}^m \cos^2\alpha + \alpha_{min}^m \sin^2\alpha) \ , \tag{8}$$

$$(Kd\ell)_\phi = H_z^{pr}(\alpha_{min}^m - \alpha_{maj}^m)\sin\alpha\cos\alpha \ , \tag{9}$$

$$(Kd\ell)_\rho = H_\rho^{pr}\alpha_{min}^m \tag{10}$$

In (5)-(10), the primary field components E_ϕ^{pr}, H_z^{pr}, H_ρ^{pr} are evaluated at ρ', ϕ', z'. For the magnetic current loop source, the induced electric and magnetic dipole moments are given by similar expressions [22]

$$(Ids)_z = E_z^{pr}(\alpha_{maj}^e \cos^2\alpha + \alpha_{min}^e \sin^2\alpha) \ , \tag{11}$$

$$(Ids)_\phi = E_z^{pr}(\alpha_{min}^e - \alpha_{maj}^e)\sin\alpha\cos\alpha \ , \tag{12}$$

$$(Ids)_\rho = E_\rho^{pr}\alpha_{min}^e \ , \tag{13}$$

$$(Kd\ell)_z = H_\phi^{pr}(\alpha_{min}^m - \alpha_{maj}^m)\sin\alpha\cos\alpha \ , \tag{14}$$

$$(Kd\ell)_\phi = H_\phi^{pr}(\alpha_{min}^m \cos^2\alpha - \alpha_{maj}^m \sin^2\alpha) \ , \tag{15}$$

$$(Kd\ell)_\rho = 0. \tag{16}$$

The scattered field external to the rope is the observable quantity in any nondestructive testing system. Bearing in mind the use of small coils for sensors, we will consider primarily the scattered magnetic field components $H_\rho^{sc}(\rho,\phi,z)$, $H_\phi^{sc}(\rho,\phi,z)$, and $H_z^{sc}(\rho,\phi,z)$. Each of these components can be written as a superposition of the contributions from each of the six dipole sources in the following compact matrix notation

$$\begin{bmatrix} H_\rho^{sc} \\ H_\phi^{sc} \\ H_z^{sc} \end{bmatrix} = \begin{bmatrix} G_{\rho\rho}^e & G_{\rho\phi}^e & G_{\rho z}^e \\ G_{\phi\rho}^e & G_{\phi\phi}^e & G_{\phi z}^e \\ G_{z\rho}^e & G_{z\phi}^e & G_{zz}^e \end{bmatrix} \begin{bmatrix} (Ids)_\rho \\ (Ids)_\phi \\ (Ids)_z \end{bmatrix} + \begin{bmatrix} G_{\rho\rho}^m & G_{\rho\phi}^m & G_{\rho z}^m \\ G_{\phi\rho}^m & G_{\phi\phi}^m & G_{\phi z}^m \\ G_{z\rho}^m & G_{z\phi}^m & G_{zz}^m \end{bmatrix} \begin{bmatrix} (Kd\ell)_\rho \\ (Kd\ell)_\phi \\ (Kd\ell)_z \end{bmatrix} \tag{17}$$

In the first 3 by 3 matrix, $G_{\rho\rho}^e$ is the ρ component of H at (ρ,ϕ,z) produced by a unit ρ-directed electric dipole at (ρ',ϕ',z'), $G_{\rho\phi}^e$ is the ρ component of H produced by a unit ϕ-directed electric dipole, etc. The second 3 by 3 matrix is similar except that the G^m elements correspond to unit magnetic dipoles. The specific expressions for the matrix elements have been derived previously [22]. A similar matrix expression is available for the scattered electric field [21].

The matrix expression in (17) is actually a general statement of the scattered fields produced by any small flaw in the rope. For the specific case of the thin prolate spheroidal flaw considered here, the induced dipole moments are given by (5)-(10) for electric current loop excitation or by (11)-(16) for magnetic current loop excitation. For other flaws or excitations, only the induced dipole moments will be different.

Computer programs have been written to obtain the scattered and primary fields for both electric [21] and magnetic [22] current loop excitation. Extensive numerical results in the form of computer plots have been generated for the electric current loop source [21] because this case is most representative of current rope testers. In Figs. 2-4, we present three representative plots of the azimuthal variation of the three scattered magnetic field components (H_ρ^{sc}, H_ϕ^{sc}, H_z^{sc}) normalized to the primary axial magnetic field H_z^{pr}. The parameters are: a = 1 cm, $\sigma_w = 1.1 \times 10^6$ mho/m, $\mu_w/\mu_o = 200$, $\sigma_v = 0$, $\mu_v/\mu_o = 1$, $\alpha = 0°$, $\phi' = 0$, $z'/a = 5$, $z/a = 5$, $\rho/a = 2$, and f = 10 Hz. The volume V of the void is taken to be $10^{-8}m^3$(= 1 cm × 1 mm × 1 mm), but the results are easily normalized to any other volume since the scattered fields are directly proportional to V.

Fig. 2 demonstrates that, for the void located near the center of the rope ($\rho'/a = 0.1$), the azimuthal variation of the scattered field is small. However, in Fig. 4, a large azimuthal variation is evident for the void located near the rope boundary ($\rho'/a = 0.9$). Most AC rope testers sense only the average azimuthal scattered electric field with a solenoid sensor which completely encircles the rope. In Fig. 5, the normalized scattered electric field $|E_\phi^{sc}/E_\phi^{pr}|$ is shown for the same parameters. It is again evident that a significant amount of information (and signal strength) is lost by sensing only the ϕ-averaged electric field.

CONCLUDING REMARKS

We have begun with a brief review of electromagnetic methods of wire rope testing. As indicated, the very practical and perhaps most useful work to date has been empirical in nature. There has been no attempt to deduce the magnitude of the secondary fields due to internal flaws; instead a large number of ropes have been tested and the results have been presented.

On the other hand, the analytical approaches have considered only highly idealized situations. Here we have analyzed a more realistic three-dimensional configuration without azimuthal symmetry. We have computed the three-dimensional scattered fields for a thin prolate spheroidal void which represents a broken strand. The results indicate that an effective configuration of sensing coils should make use of large asymmetrical portions of the scattered fields. Possible extensions of this work would include other types of small flaws which could be treated in a similar manner.

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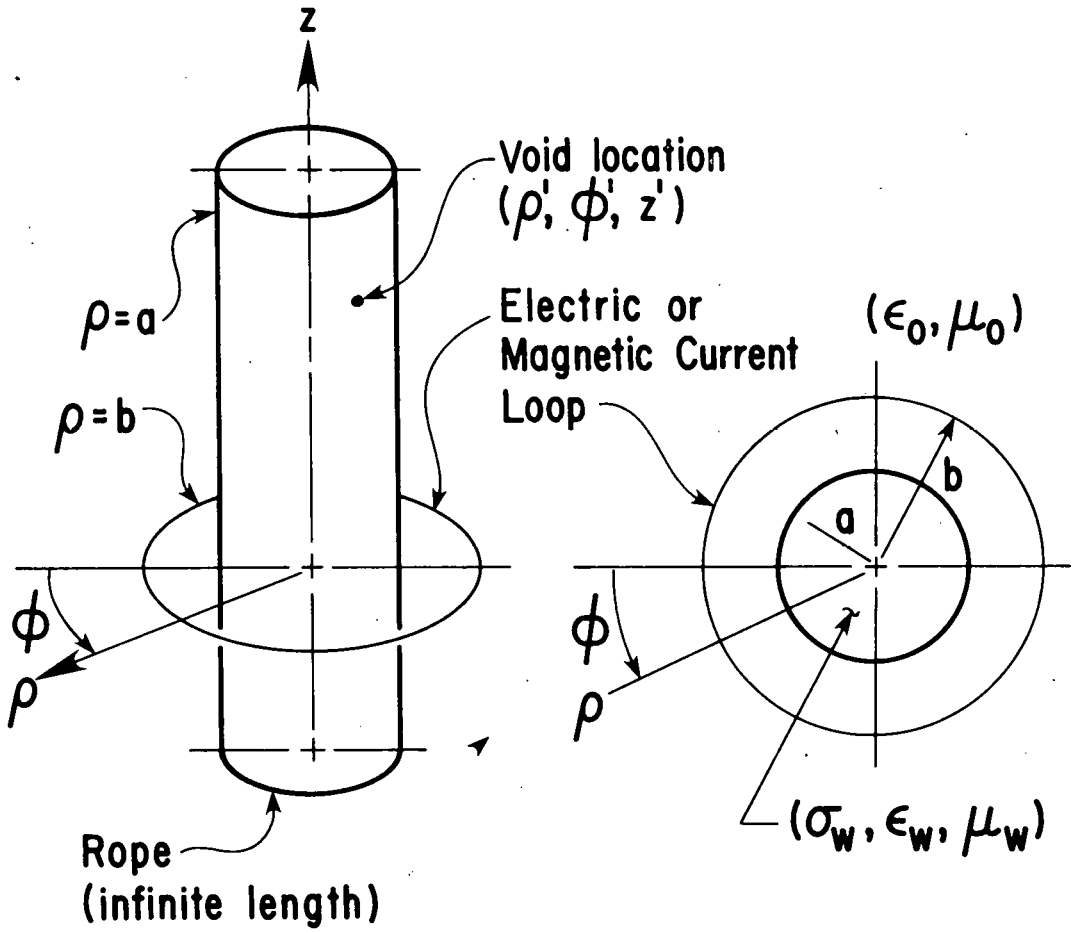


Fig. 1a

Fig. 1b

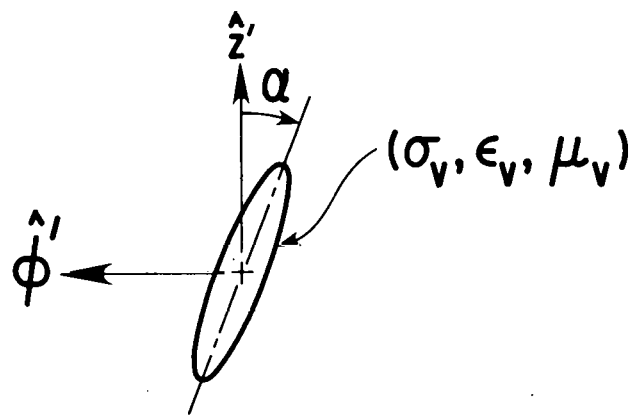


Fig. 1c

Fig. 1. Electric or magnetic current loop surrounding a metal rope of infinite length. a) Perspective view, b) Top view, c) Prolate spheroidal void.

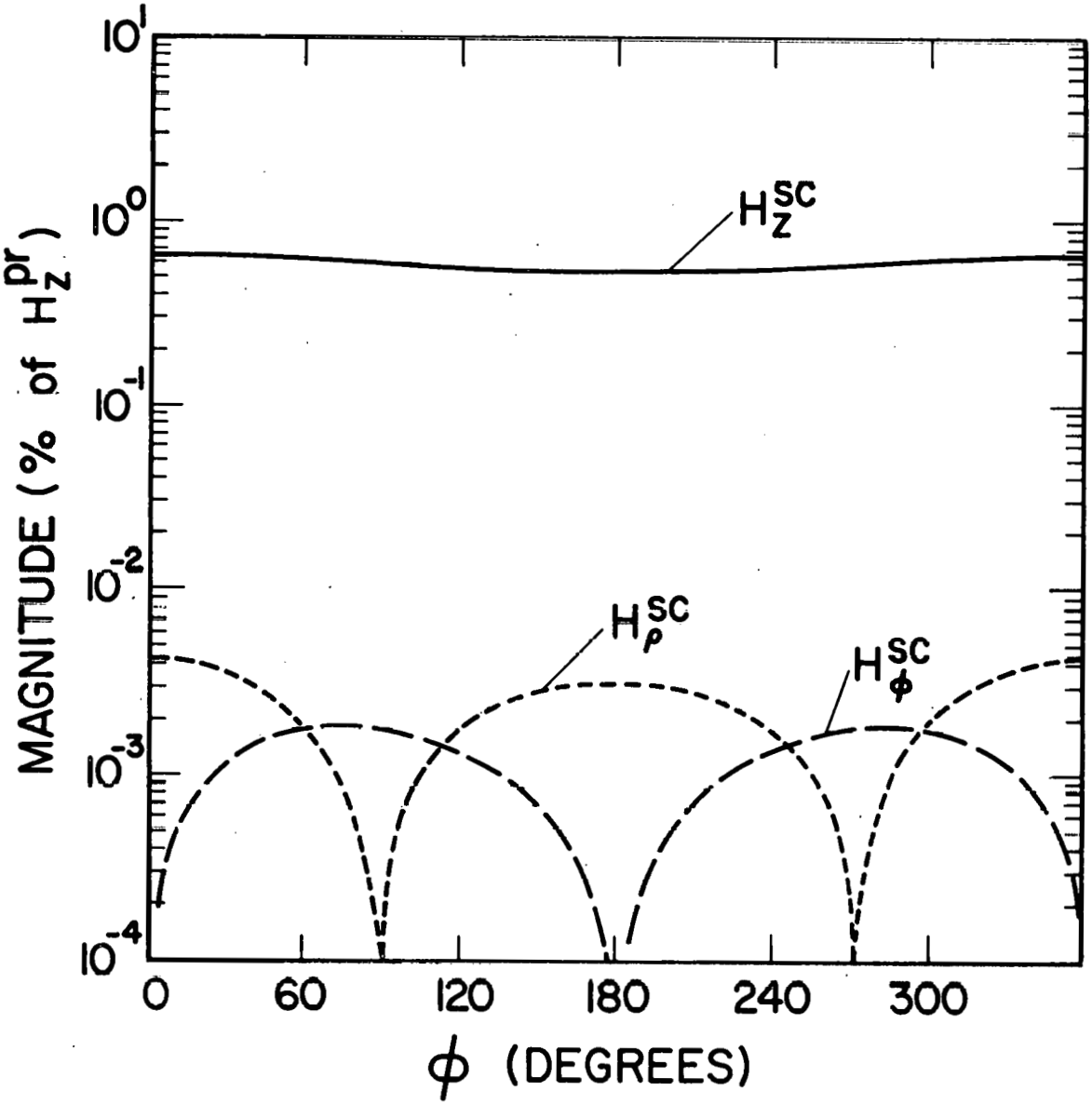


Fig. 2. Normalized magnitude of the scattered magnetic field components for $\rho'/a = 0.1$.

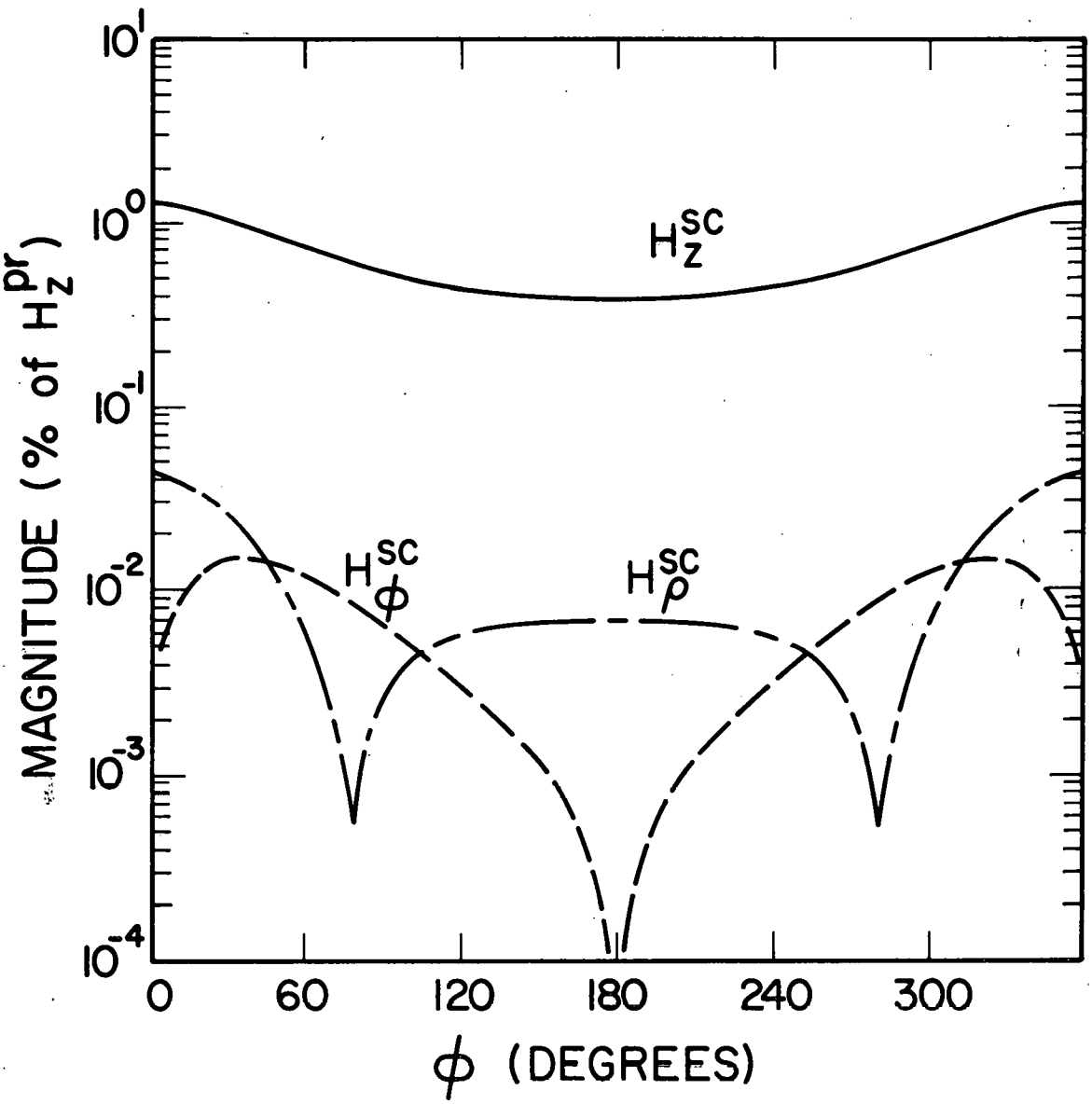


Fig. 3. Normalized magnitude of the scattered magnetic field components for $\rho'/a = 0.5$.

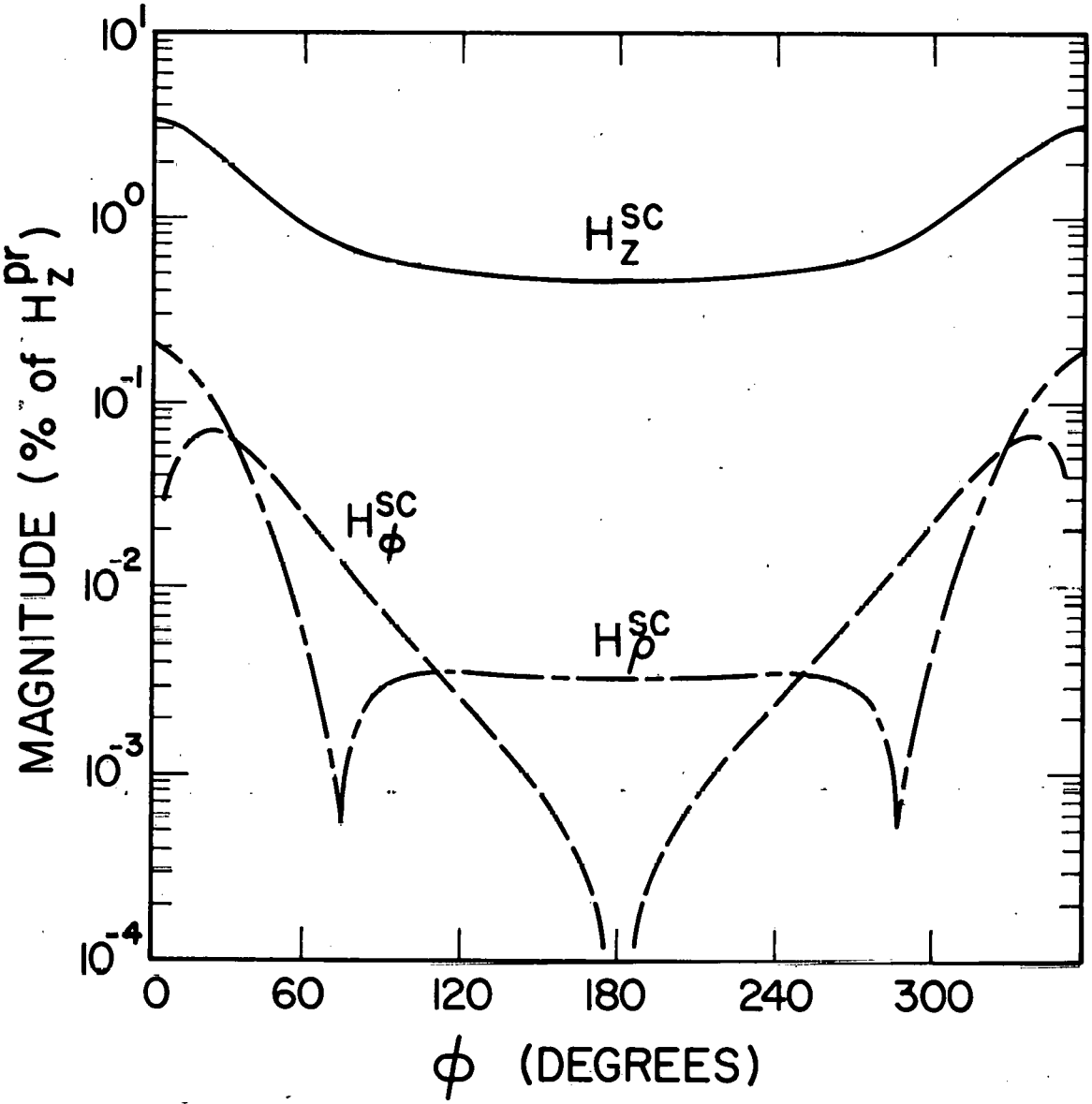


Fig. 4. Normalized magnitude of the scattered magnetic field components for $\rho'/a = 0.9$.

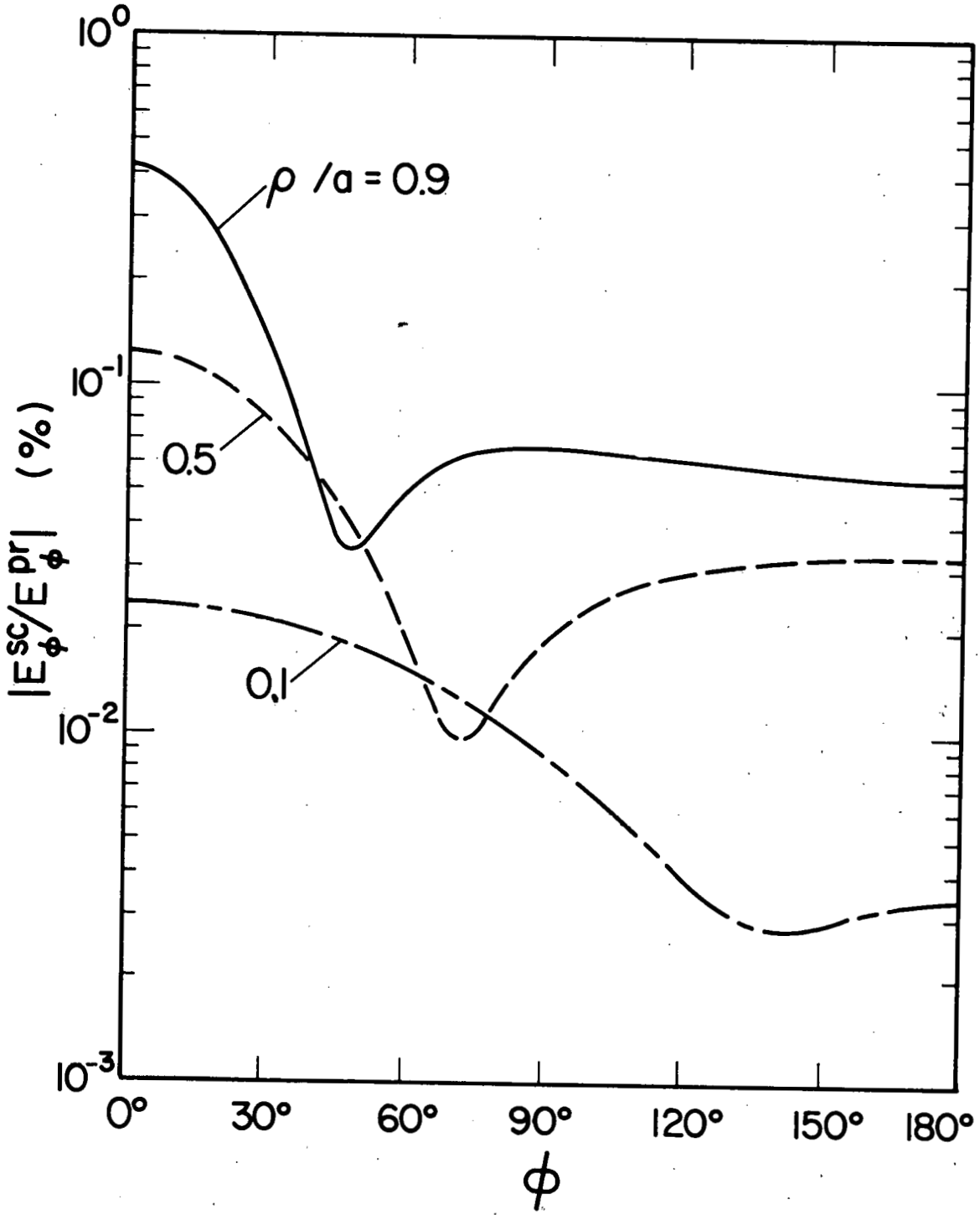


Fig. 5. Normalized magnitude of the scattered azimuthal electric field.

NEW INSTRUMENTS FOR VENTILATION MONITORING IN THE WEST GERMAN COAL MINES

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Federal Republic of Germany

ABSTRACT

To secure the health and safety of miners and to maintain high productivity requires close monitoring of underground ventilation. This paper will review some newly developed instruments for two main fields of air monitoring: air quantity and climatic measurements as well as gas analysis. Whereas the development of the new intrinsically safe thermoelectric Pitot tube "MINI-VENTOR" has been successfully completed, the evolution of an instrument for climatic values is still underway. In the field of gas analysis, one of the principal developments is the reduction of power demand of non-dispersive infrared gas analyzers which led to intrinsically safe portable as well as stationary remotely fed devices "MINI-UNOR" for the determination of CO, CH_4 and CO_2. Such systems also are incorporated in a compact station for analyzing fire gases. Another new instrument is a probe for simultaneous measurement of methane contents and gas flow velocities in methane drainage pipelines. Practical experience with these instruments will be reported.

INTRODUCTION

The penetration of mining into greater depth and the increasing concentration and mechanization of coal mining raise problems with regard to mine ventilation in the West German coal mining industry. Thus, a depth of more than 1 000 m has been reached in more than half of the Ruhr collieries and the deepest haulage level in two collieries is situated deeper than 1 300 m. In such depth, rock temperatures up to 58° C prevail. The mean daily disposable coal output per working point doubled within the last ten years to more than 1 200 tons, accompanied by a strong increase of the emission of heat and of methane. The progressing mechanization is reflected in the installed electrical machine capacity which increased from 103 kW per 100 tons of disposable output in 1961 to 244 kW per 100 tons of disposable output in 1977. This increase, too, caused a considerable increase of the heat development.

These mining-technical developments require a comprehensive monitoring of mine air in order to secure the health and the safety of the miners and to maintain coal-outputs. This ventilation monitoring covers the following subjects:

- continuous monitoring that sufficient fresh air is available to the miners and to dilute noxious gases,
- to secure human working conditions with regard to mine climate,
- to detect toxic and explosive gases,
- to detect early, by indicator gases, hazards such as mine fires, etc.

Therefore, two important duties of air measurements are:

1. measurements of air quantities and mine climate,
2. gas analysis.

In this paper, several new measuring instruments for these two purposes, developed by the Bergbau-Forschung GmbH of Essen-Kray, will be described.

The measuring and control instruments, too, are exposed to the hazards and environmental conditions which they are to detect and to monitor. Thus, the methane contents of the mine air require the instruments to be flameproof. Electrical devices in mine workings endangered by firedamp must be switched off normally when the mine air exceed 1 % methane or, in exceptional cases, 1,5 %. This means, that ventilation monitoring instruments will have to be switched off, too, when these percentages are exceeded, just in the moment, when they are urgently required for measuring the further development. However, intrinsically safe instruments and devices with their own power supply need not to be switched off, so that one of the main requirements on ventilation monitoring devices is a flameproof protection of the type "Intrinsic Safety". Besides, intrinsically safe devices have a much lower weight than flameproof installations, which is indispensable for portable instruments.

MEASUREMENTS OF AIR QUANTITIES AND MINE CLIMATE

The release of heat and methane much increased by the above mentioned technical developments required more efforts in the monitoring of air quantities and the mine climate.

Measurements of air quantities can be based generally on measurements of the air velocity. A thermo-electrical anemometer, type VENTOR, specially developed for

the needs of the coal mines, has proved well for these duties in the West German coal mining industry. This air velocity meter is manufactured by Maihak AG, Hamburg, and consists of a measuring probe in the form of a Pitot tube and a receiver unit (Fig. 1). Figure 2 shows the inner design of the measuring probe. A pressure difference is developed at the two openings of the probe which causes a transverse flow through the measuring head. This transverse flow is measured by a pair of heating coils which is heated up to about 100° C. A special filter consisting of several layers of small adhesive coated glass spheres, the diameter of which decreases in the direction of flow, serves the dust extraction from the flow which, with about 10 cm^3/min, is very small. This filter is most effective for fine grained dust and has a very low flow resistance. Thus, the measuring probe is well protected against dust, without any influence on its sensitivity, as its use in many years and in many collieries showed.

A further development of this VENTOR, a portable MINI-VENTOR equipped with an intrinsically safe electronic system, serves special mobile measurements, particularly in case of low air velocities or investigations of the ventilation cross-sections. As can be seen from Fig. 3, this MINI-VENTOR comprises a measuring probe working in the same way as that of the VENTOR, and a power supply and an indicating instrument. The basic range of measurement is between 0,1 and 0,5 m/s. A combined pneumatic and electrical conversion system, however, permits measurments over the ranges 0,4 to 2,0 m/s and 1 to 5 m/s, too. The measuring signal is raised by a preamplifier still within the probe to a standard voltage such as to render an exchange of measuring probes more easy. The measuring probe weighs less than 2 kg. The power supply part includes a rechargeable battery for a service life of 10 hours and an linearising amplifier.

The VENTOR, equipped with an intrinsically safe measuring circuit and a standard signal output can be applied, too, as a measuring head in larger monitoring stations. At present, the ultrasonic vortex instruments already successfully used in USA, are tested in our mines as air velocity sensors for such monitoring plants and special quite robust hot-wire anemometers are tested for smaller hand measuring instruments. Compared with the Pitot tubes and the vortex instruments, these hot-wire air velocity meters have the advantage of a high sensitivity in the lower velocity range down to 0,02 m/s.

Because of its small size and its high sensitivity especially at low air velocities, such a hot-wire anemometer will be used in a new climatic measuring instrument being in the stage of development. By "mine climate" we understand the relationship between temperature, air

humidity and air velocity according to the Yaglou-diagramm of the socalled "basic effective temperature" shown in Fig. 4. This effective temperature decreases with increasing air velocity and increases with an increasing air humidity. The regulation about Mine Climatic Values of the Chief Mines Inspectorate of the Land Nordrhein-Westfalen applicable since more than 1 year for the Ruhr coal mines provides that no workers must be employed anymore at an effective temperature of more than 32 °C on the working point and that the working time must not exceed 5 hours at effective temperatures of more than 29 °C and must not exceed 6 hours at dry-bulb temperatures above 28 °C.

As the present measurements of climatic values are quite costly and time-consuming, there is a need for a compact hand-held instrument measuring the three factors dry-bulb temperature, air humidity and air velocity and connecting these three factors to the effective temperature. Such a measuring device must fulfil, inter alia, the following requirements:

- high precision as the measured results are used for decisions concerning directly the health of the miners and the productivity of the working point,
- small size and intrinsic safety which furtheron requires low energy demand of the measuring sensors and the electronic system used for analysis,
- low maintenance and easy calibration.

It is difficult to find the right sensors which fully comply with the above requirements. Thus, many sensors have a high precision, but also a high energy demand with the result, that a compromise must be found. The use of platinum resistance thermometers is possible to measure the dry-bulb temperature. As for the measuring of the wet-bulb temperature we selected, after thorough investigations with many humidity measuring methods, a thin-layer capacity sensor of the Finnish firm Vaisala. This sensor is distinguished by a high precision, a linear output signal, a negligible dependance on temperature and a low energy demand. For air velocity measurement, the afore mentioned hot-wire anemometer is to be used heated up to about 30 °C above the ambient temperature. A microprocessor method is being developed for linking up the measured values to the effective temperature.

GAS ANALYSIS

Non-dispersive infrared UNOR instruments have been used so far nearly exclusively in the West German coal mines to detect noxious or dangerous gases.

Carbon Monoxide

Carbon monoxide is thought to be the best indicator for

the early detection of slowly developing mine fires due to spontaneous combustion. The CO content is measured nearly exclusively by the flameproof UNOR 1 instruments shown in Fig. 5. Generally these UNOR 1 have a measuring range of 300 ppm CO, some of only 100 ppm CO. They distinguish themselves by a high stability and a low cross sensivity against disturbing gases such as CH_4, NO_x, H_2 and H_2O. There are also no problems with regard to the service life. A disadvantage of the UNOR 1 instruments is their relatively high sensitivity against shocks and vibrations with the result, that no more sensitive measuring ranges can be realized. Other drawbacks are the high price and the high weight.

The further development of the method of measuring CO aiming at an improved and more rapid early detection of mine fires pursues the following three objectives:

1. Improvement in the application of the UNOR 1. The high number of the UNOR 1 used underground requires, in order to reduce maintenance work, an automatic check of the zero point and sensitivity from the pit control center at the surface. At present, this is done by different more or less comfortable systems at the mines. With the first device, zero point and sensitivity can be checked at any time when demanded by the pit control center, but there is no possibility to adjust the instrument. Air free of CO or span gas is sent to the measuring instrument via a remotely controlled valve. The individual instruments can be dialled by a selecting disk or a key board and the measured values are displayed digitally. With the second development, the UNOR 1 is checked automatically in a fixed cycle which can be set in advance. Control is exercized by a switching clock. Thus, remote transmission channels for the commands can be saved. With a third development, the zero point is not only automatically checked in cyclic intervals, but it is also corrected. Wrong alarms by a drift of zero or sensitivity can be avoided by these remotely controlled methods of checking.

   A great step forward for a safe early detection of mine fires by CO-measurements is the automatic supervision of all measured CO values in a colliery by the use of process computers. The function of such a control system is above all the rapid giving of alarms of usually slowly developing concealed mine fires what requires a determination of trends by filtering-off short-term increases of CO, e.g. by fumes from blasting or by Diesel cars passing by. Program systems processed on process computers are tried at present on three Ruhr mines. In this stage of the trials, different mathematical processes are studied for the determination of trends and the filtering of troublesome CO increases. The process applied by us is based on a comparison of subsequent measured values, the

previous value being stored for further processing only if it is smaller or equal to the actual value. By getting average values for every ten minutes or every hour, predictions can be made, after what time a figure justifying a warning or alarming will be reached. The development of measured values with time can be shown for up to four CO measuring instruments simultaneously by means of a tv-color-monitor. Fig. 7 shows such an optical display where a difference is made by different colours between the normal measured value, a defect of an instrument, a calibration-procedure or an excess above a limit value. Daily and weekly reports which can be issued by a printer, save the tedious taking down of the values by hand. It may still be mentioned that air velocity and methane contents, too, can be controlled and recorded by this software system.

For a smaller number of CO measuring instruments to be controlled, a hardware system installed at the surface and permitting the control of, at the maximum, 16 measuring devices, has been developed. A compensation of the previous load by CO is possible by measuring the CO content of the intake air and by an adequate delay formed electronically. Peaks of CO owing to fumes from blasting or waste gas clouds of Diesel drives can be eliminated either by manual communication or by a delay of the alarm selectable according to the local conditions. Fig. 6 shows this measuring installation.

2. Development and Testing of new CO-Sensors. Needless to say, we tried for a long time to replace the expensive, heavy and relatively insensitive UNOR instruments by improved sensors. The sensors available in our country, however, could not yet fulfil the requirements of the coal mining industry. Though metal oxid detectors are very sensitive so that instruments with measuring ranges down to 20 ppm CO can be designed, their cross sensitivity against CH_4 and other reducing gases lies in the same magnitude as the sensitivity against CO and other problems as for the compensation of temperature arise. The electrolytic measuring cells recently used frequently especially for measurements in the field of environmental protection have at present still the disadvantage of long delays in the indication of results, the danger of intoxication by high CO contents and by traces of hydrocarbons, of a not yet sufficient stability and a serious cross sensitivity against nitric oxids and hydrogen. AEG develops at present, in cooperation with Ruhrkohle AG, a new electrolytic sensor which will not have these disadvantages, but we are most interested to learn of the results of the tests and experiences obtained

here in USA with the sensors developed by G.E.C..

3. The MINI-UNOR
The newly developed portable MINI-UNOR (Fig. 8) which proved itself sucessfully already in many trials, is operated also according to the principle of the non-dispersive infrared adsorption as the UNOR. In this MINI-UNOR, the amplitude-modulation is not realized by a wheel of apertures driven by an electric motor but by the use of a pulsed infrared source. The power demand of the MINI-UNOR has been decreased by a reduction of the volume of the measuring chamber of the infrared analyser to about 1/10 of that of the full-size UNOR and by the then possible reduction of the power input for the infrared radiator and by an energy-saving diaphragm pump incl. flow controller, so that the instruments can now be built to be intrinsically safe. Power is supplied by rechargeable batteries permitting continuous operation for eight hours. The instrument is designed with two settable measuring ranges from 0 to 200 ppm CO or 0 to 500 ppm CO as well as 0 to 5.000 ppm CO and settable optical and acoustic alarming systems. The light portability of the instrument makes it possible to localize concealed sources of mine fires and to control split air currents and special working points, especially during mine fire fighting. The MINI-UNOR can also be used for setting the injection of Diesel oil into Diesel drives directly at the working point.

Methane

Especially for monitoring of methane it is important to have intrinsically safe measuring instruments remotely supplied in an intrinsically safe way from a center unit placed in the fresh air. A further development of the above described MINI-UNOR is to fulfil this purpose, as the power supply to the MINI-UNOR can be done in an intrinsically safe way due to the low power input of 1,5 W only. With this further development, the infrared measuring system is incorporated into a measuring head which is supplied with energy through a two-core line from a central unit provided for main circuit connection. The measured value is indicated in the measuring head (Fig. 9) on a built-in instrument and is transmitted by low frequency through the same two-core line to the central unit. This transmission of the measured value by a modulated audio frequency secures a check of the line and safety against a falsification of the measured value by uncontrolled contact resistance. This enables lengths of the line of more than 1.000 m. The commands for prewarning and switching off in case the set limit values are exceeded, are formed directly in the measuring head, are indicated by LEDs and are remotely transmitted, again by low frequency, to the central unit. This production of commands directly in the measuring head instead of in the central unit has the great advantage of

an easy calibration and setting of the limit values which can be done independent from the possibly distant central unit. The measured value, too, is indicated in the central unit and is offered as a current signal as well as a frequency signal of 5 to 15 Hz for further processing such as recording or remote transmission. Five prototypes of this further development of the MINI-UNOR have been in use for two years, without any defects, on Ruhr and Saar mines The MINI-UNOR also is eminently suitable for the equipment with infrared CO_2-analysers whether in the portable or in the stationary, remotely-fed version. CO_2 monitoring may be a very valuable tool to get informations about the effectiveness of mine ventilation and the quality of mine air, especially in mines with Diesel-drives.

In addition to these infrared instruments, other instruments are employed in the West German coal mines which utilize for measuring the heat of reaction during the catalytic combustion of methane. This instrument manufactured by Auergesellschaft of Berlin, a subsidiary of Mine Safety Appliances, incorporates the known pellistors to measure the heat of reaction as well as, in a second measuring bridge, temperature dependent resistances which measure the heat conductivity of the gas. Thus, the double meaning of measured results occuring with high methane concentrations by a deficiency of oxygen, is eliminated. The Bergbau-Forschung GmbH of Essen-Kray developed, with these sensors, a measuring installation monitoring the CH_4 contents of working points underground operated under an exemptional approval of the mines inspectorate at several measuring points. Five intrinsically safe measuring heads are supplied with intrinsically safe energy, similar to the above described MINI-UNOR, from a central unit through a two-core line for each measuring head. It is preferred to place one of them at the end of the coal face, the next one in the advanced heading and the three others in the gate road equipped with the belt conveyor. Thus, the methane emissions from the extracted coal, from the still unworked coal and from the waste can be measured separately and special counter measures can be taken in case the limit values are exceeded. An analog and linearized signal of the measured value is formed in the measuring heads and this signal is transmitted by audio frequency modulated at 5 to 15 Hz to the central unit. A second audio frequency channel serves the indication of calibration and testing procedures in order to suppress alarms and disturbing signals of defective measuring heads and interrupted lines. The output signal can be measured and indicated by an external measuring instrument which can be attached from the outside. Fig. 10 shows a measuring head with this attachable instrument. The central unit (Fig. 11) has the duty to provide the five measuring heads with intrinsically safe energy, to receive and to prepare the informations supplied by the

measurung heads. The measured value is indicated digitally by LCDs and is compared with digitally settable limit values. The power supply and the processing of the measured values are identical for all measuring heads and are accommodated separately in plug-in. In a sixth plug-in, the signals of all five measuring heads are analyzed and summarized. By means of delay times for the giving of receipts for alarms separately for each measuring head and selectable according to air velocity and the distance of the measuring heads from each other, i.e. the delay of the possibility to switch the electrical equipment on again, methane clouds in the return air passage can be determined, thus enabling rapid measures to take coal winning operations up again. In case more than one measuring head becomes defective, the electronic processing system takes care that all electrical equipment and devices are switched off. Besides, a command for switching off is given in case one very important measuring head becomes defective which can be addressed and is generally the head at the end of the coal face. All information from the central unit can be transmitted to superordinate installations such as the coal face center, the underground district center or the entire pit control center. Provision is made that, instead of the sensors utilizing the heat of reaction for measuring, the measuring heads of the MINI-UNOR or other ventilation monitoring instruments, too, can be connected.

Analysis of Fire Gases

In order to support the efforts for a rapid construction of stoppings or combatment and control of mine fires, we developed a portable and intrinsically safe instrument for the analysis of fire gases. This measuring instrument is to be used in direct vicinity of the source of the mine fire and thus enables quick information about the effectiveness of the measures of mine fire fighting and increase, by a better knowledge of the composition of fire gases, the safety of the miners working near the zone of the mine fire and threatened by the dangers of explosion, poisoning and suffocation.

The following components of fire gases are measured:

- carbon monoxide in the three ranges 0 to 500 ppm CO, 0 to 5000 ppm CO and 0 to 10 % CO. To carry out these measurements, 2 MINI-UNOR measuring systems are used with absorption cells of different lengths;
- methane in the two measuring ranges from 0 to 5 % and from 0 to 50 %. Again, a MINI-UNOR system is used;
- carbon dioxide in the measuring range from 0 to 25 %, again with a MINI-UNOR measuring system;
- oxygen in the range from 0 to 21 %. The measurement is based on the paramagnetic properties of oxygen and uses the principle of the pressure effect between two gases of a different magnetic susceptibility, inter-

acting with an inhomogeneous magnetic field. CO_2 carried in a small pressure gas cylinder serves for comparison;

- hydrogen in the range from 0 to 5 %. The high heat conductivity of this gas which is about 7 times higher than that of air, is used for measuring. However, as CO_2 and CH_4 have also heat conductivities differing from those of the air, although to a less extent, there is a high cross sensibility against these gases. This cross sensibility is eliminated electronically by means of the separate measurements of these gases by the MINI-UNOR measuring systems.

As it may be that the gases from the mine fires are saturated with water vapor, they are dried by a Peltier-cooling trap in order to avoid the formation of condensates.

Such a complex instrument requires very much energy with the result, that the individual sensors must be switched on and enquired automatically and cyclically in order to secure the intrinsic safety. In an other mode of operation only one freely selectable component of the fire gases is measured continuously but a manual reswitching to another gas is possible at any time. In the first stage of development which has now been concluded, the instrument is supplied with energy from a battery. A remote energy supply is to be developed in a second stage. Fig. 12 shows the set-up of the instrument.

Methane in Gas Drainage Pipelines

Methane drainage from boreholes into the coal seam and the adjoining rock is an important measure to reduce the methane contents of the mine air what is important especially with regard to the above described mining-technical developments for an increase in productivity. Different methods are used at present to control and to optimize the gas drainage pipelines. They concern essentially the separate measuring of the methane concentrations and the flow velocities by means of UNOR instruments and anemometers. For more simple and continuous monitoring an instrument has been developed, which measures the flow velocity and the methane concentration in pipelines in a combined way. This gas pipeline probe (Fig. 13) consists of a central unit, to which one or two intrinsically safe measuring heads (Fig. 14) can be connected which are introduced into the drainage pipeline by 3/4" connections in distances up to 150 m. The pressure gradient at the Pitot tube introduced radially into the pipeline produces a flow through the measuring head. In this measuring head, the methane concentration is then measured by a diffusion heat conductivity bridge, whereas the flow velocity is measured by a thermal measuring bridge, the same which is used in the VENTOR, too. In order to obtain a linear indication of the flow velocity, the root is electroni-

cally extracted of the measuring signal. As an additional advantage of this root extraction an indication is obtained proportional to the static pressure in the pipeline what means that a correction of pressure is not required for a determination of the quantity of methane drainage. The influence by a change of the properties of the gas on the measurement of the flow velocity is electronically corrected via the measurement of the methane concentration. This enables a simple calculation of the quantity of the drained methane. The measured values for the methane for a measuring range from 0 to 100 % and for a flow velocity of e.g. 0 to 5 m/sec can be read off from two instruments on the central unit. A recording and remote transmission is possible the same as an indication of an excess of a limit value for methane. After overcoming initial defects, especially owing to water in the pipeline, five of these new measuring instruments are now operating most satisfactorily on different Ruhr mines.

CONCLUSIONS

Although this report about some new instruments developed by the Dept. Remote Control of the Bergbau-Forschung GmbH of Essen-Kray had to be short by necessity and could thus only be incomplete, it should yet have made clear, that many efforts are made to secure the health of the miners and to improve their safety by developing and using new measuring instruments to monitor mine ventilation. These new developments were financially sponsored by the Government of the Land Nordrhein-Westfalen and of the Federal Republic of Germany as well as by several research associations.

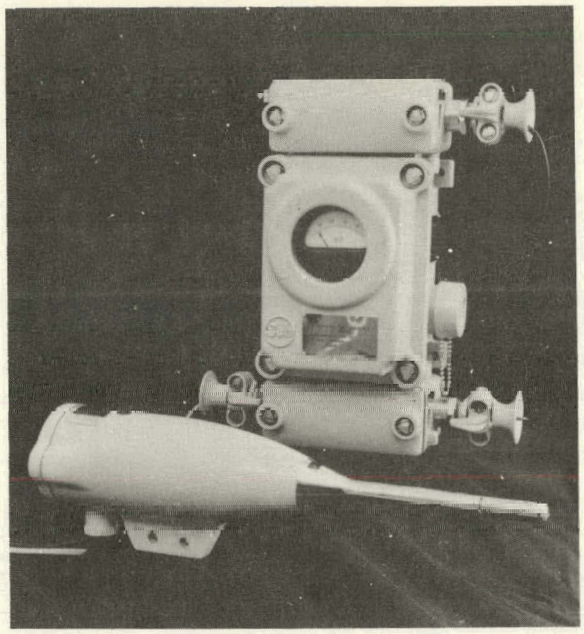


Fig. 1: The Ventor

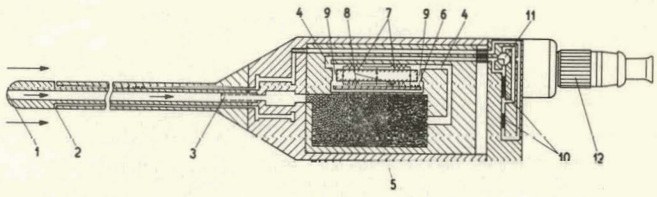


Bild 1 Funktionsschema der Meßsonde mit Meßbereichsumschaltung

Fig. 2: Inner design of Ventor Probe

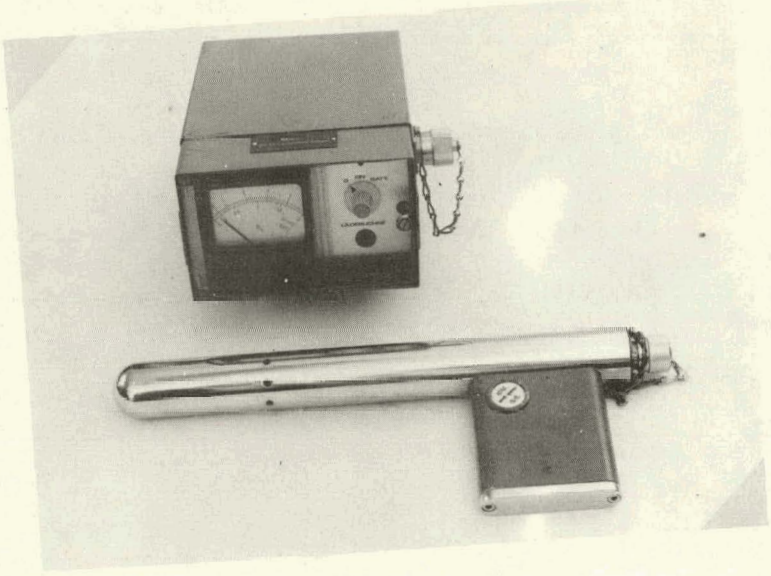


Fig. 3: The Mini-Ventor

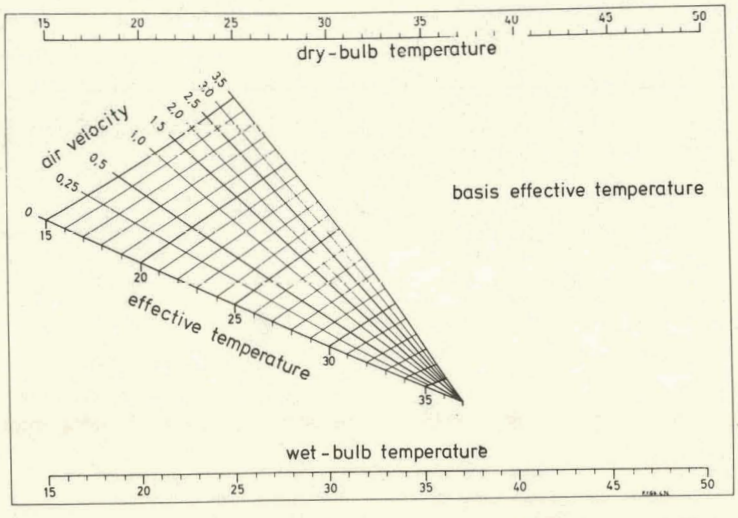


Fig. 4: Yaglou-diagram of basis effective temperature



Fig. 5: The Unor 1

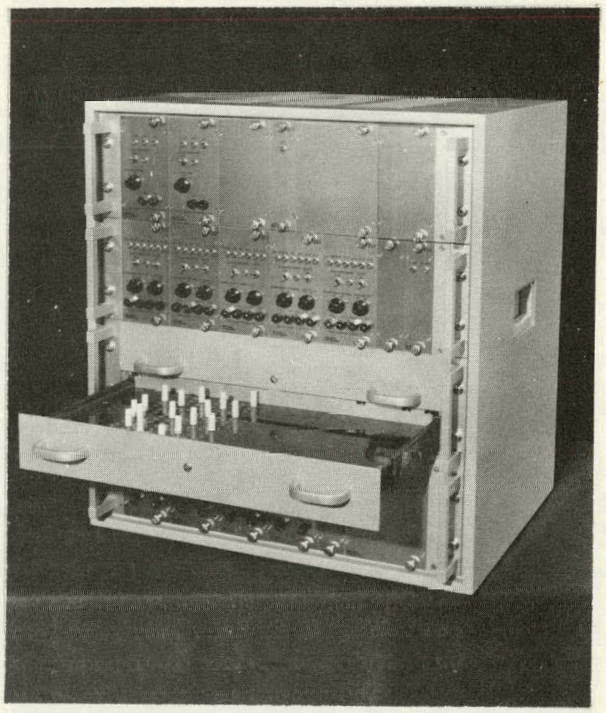


Fig. 6: CO monitoring device with programming desk

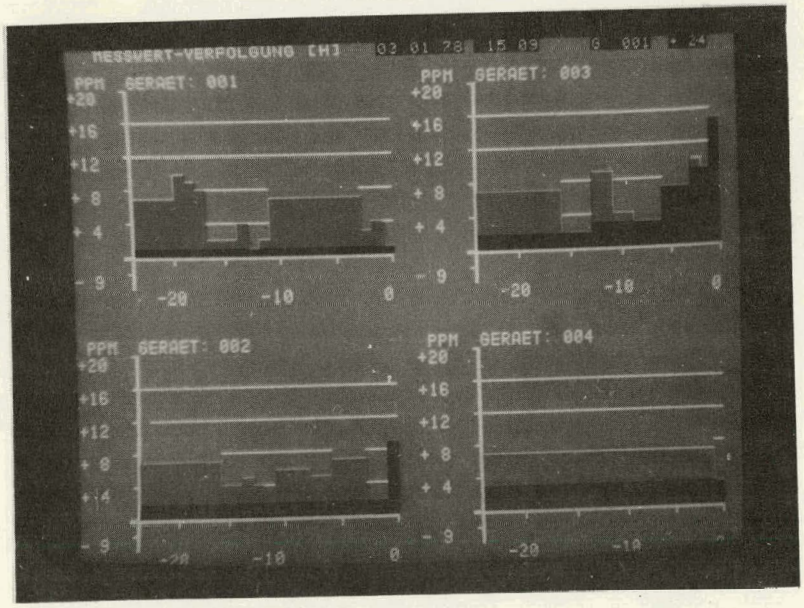


Fig.7: Monitor-display of mine-air CO content

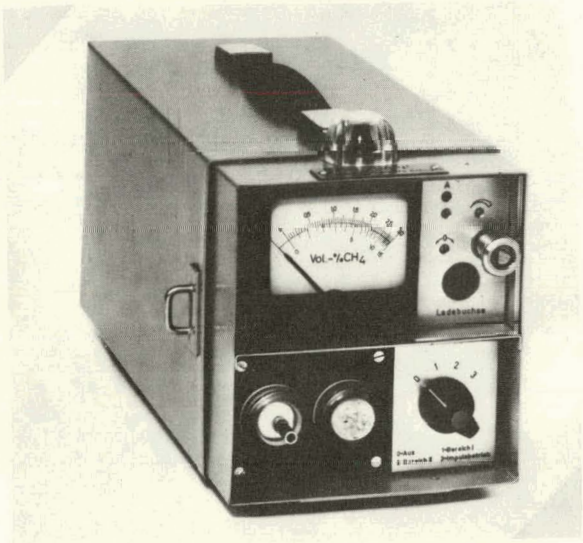


Fig. 8: Portable Mini-Unor

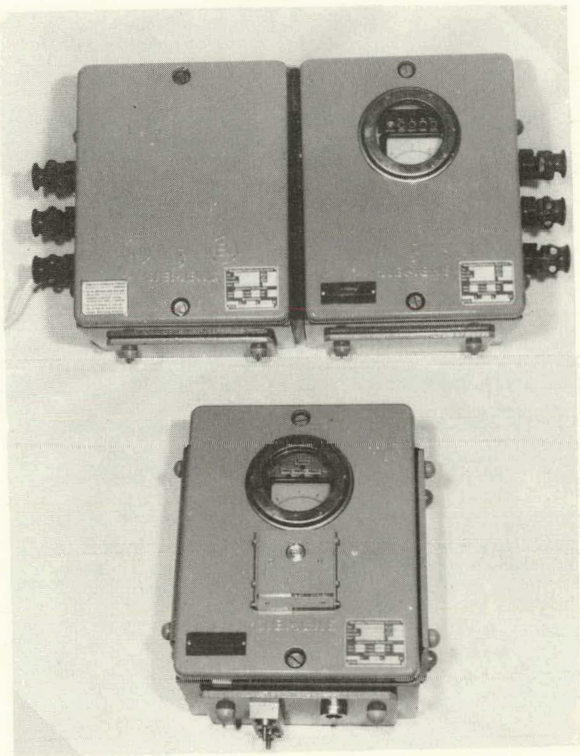


Fig. 9: Stationary, remotely fed Mini-Unor

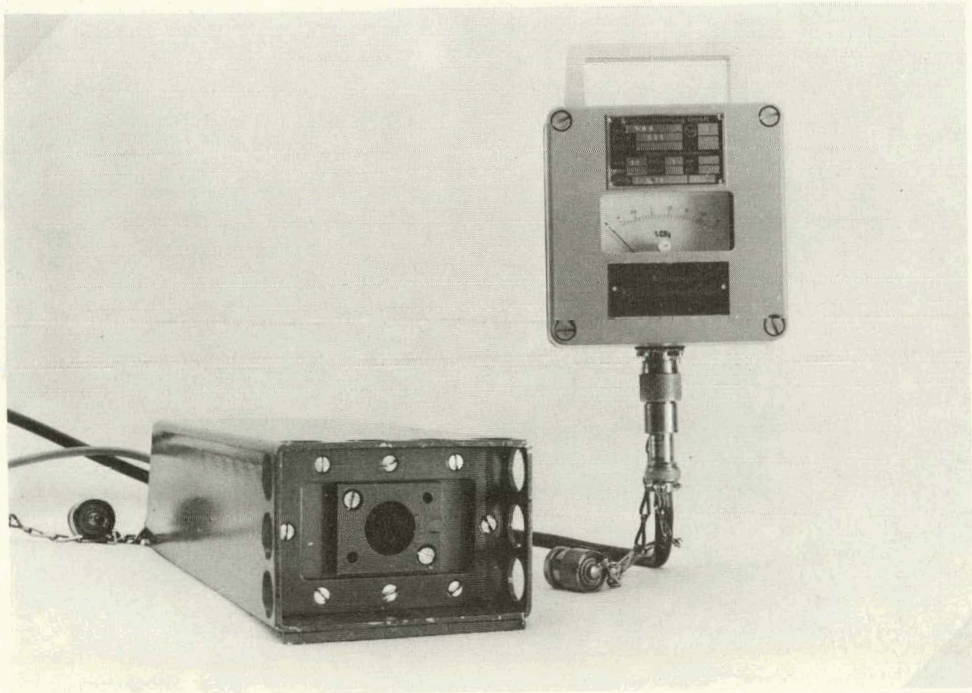


Fig. 10: Measuring head and attachable instrument of CH_4 measuring device

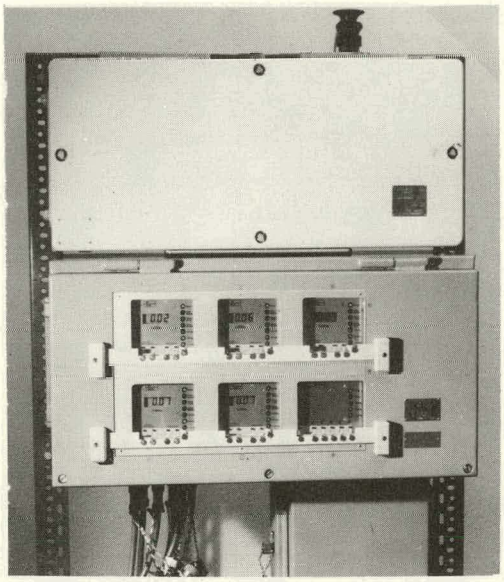


Fig. 11: Central unit of CH_4 measuring device

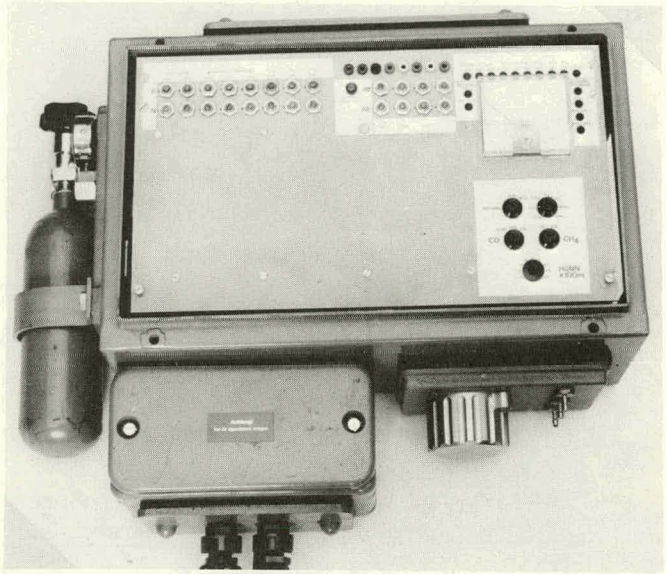


Fig. 12: Mine fire gas analyzer

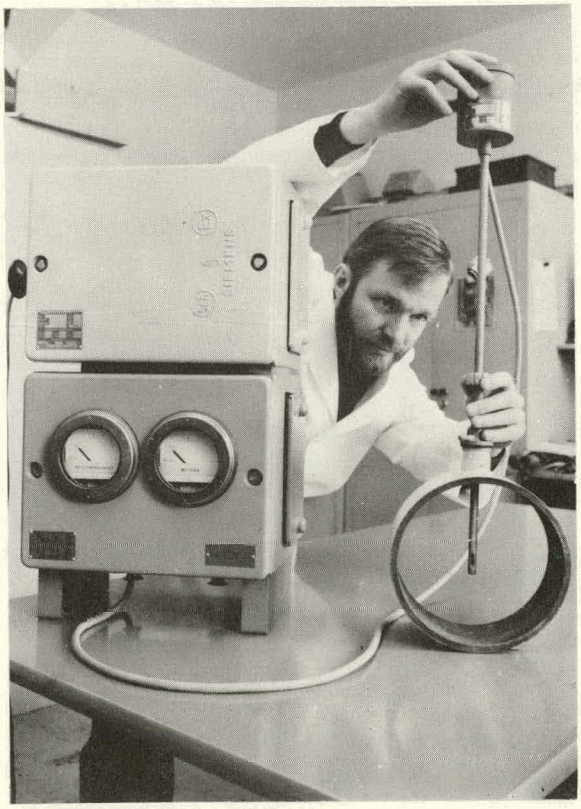


Fig. 13: Gas pipeline probe

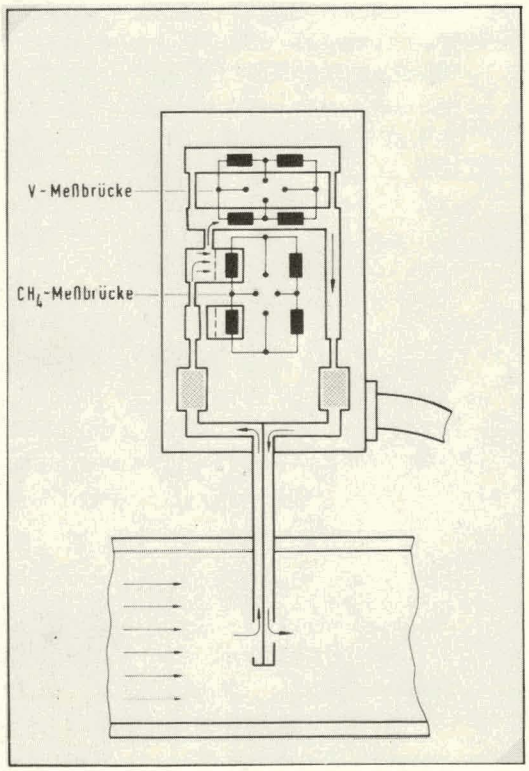


Fig.14: Measuring head of the gas pipeline probe

IN-MINE EVALUATION OF UNDERGROUND FIRE AND SMOKE DETECTORS

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U.S. Department of the Interior, Twin Cities, Minn. 55111

ABSTRACT

The current state of the art of fire and smoke detection technology is reviewed from the standpoint of suitability for use in underground metal and nonmetal mines. Detection modes, fire signatures, and environmental considerations are included. Preliminary results of long-term, in-mine tests are presented.

INTRODUCTION

Classification of the various smoke and fire sensors into three categories--optical view field, direct contact, and products of combustion (POC)--requires some explanation since most off-the-shelf devices (systems) have more than one principle of response and may respond to more than one parameter. Hybrid sensors may include heat detectors as well as photoelectric- and/or ionization-type product of combustion (POC) detectors, thus classification is according to the primary intended use. For the nonconventional fire detectors that have been developed for gas analysis and detection, the classification may be less obvious. Most of the nonconventional detectors are described in this report under POC types of sensors, even though they may depend on optical or filament-type principles. A further distinction of detector types divides them into point (or local) source, as opposed to the extended-area-type sensor.

For more detailed information on the types of fire detectors available, there are good survey articles (5; and National Fire Protection Association (NFPA) reference books (6), as well as manufacturers' sales literature.

FIRE SIGNATURES

Any product of a fire that changes the ambient conditions is called a fire signature (5) and is potentially useful for detection. To be practical for detection, a fire signature must cause a measurable change in some ambient condition. With other factors being equal, such as hardware costs and detection times versus hazard level, the preferred fire signature is one that generates the highest signal-to-

[1]Electronics engineer.
[2]Underlined numbers in parentheses refer to items in the list of references at the end of this report.

noise ratio in the earliest period of fire development. The principal fire signatures used in the detectors discussed herein are aerosol, energy release, and gas signatures.

Aerosols

Aerosols are particles suspended in air. The process of combustion releases large quantities of solid and liquid particles into the atmosphere ranging in size from 5×10^{-3} to 50 μm.
Aerosols resulting from a fire represent two different fire signatures. Particles less than 0.3 μm do not scatter light well and are classified as invisible. Those larger than 0.3 μm do scatter light and are classified as visible. The invisible aerosol signature is usually referred to as "products of combustion," and the visible aerosol signatures, as "smoke." Invisible aerosol is the earliest appearing fire signature in most cases. (Coal can evolve CO before particulates--PVC materials evolve HCl quite easily).

Energy

Fire also constantly releases energy into the environment and this energy release provides some useful fire signatures. The earliest energy signatures detectable with available hardware are the infrared (IR) and ultraviolet (UV) portions of the electromagnetic spectrum. With the exception of highly unsaturated hydrocarbons such as acetylene, infrared emissions from hydrocarbons are particularly strong in the 4.6-μm region owing to carbon dioxide and in the 2.7 μm region owing to water vapor. This radiation signature can be used effectively for detection but there is the possibility of noise from manmade IR sources. UV fire signatures appear in flames as emissions from OH, CO_2, and CO in the 0.27- to 0.29-μm region. They have been used effectively for flame detection, especially in cases where magnesium or its alloys may be involved.

Gases

Many gases are added to the atmosphere during a fire and these changes are called evolved gas signatures. A related change is the reduction of oxygen content (the oxygen depletion signature). These gases may include CO, CO_2, HCl, HCN, HF, H_2S, NH_3, and NO_x depending on the type of material burning. The most useful gas for detection is CO, since it is present in almost all fire situations. Slow burning and smoldering fires in particular produce large quantities of CO.

OPTICAL VIEW FIELD FLAME DETECTORS

Fire detection devices of the optical view type respond to radiant energy portions of the electromagnetic spectrum generated during flaming combustion of materials. The principal sensing elements used include solid-state detectors--junction type and bulk effects type, vacuum or gas-filled tubes, and thermocouples and thermistors for special applications. The photodetector family type fills a large part of the solid-state category.

Ultraviolet Sensing Detectors

The variety of UV detectors (wavelengths less than 0.4 μm approximately) is small compared with those in the IR region because of the basic problems associated with UV detection. Ordinary glass windows cut off radiation below 0.3 μm and quartz and UV grade sapphire become opaque below 0.18 μm. Below 0.1 μm there is essentially no suitable window material.

Infrared Sensing Detectors

Infrared detectors have the problem of background radiation at ambient temperatures (25°C) being entirely in the IR region (wavelengths greater than 0.8 μm). A common method of discriminating against this background is that of chopping the incident radiant flux so that the detector receives only a fixed radiant frequency, typically 4 to 30 Hz.

Infrared detectors are classified according to their method of response to heat and photon flux: thermal detectors and quantum detectors.

Thermal detectors respond to energy absorbed by a temperature-sensitive material or an absorbing film in contact with the temperature-sensitive material. Included in this category are such devices as thermocouples, metals or semiconducting layers with resistance a function of temperature (bolometers and thermistors), pyroelectric detectors whose polarization is temperature dependent, and gases with pressure being temperature dependent.

Quantum, or energy, detectors respond to photon flux falling on a sensing element and exciting electrons in a bound state to a free or conducting state.

Within these broad classifications the IR spectrum is divided into three divisions: near infrared (0.8-1.4 μm), intermediate infrared (1.4-7 μm), and far infrared (>7 μm). Near infrared requires conventional silicon and germanium detectors; intermediate infrared uses special IR detectors; and far infrared calls for thermal detectors such as thermocouples and thermistors. In some cases there is an overlapping spectral area of response. PbS is one of the most versatile photoconductors and responds to IR from approximately 1 to 4 μm. Doping Ge with Hg and Cu extends its response to 8 to 14 μm. Sensitive areas range from 0.01 mm^2 to 10 mm^2, with time constants shorter than 1 ns. (InSb may be more useful than PbS since it covers the CO and CO_2 bands).

Pyroelectric-type detectors are a more recent development (1-2). These consist of a slice of ferroelectric material sandwiched between electrodes to provide a sensing cell. Typical materials include triglycine sulfate, tourmaline, Rochelle salt, barium titanate, and polyvinyl flouride. Electrically, the material behaves similarly to a capacitor dielectric with a strong temperature-dependent polarization

of the magnetic domains. One of the electrodes is transparent to IR radiation which causes a small temperature increase producing a polarization charge on the pyroelectric material and a corresponding voltage change across the two electrodes of the cell.

HEAT SENSORS (DIRECT CONTACT DETECTORS)

There are two general types of heat sensors; those that employ the fixed temperature principle, and those that employ the rate-of-rise principle (6). With the fixed temperature approach an appropriate temperature level setting is selected. When the active element is heated to its operating temperature the active element will bend, expand rapidly or maximally, fuse (change its electrical conductivity and physical state), or produce an electromotive force (emf) which can be amplified to actuate an alarm. Commercially available devices include temperature-sensitive ampoules or pellets, bimetallic elements, eutectic solders and salts, snap discs, thermocouples, and thermistors as point sensors. There is a subcategory of continuous line or wire types using thermistor material, eutectic salt, pressurized gas, or twisted insulated wires under tension. Excluding specially prepared rapid response thermocouples, a major disadvantage of these fixed-temperature types of sensors is slow response time because of thermal inertia. Another disadvantage is their inability to detect low level incipient combustion in its early stages. Commercial fire detector thermocouples have response times on the order of a few tenths of a second. Advantages of direct contact detectors are low cost, reliability, freedom from maintenance, and insensitivity to vibration and dust-laden atmosphere. Continuous line heat sensors have been tried in underground mines.

PRODUCTS OF COMBUSTION DETECTORS

Products of combustion in fires include solid particulates and liquid mists (including invisible and visible particle sizes), ionized species gases, and radiant energy. Combustion product detectors sense one or more of these constituents including heat or flame.

Ionization-Type Detectors

Ionization-type detectors constitute one general class of POC detectors. The most common source of ionization is an alpha emitter such as $Ra_2{}^{226}SO_4$, or Am^{241}. Beta emitters are sometimes used but not as often since stronger sources are needed to create ionization currents similar to those produced in alpha detectors. Kr^{85} is one beta emitter that is used. The radioactive emitter ionizes the air in a chamber between two electrodes. Current measurable in picoamperes is produced from production and transport of positive and negative ions to the opposite poles of the plates. A decrease in current, relative to clean air, is obtained when combustion products enter the chamber because the combustion particles become ionized and, being larger and heavier than the air molecules, move much more slowly toward the end of the chamber. An electric circuit detects the drop in current and initiates the alarm.

which is a function of gas concentration. The adsorption of a gas molecule on the surface of a semiconductor generally results in the transfer of electrons due to the differing energy levels of the gas molecule and the semiconductor surface. Oxygen, which can accept electrons is adsorbed on the surface of n-type semiconductor. The transfer of electrons from the donor level of the semiconductor to the layer of adsorbed gas results in decreased conductivity of the semiconductor material.

When a TGS that has adsorbed oxygen in this manner comes into contact with reduction or combustible gases such as CO, hydrocarbons, etc. the molecules of these gases are adsorbed and the transfer of electrons is in the opposite direction to the oxygen reaction, releasing the electrons to the semiconductor space charge layer and causing a large increase in the conductivity of the sensor. The sensor output is sufficiently large to allow gas detectors to be designed using a minimum number of components, thus allowing the production of low-cost detectors.

Significant advantages of these solid-state electrolytic cells are the very long lifetimes expected, usually several years, the low cost and simplicity inherent in its design, and the relatively simple electronics needed to utilize the sensor output. Other features include not being permanently poisoned by the toxic gases, resistance to vibration and shock, and having no loss of sensitivity even at gas concentrations so high that air (oxygen) is displaced.

A severe disadvantage is its ease of alarming in areas containing engine combustion products, and its sensitivity to a wide variety of easily oxidized gases not necessarily associated with products of combustion.

For fire detection use the TGS responds readily to CO, which is one of the principal gases given off in the early stages of fire. However, because of the sensitivity of TGS to gases other than POC and exhaust gases, its practical use may be limited to a hybrid system involving ionization or photoelectric detectors. TGS detectors can be calibrated within the range of 200 to 1,000 ppm CO and to a limited extent can be made selective to CO.

Electrochemical Detectors

Electrochemical (fuel cell) techniques for CO sensing depend on the electrochemical reaction of the detected gas, oxidized or reduced (depending on the sensor) at an appropriate electrode potential, producing an electric current that, because of membrane diffusion controlled conditions, is directly proportional to the gas concentration. These devices are subject to interference from gases other than the desired one for detection, but are capable of precision calibration, making them useful as an analytical device. They tend to be expensive and not too rugged for the mine environment but are useful for establishing alarm levels and personnel monitoring.

Photoelectric-Type Detectors

Photoelectric fire and smoke detectors constitute another general type POC detector. The requirements are a source of light and a detector of that light to measure its radiant power. Four different modes of operation are possible depending on the amount of light transmitted or absorbed by the medium, the amount reflected, scattered, or refracted. Most, if not all, commercially available units possibly suitable for mine use are of the light scattering type. A beam of light from a source travels across a light-tight chamber to a light trap or collector opposite the source. A photocell positioned at right angles to the beam senses no light as long as the air inside the chamber is clean. The chamber is open to its surrounding atmosphere through baffling, and if smoke (POC) enters the chamber, light from the beam is reflected or scattered in all directions. Some of this scattered light reaches the photocell, changing its resistance and via suitable electronics initiates an alarm. The alarm level is adjusted to actuate at a given smoke concentration level (normally 3.3 to 6.6 pct $-m^{-1}$ or 1 to 2 pct per foot). In a few models the electronics are also designed to give an alarm if the rate of rise of smoke obscuration exceeds a given value (commonly 0.33 pct $-m^{-1}-min^{-1}$ or 0.1 pct per foot per minute). Auxiliary or backup heat detection is often provided with these detectors.

Photoelectric detectors, like ionization detectors, may respond to particulate matter from sources other than fire. A dusty mine environment might cause some false alarms and would also require periodic cleaning of the sensor chamber internal surfaces. In a wet mine, aerosol forms of fog and mists might cause some false alarms. These detectors are also somewhat dependent on airflow velocity (good chamber design) and location because of the tendency for smoke to stratify in airstreams. They are also less sensitive to black smoke compared with white or grayish smoke.

Solid-State Detectors

Solid-state devices for detecting and sometimes sampling the gaseous components of products of combustion are a third general type of detector that can function as a fire detector.

A promising new development in POC gas detection is the semiconductor cell detector known as the Taguchi gas sensor (TGS). This device uses a selection of bulk n-type metal oxides such as SnO_2, ZnO, and Fe_2O_3 impregnated in a solid-state matrix material supported by conductive filaments. Variation of the chemical formulation allows the cell to be made sensitive to a particular gas or group of gases, and insensitive to others.

A heater electrode imbedded in the cell provides sufficient increase of cell temperature above ambient to allow the cell to operate by diffusion, eliminating the need for a sampling pump arrangement. A collector electrode provides for measurement of cell conductivity,

Optical Gas Detectors

Optical gas sensors are generally confined to laboratory use, although recent prototype models have been developed for personnel hazard monitoring. These use the strong adsorption band of CO at 4.65 μm (infrared). Radiation from an infrared source is passed through a cell through which the sample gas flows and is adsorbed by the CO molecule. A filtered infrared detector responds to this change in radiation and its output is compared with a reference cell, conditioned by suitable electronics and read out on an appropriately marked meter.

A technique useful with different POC detectors is the use of tube bundles (7). With these, air samples can be drawn from different sections of the mine and sequentially monitored at a central location. This procedure provides protection and stability for the detector systems and can add to the flexibility of the mine fire protection system. Slow response times and wall-diffusion losses for submicrometer particles provide inhibitions to its use.

ENVIRONMENTAL CONSIDERATIONS

There are really two environmental problems to be faced in designing a reliable fire detection system: Will it continue to operate, and if it does, can it discriminate against the normal mine ambient parameters?

The latter problem can possibly be dealt with by the use of combinations of detector types in the system to provide detection of cellulosic and hydrocarbon fires, spontaneous combustion, and electrical fires. Ambient parameters will differ in various parts of the mine. Occasionally air ventilation networks can improve a POC detector's sensitivity and effectiveness. Sampling tube networks or remote detectors with telemetry can monitor several locations individually. A detector type useful in one location may be quite unsuitable for use in another location; therefore, if conditions change with time, the sensor system should be reevaluated.

Ability to discriminate against normal mine ambient parameters is an important consideration in the design of reliable fire detection systems. Mine Safety Appliances Research Corporation (MSAR) under a Bureau of Mines contract (9) provided CO, CO_2, NO_x, and NO concentration data for shot firing and diesel operations in selected mines and sampling sites, within the mine. Table 1, taken from their report, summarizes much of their data in the form of ratios of contaminants measured in connection with various mine operations. The report suggests that a duel sensing system involving contaminant ratios may provide an approach to minimizing or eliminating false alarms. The CO/CO_2 ratio of diesel exhaust is quite comparable to that of the combustion products of a metal mine fire. Similarly, the CO/NO_x ratios for shot firing and a leaking bulkhead, where a hot spot was thought to exist, are also quite similar. The CO_2/NO_x ratio is significantly different for shot firing and diesel operations compared with the ratio for a fire and/or the leaky bulkhead with a hot spot. This much higher latter ratio may

TABLE 1. - AVERAGE CONTAMINANT RATIOS FOR VARIOUS UNDERGROUND ACTIVITIES

Activity	CO/CO_2	CO/NO_x	CO_2/NO_x
Shot Firing.....	0.38(0.44)	6.35(4.58)	11.6(12.4)
Diesel.....	0.04	6.9	97
Fire.....	.025	100	4,000
Leaking Bulkhead... (4,200')	.03	--	--
Leaking Bulkhead... (3,600')	.001	12	8,000

provide a reliable means of fire detection. The report also suggests that studies of the time-distance behavior of the NO/NO_x ratio be made if the CO_2/NO_x ratio were to be employed as a means of incipient fire detection.

In an early Bureau contract study by the Gillette Research Institute, two different types of battery powered ionization detectors were taken underground for a short-term performance evaluation. One, taken into the Bunker Hill mine, Kellogg, Idaho, performed poorly after a few hours of operation in areas contaminated with high concentrations of aerosols and in regions of high ventilation airflows. It did operate satisfactorily in regions supplied by low-velocity intake air.

The other detector, a dual-gate reference type, performed reliably at Central Rock Company, Lexington, Ky. The latter device was not necessarily superior to the former because the Bunker Hill mine presented far greater extremes of environment. What the experiment tended to demonstrate was that state-of-the-art ionization sensors, developed for normal industrial or residential use, military or commercial aviation fire protection, have questionable reliability or, in this case, are inappropriate for use in a mine environment.

It is fair to say that there is no universally applicable fire and smoke detector--one that responds equally well and consistently to all types of fires. The various detector types have been discussed along with some of their advantages and disadvantages. Long-term evaluation of the various detectors' sensitivity and reliability under the extremes of ambient conditions just described are underway and may lead to viable, cost-effective detection and suppression systems for underground mine fires. Until such systems have been developed and environmentally and operationally reliable detectors of various types have been found through long-term testing, it is best that unreliable fire detectors not be installed in mines.

Another important consideration involving mine environment is the spacing and positioning of fire and smoke detectors in the mine. Little is known about optimum detector positioning and spacing in mines. NFPA installation standards do not offer any criteria directly useful in a mine because their codes deal with residential and commercial situations. Mines vary so widely in distribution of combustibles, ventilation patterns, and roof support systems that an installation code may not be feasible or even desirable. Complicated ventilation patterns, random ceiling obstructions, and a variety of tunnel shapes would not allow general applicability of such a code.

Because of the fuel-to-air ratio influence, a knowledge of the relationship between fire and mine ventilation is essential for the proper selection and location of sensors and detectors and also for the correct interpretation of information provided by them. Mine fires can be categorized into two distinct types--oxygen rich (overventilated) or fuel rich (underventilated). The fuel-to-air ratio determines which type occurs; the oxygen-rich type can evolve into the fuel-rich type by spontaneous growth. Timber fires (fuel rich) represents an extreme toxicity hazard because of low O_2 concentrations and high CO and CO_2 concentrations. Fuel-rich conditions also produce more smoke, increasing toxic species exposure and obscured vision.

The greatest hazards of mine fires are caused by toxic and sometimes explosive products of combustion being carried through the mine by the ventilation system and unexpected airflow reversals carrying toxic fumes to intake air ventilation areas, such as fire escape routes, hoist areas, and other areas usually thought of as safe in the event of a fire. The analytical prediction of airflow reversal in a mine, even under ideal conditions, is a very difficult task and beyond the scope of this report. However, Greuer (4) has compiled a large number of references and provided a detailed accounting of the influence of mine fires on the ventilation of underground mines for those who wish to pursue the problem in detail.

Greuer (4) makes a few quantitative predictions of ventilation disturbances that may be useful in interpreting the indications seen when monitoring a fire detection system. These include the following:

Throttling and natural draft changes can cause changes in the quantity of ventilating air currents and sometimes a reversal of their direction. Similar changes can occur in neighboring connected airways as well as that of the fire scene.

Air (oxygen) quantity reductions (O_2 pct) in non-gassy mines can be neglected. In gassy mines air reductions can lead to explosive mixtures which may be carried back through the fire zone.

Ventilation disturbances can also cause smoke layering, which may affect the ability of a detector to operate effectively.

BUREAU OF MINES EXPERIENCE

Shaft Installations

To work toward an answer to the major problem of fire hazard and contaminated air resulting from fire, the Department of the Interior, U.S. Bureau of Mines, contracted with the FMC Corporation, San Jose, Calif. (Contract H0252016 "Mine Shaft Fire and Smoke Protection System"), to evaluate mine shaft fires and their hazards (10). The contract also called for the development and demonstration of a low-cost, reliable mine shaft fire protection system that could be adapted to a majority of metal and nonmetal mine shafts, raises, and winzes, especially in deep mines. Results of this contract have been published elsewhere (10) and so need not be extensively detailed here. However, as an example of a proven system, sensor and detector components that performed successfully in underground tests with controlled fires will be highlighted here.

As part of the background preparation for designing the systems produced, it was found that approximately 67 pct of shafts in metal and nonmetal mines are of wooden construction, and contaminated air was responsible for about 77 pct of fatalities from fires. Causes of fires in mines were predominantly electrical (35 pct) and welding (18 pct). Most fire-connected fatalities occurred in or near shaft stations, the gathering point to normally leave the mine.

Four major needs were brought out for providing effective mine fire and smoke protection:

1. Reliable sensors are necessary for early fire and smoke warning.

2. Isolation barriers (vent doors) would stop the flow of contaminated air.

3. Available water supplies could assist with fire extinguishing.

4. Recording sensor systems and control ventilation doors and extinguishing systems from the surface would be preferable to sending fire-fighting teams underground.

The possibility of a fire starting somewhere in the mine is always present because combustibles and ignition sources are usually close together. In addition to constant fire prevention efforts, provisions should be made to detect and extinguish fires as soon as possible. Sensors and detectors are needed because fires can occur spontaneously in abandoned workings or accidently in unattended areas. Furthermore, depending on detection by persons working in or passing through an area is not realistic.

Besides simply detecting a fire, systems should be made to alert an attended area that fire exists, and where it exists; to alert people underground that a fire has been observed or reported; to take some action to isolate the fire area; and to extinguish the fire if possible. The contract showed that meeting these requirements was possible with

existing equipment integrated into a functional protective system consisting of sensors, extinguishing agents, isolation barriers, and appropriate controls.

The harsh environmental conditions that exist underground have already been mentioned. Because labor and maintenance efforts are expensive underground, all components used in this detection-alarm system must be extremely rugged to limit repair needs or they may be out of service at just the wrong times, and reliable to instill confidence in their ability to perform and alarm only in a fire situation.

The first prototype mine shaft fire and smoke protection system was tested at an inactive silver mine in the Coeur d'Alene district of Idaho. The system was installed 3,000 feet below the shaft collar to protect 50 feet of shaft, the shaft station, and 100 feet of drift in two directions.

Environmental conditions in the mine were 100° F and essentially 100 pct humidity. Because the mine was inactive, there was little or no dust present during the 2-week test period. Ventilation was not typical of that found in a working mine because the mine was used exclusively for ventilation and as an alternate escape route for an adjoining working mine. Airflow was temporarily modified for the test.

The sensors installed included thermal wire, CO gas, and ionization types. The thermal wire provided backup reliability for the other two early warning types. CO gas and ionization sensors were placed in sections of the shaft and shaft station areas. A pair of each was placed near the shaft crown plate to sense air entering the area, which was downcast intake air. Others were located near the ventilation barriers towards the mine workings (away from the shaft station area).

Sensors were selected to respond to health hazard levels of fire or contamination by fire (gas/smoke), while being available off-the-shelf, reasonably priced, and compatible with metal/nonmetal mine environments as previously described. Sensors consist of smoke (ionization) detectors, Becon Mk II, manufactured by Anglo American Electronics, Republic of South Africa; two carbon monoxide (gas) sensors, Model CO-181 manufactured by Dynamation Enterprises, Inc., Ann Arbor, Mich., and Model MSD-1 manufactured by Emmet Co., Ann Arbor, Mich.; and thermal wire, type WPP, 155° F manufactured by the Protectowire Company.

The ionization detectors are referred to as smoke detectors in the system even though they primarily sense invisible submicrometer particles (as distinguished from the second detector type, labeled CO, which responds to CO and other gaseous components of smoke).

During extensive factory tests, the ionization detectors performed well. They consistently detected wood fire smoke within habitable levels. Test data indicate the detectors alarmed at smoke density levels of 2 to 6 pct obscuration per foot. This compares favorably with industry standard set points for commercial and residential smoke detectors. A poor response to burning plastic produced smokes appears to be characteristic of ionization-type detectors, and the Becon unit is no exception. On the other hand, the Becon unit is designed for mine

environments and does not readily exhibit problems from corrosive environments, dust, humidity, and very high air velocities.

The CO detectors in the system used the Taguchi gas sensor (TGS). They were used at factory settings of 200 ppm CO on the Enmet unit and 75 ppm CO on the Dynamation units. These levels were felt to be realistic for the conditions expected in the test shaft. There was no blasting or diesel equipment operating to increase the CO level at the demonstration site. On the other hand, a CO concentration of 150 ppm may be encountered with some regularity in active mines where blasting and diesel equipment are present. A level of 150 ppm is tolerable without hazard for up to 3 hours so setting alarm points of CO monitors above 150 ppm would represent a realistic alarm level within the limits of safe habitation for short periods, but an extended 200 ppm concentration could be a valid indication of a fire, especially if combined with other indications.

A surface control unit was installed in the hoist room and connected to an underground control unit in the shaft station at the 3,000-foot level. The latter was the junction point of the sensors and related components and provided the interface between them and the surface control unit. The two control units were connected by a single pair of wires with a second pair for redundancy. Multiplexing the signals allowed the surface control unit to handle 16 inputs from the underground control and process 8 outputs for transmittal back to the underground unit.

A fire fueled by wood cribbing was built in a steel pan 3 feet square, about 18 inches deep, with a lid that was raised and lowered to adjust smoke density and products of combustion.

Smoke and CO sensors responded to contaminants in the fire area, all system commands were remotely operated from the surface control unit and performed as designed. The fire was extinguished by the installed sprinkler system during a 6-minute discharge. The thermal wire was located too high above the controlled pan fire to reach its alarm temperature and did not respond.

The first prototype system was installed for only 2 weeks in the Idaho mine, which was not sufficient to test the long-term reliability of the system and components. A second mine was found where the system could be installed and given a longer test. This mine was the Union Carbide Corporation's Pine Creek mine in Bishop, Calif.

Several modifications were made to adapt the system to the particular mine location and it was installed and continuous monitoring began on December 19, 1975. A second prototype system, based on recommendations derived from developing, installing, and monitoring the first prototype, was built and installed at a second level in the second mine as an add-on unit to the first prototype using the same interconnecting wires, and monitoring began during April 1976. The latter system included a new micro processor controlled surface unit, replacing the original prototype surface unit. System testing was concluded in July after controlled fire test demonstrations were performed in each of the two underground areas. Final results include 3 months operation of the

first prototype system, providing 87 days of 24-hour-per-day monitoring. The second prototype system was installed $3\frac{1}{2}$ months, providing a combined total of 105 additional days of monitoring for both levels.

The surface control unit was installed on the surface at a guardhouse where it was monitored 24 hours per day. The fire and smoke protection systems were placed at two locations deep within the mine--one in a large compressor station 14,000 feet in the mine at the 9,400-ft level, and the second 14,000 feet in the mine at a shaft and shaft station area at the 11,271-ft level. Because the mine is inverted, deeper operations are at higher elevations.

Three types of sensors were used at each of the two test sites as in the initial test setup. Thermal wire was strung throughout the drift and machinery space of the compressor station and around and down the shaft at the 11,271-ft level.

Ionization sensors and CO sensors were used in pairs except none of the CO sensors were used in the compressor room. An ionization sensor was located above the compressor machinery, one located on each side away from the compressor station, in intake air, beyond remotely controlled vent doors, and one in the upcast exhaust airflow from the shaft at the 11,271-ft level. An additional ionization sensor was placed in the diesel maintenance area, three fans were monitored, one fan was controlled, and air lock doors were monitored.

During the monitoring period, the smoke (ionization) detector in the diesel maintenance bay regularly alarmed during vehicle activity, requiring an adjustment of its sensitivity. Also the smoke detector in the compressor station was readjusted because of alarms. The sensors responded to various known mine activities--diesel exhaust, blasting, and gas and electric welding and cutting vapors.

After the monitoring period, each location was subjected to a test fire using a pan and cribs identical to those used in the Silver Summit test. At the 11,271-ft level it was allowed to burn for nearly 12 minutes before the surface control unit was used to turn on the installed sprinkler system. During the fire the sensors detected over 100 ppm gas, and smoke, and alerted surface personnel that they were in alarm. All alarming and control functions performed satisfactorily through the surface control unit operated by surface personnel.

At the 9,400-ft level a similar fire was allowed to burn for 13 minutes before the surface control unit turned on the sprinkler system there. Again, alarm and control functions operated satisfactorily through the surface control unit, responding to the buildup of smoke.

Other Underground Tests

Among the recommendations coming from this contract testing was that continued laboratory and in-mine tests should be conducted to learn more about gas and smoke detectors. Responses to fires from different fuels, nonfire stimuli, effects of air velocity variations, dust buildup, and effects of mine environments would be of interest over long-term testing.

Based on the recommendations, the Bureau's Twin Cities Mining Research Center entered into a purchase contract with the FMC Corporation to obtain a sensor package suitable for long-term, in-mine testing. A cooperative agreement for locating and testing the package was also negotiated with the Hecla Mining Company, Lakeshore mine near Casa Grande, Ariz.

A three-sensor package containing a Becon Mark II smoke sensor, and Enmet ISA-33 CO gas sensor and two Pyrotector 30-2013-9 flame detectors were assembled An additional CO gas sensor (Dynamation CO 181) was supplied for additional testing at a different location in the Lakeshore mine.

The three-sensor package was installed in the underground diesel fuel storage area located at the end of the 500 East Exhaust Crosscut. The single CO gas sensor was installed in the south decline near the mine's existing Ecolyzer gas sensor. Each sensor is connected by telephone wire pairs to a 9-channel recorder on the surface. These packages were installed in late March 1976 and have been undergoing long-term testing.

Periodic visits to the mine to observe the sensors have shown some problems. The interface electronics and the CO sensors are packaged in NEMA IV electrical enclosures and well protected from the mine environment. The TGS sensors that are part of the CO monitors are not stable in that their base level output drifts over time and with temperature variations. They are also quite sensitive to diesel exhaust emissions and blasting residue gases in the mine.

The flame detectors have remained operational but they do become dust covered with time since they are in an exhaust crosscut where the air velocity is quite high and dusty.

The smoke sensor has remained operational but after several months was not alarming as it should. Blowing accumulated dust away from its baffle entrances caused it to go into alarm, and we were able after some trials, to get it to alarm by checking its trigger level adjustment. Dust does build up in the baffle area in this location to the extent it needs to be cleaned out every few months. The alarm level was adjusted later so it would not respond to the exhaust fumes from standing, idling load-haul-dumps in the passageway to the fueling area. Additional long-term testing continued in this mine as well as others with differing environmental conditions.

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SILICONE VAPOR POISONING OF CATALYTIC METHANE SENSORS

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ABSTRACT

Various silicone materials have been proposed for use in mining environments. The catalytic methane sensors now employed are poisoned by silicone vapors. Activated charcoal and carbon cloth adsorbents will increase the sensor lifetimes in an environment containing significant silicone vapor concentrations. The results of a study of catalytic methane sensor operation with adsorbent materials in hexamethyldisiloxane-air mixtures are reported. Sampling techniques developed for measuring silicone vapor content of mine atmospheres are described.

INTRODUCTION

The Code of Federal Regulations Title 30, section 75-300, administered by the Mining Safety and Health Administration (MSHA), Department of Labor, requires methane gas concentration monitoring in coal mines. Such monitoring is used both as a measure of mine atmosphere flammability and as a criterion for ventilation adequacy. Methane (CH_4) action levels are set at less than 0.25 % CH_4 for use of air from nonworking areas, less than 1% CH_4 for permissible work areas, greater than 1% CH_4 for increase in ventilation, and greater than 2% CH_4 for automatic deenergizing of machinery operating at the coal face. Thus accurate and reproducible methane sensors are mandated to insure productive and safe coal mine operation.

Methane measurement by the use of catalytic methane sensors is used currently in most of the methane monitors and detectors underground. With this sensor methane is catalytically oxidized in air on finely divided platinum (Pt) or palladium (Pd) ceramic surfaces. The resulting heat of combustion increases the temperature of the surface and of an inner platinum wire spiral. The methane concentration in air is proportional to the resistance increase of the inner platinum wire. The catalytic methane sensor was developed in Great Britain by the Safety in Mining Research Establishment (SMRE) and has an excellent record of accuracy and stability for measuring combustible gases such as methane in mines as long as there is an adequate oxygen supply (greater than 10%) and absence of several specific poisons for the catalytic surface.

The catalytic methane sensors using Pt or Pd catalysts are poisoned--suffer a loss in response--when brought in contact with several classes of chemicals. Manufacturers' literature and earlier reported studies list several of these chemicals (table 1). In general, compounds that react with Pt or Pd metals to form highly stable metal compounds--sulphides or phosphates--are poisons as well as silicones and lead oxide film formers. Organic halides are strongly adsorbed and are classed as temporary poisons. Although such poisons for catalytic methane sensors have been identified for some time, they did not become a problem during mining operations until lately when the commercial use of silicones or phosphate esters became more common.

Silicone compounds may be used in the mining industry in many commercial preparations. These include grease formulations, silicone rubbers, hydraulic fluids, heat transfer fluids in transformers and capacitors, oils, and water emulsions for polishes, waxes, and aerosol sprays.

Silicone compounds in these formulations consist mostly of linear dimethyl siloxane polymers. They are produced by three sources in the U.S.--Dow Corning[1], General Electric, and Union Carbide. The poly dimethyl siloxane fluids have several unique properties:

Wide fluid or liquid range, -50°C to 100°C
Viscosity range dependent on molecular weight, 0.6 cs to 10^6 cs
Extreme stability to thermal degradation or oxidation at high temperatures, stable at 200°C in air
High surface wettability.

Selected physical properties of the linear dimethyl siloxane polymers with trimethyl siloxane terminal groups are summarized in table 2. The number of silicon atoms in polymers may extend to

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[1] Use of name brands is for identification only and does not imply endorsement by the Bureau of Mines.

over several thousand for high-viscosity materials. Hexamethyldisiloxane is the lowest molecular weight species in the series and has the greatest vapor pressure. Silicone fluids have molecular weights from 300 to 1,000. More viscous liquids used in antifoam preparations range from 40,000 to 100,000 molecular weight units or 1,000 to 30,000 centistokes (cs) viscosity. Polishes and lubricants range from 100, to 5,000 cs viscosity, and gums used in rubber compounding are cross-linked to form the largest and stiffest compounds. The vapor pressure for the polymer system and the concentration of silicone vapor saturated in air is greater than 0.01 ppb for molecular weights of 977 or less. Commercial preparations of high-molecular-weight silicones are not pure compounds; molecular weights quoted are a number average, and some systems may contain small quantities of low-molecular-weight species which may be expected to volatilize during use. Commercial specifications for high-viscosity systems state less than 2% loss in weight on volatilization at 200 °C for 48 hours.(9)* This weight loss may consist of volatile silicone species or some entrained moisture. Silicone compounds in commercial preparations may be composed of silicones other than the dimethyl siloxane system replacing some of the methyl groups with phenyl, hydrogen, or halogen atoms, resulting in compounds with greater volatility and increased reactivity.

Silicone rubbers are composed of high-molecular-weight cross-linked systems and would be considered nonvolatile in finished form. However, if the rubber were to contain unreacted monomer or partially reacted short polymers, or if the rubber is of the room temperature volatilization (RTV) type and is still fluid and not completely set, then volatile silicone compounds will be present and in sufficient quantity could form poisons for catalytic sensors. RTV rubbers liberate acetic acid when reacting with atmospheric moisture to form reactive Si-OH groups. Systems in common use may involve reaction of methyl triacetoxy silane monomer with poly dimethyl siloxane α,w diols to form a cross-linked product.

The boiling point of methyl triacetoxy silane is 90° at 9 mm Hg pressure. An estimate of vapor pressure at room temperature is 1.8 mm Hg or 2,400 ppm concentration in saturated air. Until RTV silicone is fully polymerized, this compound may poison catalytic methane sensors.

Silicon plastics, resins, and two-component rubbers that have been heat-cured would be expected to have negligible volatile silicone concentrations. The vapor pressure of low-molecular-weight silicone linear compounds may be expressed by the following equation dependent on temperature and polymer length (number of silicon atoms):

$$\text{Log } p = 6.59 + 0.47n - \frac{972}{T} - \frac{392.2n}{T} \quad (10)$$

where p = vapor pressure, mm Hg
t = temperature absolute, °K
and n = total number of silicon atoms in molecule.
The constants were recalculated from more recent data.(8)

* Numbers in parenthesis refer to References at the end of the paper.

The saturated silicone concentration in air is calculated from the vapor pressure

$$C = \frac{P_s}{P} \times 10^6$$

where C = ppm silicone
P_s = silicone partial pressure, mm HG
and P = total atmospheric

EXPERIMENTAL WORK

A summary of previous work on effects of poisons on catalytic methane sensors has been presented in a series of papers by Jack Firth and Allan Jones of SMRE, Great Britain. (1)(7)(14) Unprotected and charcoal-protected sensors were poisoned with hexamethyl disiloxane (HMDS) in air-CH_4 mixtures, and loss of sensitivity of the palladium sensor to methane and other combustible gases was studied. The results of the test of poisoning of sensors by HMDS are summarized in table 3. The table includes poison effects noted for dimethylsiloxane polymer and triethyl phosphate as well. The times for poisoning reported were from 10 seconds to 10 minutes for losses in response from 12 to 50%. The shorter poisoning times were measured using direct impingement of reactive gases on the portable catalytic methane-sensing unit. The longer times noted were for poisoning effect of machine-mounted sensors which incorporate stainless steel screens and heavy porous sintered metal filters for mechanical protection and flame safety (fig. 1); these times reflect the time for diffusion of silicone vapor through the diffusion barriers and adsorption of poison on the porous metal surface. The use of activated charcoal for adsorbing the poisons before contact with the sensors was extremely effective, increasing life of the sensors by 240 times.

In this study two combustible gas detectors, one with palladium and one with platinum sensors, were tested for susceptibility to silicone poisoning with HMDS and with RTV silicone rubber fluids.

The poisoning of the catalytic methane sensors was monitored by observing the sensor response to methane during administration of a silicone-air mixture. The catalytic sensors were mounted in the top of an acrylic cylinder for initial tests and housed in an aluminum cylinder in later work. The test apparatus is represented in figure 2. The methane test gas was diluted to 2% (and 1%) concentration in a mixing tube; concentration was measured by periodic readings from a Riken methanometer by gas refractive index measurement. The methane concentration was intermittently varied from 2% concentration to 0% concentration by control of a solenoid valve run by a mechanical timer. Pulse times ranging from 6 minutes air and 6 minutes 2% methane to 50 minutes air and 10 minutes 2% methane were used in testing. The gas samples were purified by passage through an activated charcoal filter before introduction into the silicone vapor generation chamber.

Silicone vapor was generated by bubbling on air-CH_4 mixture through a fritted disk immersed in HMDS cooled to 0°C by ice in a Dewar vacuum container. The slicon vapor concentration was formed by using a fixed ratio of bypass air to airflow through the silicon bubbler and was calculated as follows:

$$C \text{ Silicone in air-}CH_4 \text{ mix} = \frac{P_s}{P} \frac{V_{\text{silicone flow-air}} \times 10^6}{V_{\text{total flow}}}$$

where C = concentration in ppm silicone
P_s = 8.6 mm Hg for HMDS at 0°C
P = total air pressure, 750 mm Hg normal
V_{silicone} = air-CH_4 mix flow rate through silicone bubbler, ml/min
and $V_{\text{total flow}}$ = total air-CH_4 mix flowrate, ml/min.

Total flow rates were measured on a wet-test volume meter, noting volume measured in a given time period. Silicone vapor generator airflow was measured using a standard soap bubble meter measuring time of bubble transit through a calibrated volume. The mixed air-silicone-CH_4 flows were introduced into the test chamber containing the catalytic methane sensors with methane response measured on a strip chart recorder. Determination of the silicone vapor concentration was initially performed by absorbing the silicone in 100 ml of methyl isobutyl ketone (MIBK) using two bubbler absorption columns in series. For runs of 30-minute duration, the second absorption column contained 1/7 the concentration of the first, and handling, cleaning, and refilling the scrubbers was time consuming. Activated-charcoal tubes containing CAL Calgon charcoal were used to monitor silicone vapor composition in test gas in later experiments. All of the silicone adsorbed was confined to the first 1-cm bed tube; HMDS concentration was 54 ppm, and the solid tubes were convenient to use.

Adsorption experiments were conducted with very low concentrations of HMDS in air by the use of diffusion tubes (fig. 3). HMDS contained in a 200-ml flask was connected to an upper flask with glass tubing of fixed diameter and length. The upper flask had inlet and outlet tubes for diluent airflow. HMDS vapor moved from the lower flask to the upper flask through the tube by diffusion. The vapor pressure of the HMDS was determined by holding the flask at a fixed temperature, either 0°C or -63°C, by external refrigeration of fluid in a Dewar vacuum container.(15) It was found that gas flow through the upper flask must be constant since oscillation of pressure caused by air from a diaphragm pump increased the rate of material transport by convection many times over the rate calculated for diffusion. The concentration of silicone in air produced by use of the diffusion tube was monitored by adsorbing silicone on activated charcoal in a tube, and analysis for adsorbed silicone was conducted after the test.

Analysis of the MIBK absorption liquid or of the activated charcoal tubes after extraction of adsorbed silicone (HMDS) by MIBK was performed using atomic absorption (AA) spectrometric measurement. The silicone solution in MIBK was aspirated into a nitrous oxide-acetylene flame at high flow rate, and the silicon optical absorption was determined at 2516 A° using a 2 A° slit width on a Perkin-Elmer Model 503 unit. Standard solutions were made by dilution of a weighed HMDS solution in MIBK. Standard analyses were also made with a solid octaphenyl cyclotetra siloxane weighed sample dissolved in MIBK.[13] Standard curves constructed with experimental optical density plotted versus weight-percent silicone were linear.

Hexamethyl disiloxane was chosen as catalyst poison for this test since it could be obtained in pure form; it is very volatile, boiling at 100°C, and since it is a small molecule, it is not as easily adsorbed on charcoal as larger molecules. In addition to experiments with HMDS, tests with both Dow Corning and General Electric RTV silicone rubber fluids were conducted.

Sensor poison tests with RTV silicone rubber fluid (before solid formation) were performed by extruding 1-1/4-inch lengths of silicone at each end of a polyethylene 3/4-inch-OD tube. The air-2% CH_4 mixture was passed through the tube and over the sensor on test. Vapor pressure of silicone in RTV rubber fluid was measured by flowing a known volume of air through sample tubes with 1-1/4-inch lengths of RTV silicone rubber fluid and adsorbing the vaporized silicone onto a CAL charcoal bed 1 cm thick in another tube. Thirty-minute runs were conducted at flow rates from 1 to 1.1 l/min with fresh RTV silicone added at 15-minute intervals. The charcoal test samples were analyzed for silicone content after extraction of adsorbed silicone with MIBK by the previously described AA technique.

DISCUSSION

The ability of silicone compounds to poison catalytic methane sensors depends on the volatility of the silicone and on transport of the silicone-air mixture by air currents to the sensor. Because of ventilation requirements in coal mines--3,000 cfm airflow minimum into the working area--formation of 5 ppm silicone vapor concentration would require vaporization and mixing of 3 grams of silicone compound per minute. Low-molecular-weight silicone compounds in fluids or as impurities in high-molecular-weight viscous systems may yield significant silicone volatile species from commercial products or by depolymerization during use. Only by tests of the poisoning ability of commercial mixtures can the actual volatile silicone content or ability to poison a sensor be ascertained.

Initial experiments in poisoning of catalytic methane sensors were performed with HMDS as poisoning agent at 54 ppm concentration in air and in air-2% CH_4 mixtures. Platinum and palladium sensor methane monitors were tested with and without CAL charcoal over the sensor to remove silicone vapor by adsorption. The time response for methane indication during poisoning is presented in figure 4. A 10% loss in response was obtained for both sensors in 15 minutes without protection. With a 2-mm thickness of CAL charcoal protecting the sensor, runs were obtained for 10% loss in response in 42 hours and 144 hours for palladium- and platinum-type sensors, respectively. Although the silicone concentration in the air inlet to the test chamber was 54 ppm, initial values of 18 to 22 ppm silicone obtained from the chamber exit air show significant adsorption by the chamber contents: four sensors, two coatings of CAL charcoal, four sintered-metal flame suppressors, and the chamber walls.

Experiments on platinum sensor methane monitors were conducted at very low levels of HMDS generated by a diffusion tube held at -63°C by mechanical refrigeration. The average assault concentration of HMDS during test was 5.5 ppb. The concentration of HMDS ranged from 16 ppb initially to 3.77 ppb in the final test, and 4,000- to 7,900-minute tests were made to obtain enough HMDS on each series of CAL charcoal sample tubes for AA analysis. The response of the monitors was measured in series for the HMDS measurement, and breakthrough of the low concentration of HMDS vapor was noted on two tests for these very long trapping times.

Silicone vapor poisoning of catalytic methane sensors may be delayed by adsorption on activated charcoal surrounding the sensor elements. The thicker the layer of charcoal, the longer the sensor can operate without being poisoned. The thicker layers of charcoal cause a longer sensor response time. Practical methane sensors require rapid response as well as protection from poison, so there is a limit to the thickness or adsorbent material that can be used. For a response time of 1 minute or less the maximum adsorption layer thickness allowed is 2mm for this sensor.

Tests previously conducted used times of sensor response loss in methane as a measure of protection offered by a given adsorbent. Possible protection effect may also be assessed for a material by measuring the amount of adsorption of silicone vapor on a known amount of absorbent. The adsorption isotherms for HMDS on various charcoal substitutes were measured by equilibrating the test substrate with known concentrations of HMDS in air, and the amount of HMDS adsorbed was measured after extraction into MIBK with analysis by AA spectrometry.

An adsorption isotherm for HMDS on CAL charcoal is presented in figure 5. The amount of HMDS adsorbed can be determined from this figure for known concentrations of silicone vapor in air. Although more HMDS is adsorbed at high concentration, sufficient HMDS is still adsorbed at low concentrations to protect sensors. Saturated (77°F) values of HMDS adsorption for various carbon materials can be used to screen possible adsorbents, and equilibrium values are listed in table 4 for dry air-HMDS systems. In terms of total quantity of HMDS adsorbed, the following preference in substrate is indicated: CAL, PCB, MDA cloth, CDE cloth (20 cal/gm heat of wetting with benzene).

CONCLUSIONS

Catalytic methane sensors are poisoned rapidly by volatile silicone materials, confirming early work performed by SMRE, Great Britain, and manufacturers' literature. Once a catalytic methane sensor in use in a mine is poisoned by silicones or related poisons, the instrument may incorrectly indicate safe methane concentrations when possibly high concentrations, in explosive range, are present. Silicone compounds may be incorporated in transformer coolants, hydraulic or brake fluids, lubricants, sealants, waxes, polishes, and other mixtures available or intended for use in mines.

Volatile silicone compounds or other poisonous agents should not be introduced into mine atmospheres when methane detectors or monitors are in use, unless appropriate steps are taken to assure that harmful monitor or detector poisoning does not occur. Should it be suspected that such a silicone has been introduced into a mine immediate steps should be taken to test each methane detector or monitor to ascertain performance accuracy using known methane-air[12] mixtures. Silicone-containing materials may be identified by labeled contents listing, or by chemical analysis for silicon by wet chemical, emission spectroscopic, or AA spectrometric techniques. The presence of volatile silicone compounds in a mine atmosphere may be tested by adsorption of the silicone vapor on a CAL activated charcoal 1-cm-thick trap using a permissible air sample pump to deliver a 1- to 4-l/min flow rate. The adsorbed silicone can be eluted into MIBK and analyzed for silicone content by the AA technique described.

Protection of catalytic methane sensors by adsorption of poison before contact with the sensor element is practical for low concentration levels of silicones. Thin layers of adsorbent on diffusion heads are necessary for protection so that minimum effect will be obtained on response times.

The Mine Safety and Health Commission of the European communities has proposed the use of a layer of activated charcoal cloth covering the sensor element for protection of handheld methane detectors. The use of such protection may become mandatory in Great Britain, France, and Germany, for example, requiring demonstration of protection of handheld methane detectors against 5 ppm HMDS concentrations in air-CH_4 mixtures for a test time of 1 to 2 hours without loss in response greater than 10%. Regular maintenance of such units is recommended, including replacement of the charcoal cloth during maintenance checks every 2 to 4 weeks.

It is recommended that manufacturers of catalytic methane sensors, recognizing the increased importance of protection from poisoning with the proposed use of new mining compounds, would market such protective devices to prevent poisoning of the sensors by attachment of an adsorbent to the present diffusion-type methane monitor or detector. Manufacturers should recommend also a maintenance program with periodic replacement of adsorbing materials to assure device reliability.

This study has focused attention only on the use of adsorbents to remove poisons from contact with catalytic methane sensors. It is also possible that suitable oxidation catalysts may be developed to react with poisons to form nonpoisonous products that will give a longer protective life and require less frequent replacement.

ACKNOWLEDGEMENTS

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Table 1.--Catalytic methane sensor poison classes

Compound	Structure	Reference
Organic Halides.....	R-X X = Cl, Br, I	1, 4, 5, 6, 7
Siliconc.....	$R-\left(\begin{matrix}CH_3\\Si-O\\CH_3\end{matrix}\right)=R$	1, 2, 3, 4, 5, 6
Phosphate.....	$R-OP-(O\ R)_3$	1, 6
Metal.....	$Pb-(C_2H_5)_4$	1, 3, 4, 5, 6
Sulfide.....	R_2-S	4, 5, 6
Other.....	H_2SO_4 (mist) HCN Cl_2 SO_2	6 7

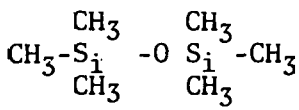
TABLE 3.--Catalytic methane sensor poisoning effects

Material	Life	Life, charcoal protected	% Response	Reference
MS 200/10 silicone oil.....	5 min	20 hr	20/2% CH_4	SMRE data
Triethyl phosphate.....	5 min	20 hr	15/2% CH_4	do.
Hexamethyl siloxane (HMDS), 26 ppm.....	26 sec	22 hr	33/1% CH_4	do.
HMDS, 40 ppm.....	10,30 sec	-	50	14
HMDS, 54 ppm.....	10 min* 8 min**	 167 hr* 99 hr**	12 12 25 50	Present work

* Pd sensor
** Pt sensor

TABLE 2.--Properties of linear dimethyl siloxane polymers

Silicon atoms N	Viscosity, cs	Molecular weight	Vapor pressure, mm Hg at 25°C	Equilibrium concentration air, ppm at 25°C
2	0.65	162	37.00	48,700
4	1.50	310	0.96	1,260
6	3.0	439	0.04	52
13	11.	977	10^{-8}	.013
130	100.	10,000		
400	1,000.	40,000		
1,000	12,500	90,000		
1,200	30,000	100,000		
1,900	500,000	143,000		

N = 2  Hexamethyldisiloxane

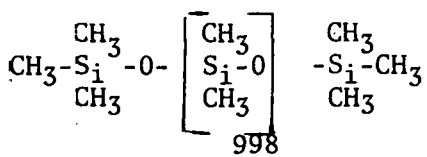
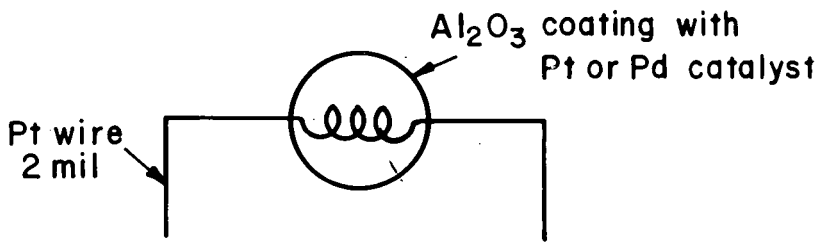
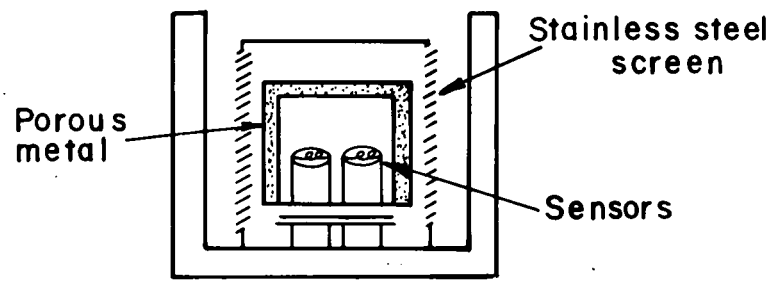
N = 1000 

TABLE 4.--Equilibrium adsorption of HMDS by charcoal substrates, 25 to 27 °C

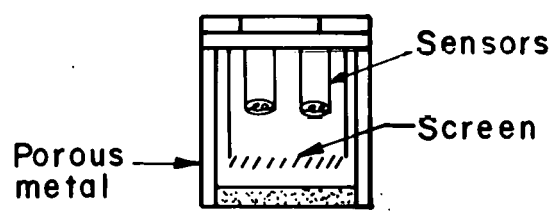
Substrate	HMDS adsorbed per g charcoal, mg
Granular:	
CAL, Calgon	331, 347
BPL, Calgon	311
PCB, Calgon	340
WVG Nuchar, Westvaco	288
WVH Nuchar, Westvaco	274
PC1333, Barneby-Cheney	159
Foam carbon:	
RVC 4X5 80A, Chemotronics Intl	33
Cloth:	
CC-2, MDA Scientific	265, 271
Chemical Defense Establishment, Great Britain:	
1	21
5	18
10	157
20	203



Catalytic sensor



Pt sensor head



Pd sensor head

PGH-78
4/14

FIGURE 1.--Catalytic Methane Sensors

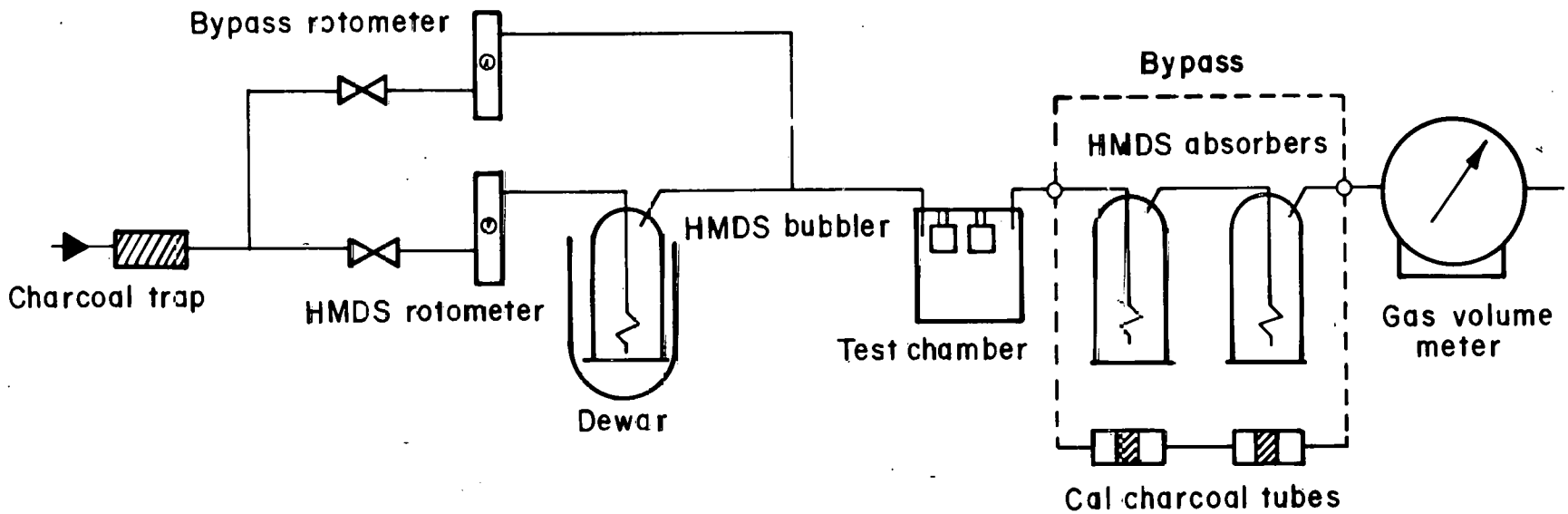
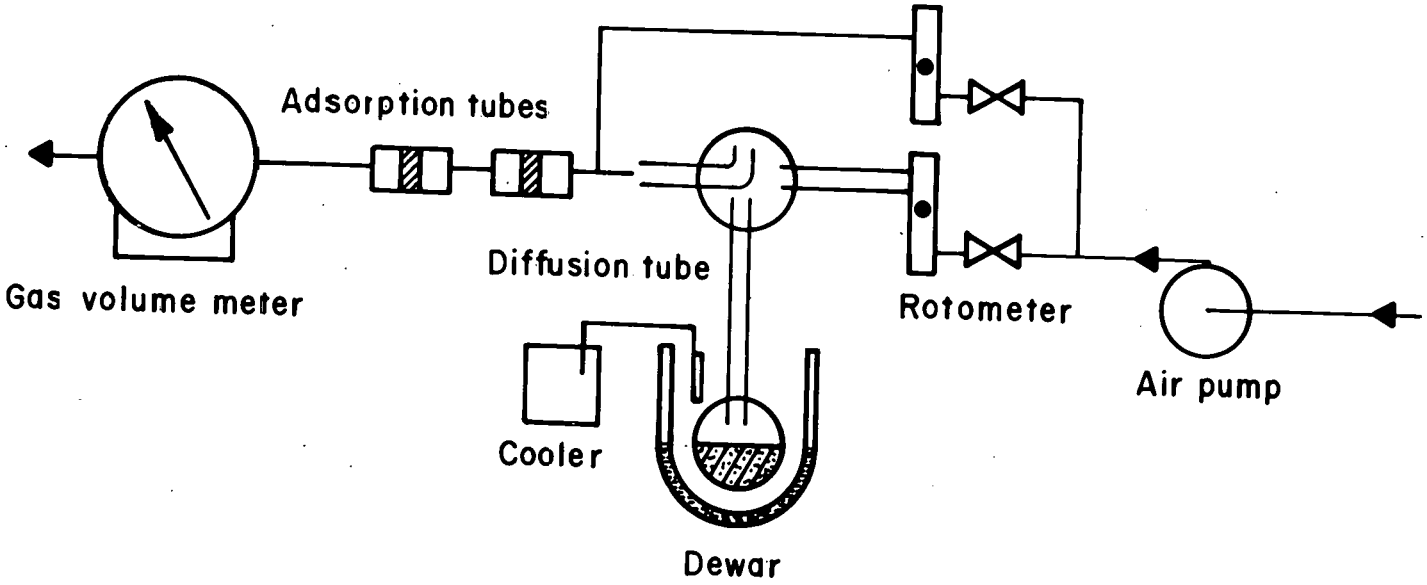


FIGURE 2.--Catalytic Sensor Test Apparatus

PGH-78
408



PGH-78
413

FIGURE 3.--Diffusion Tube Apparatus.

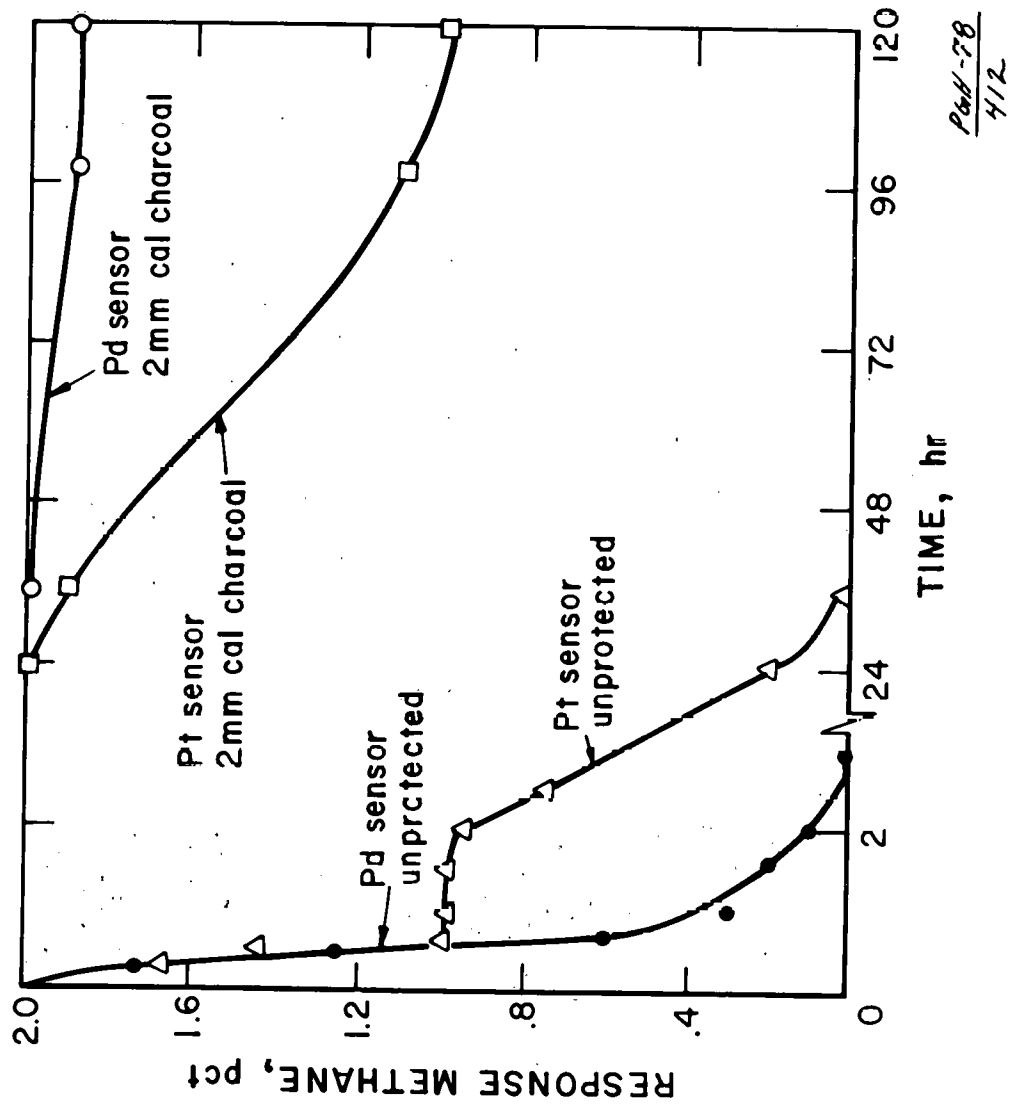


FIGURE 4.--54 ppm Hexamethyldisiloxane Poisoning of Catalytic Methane Sensors

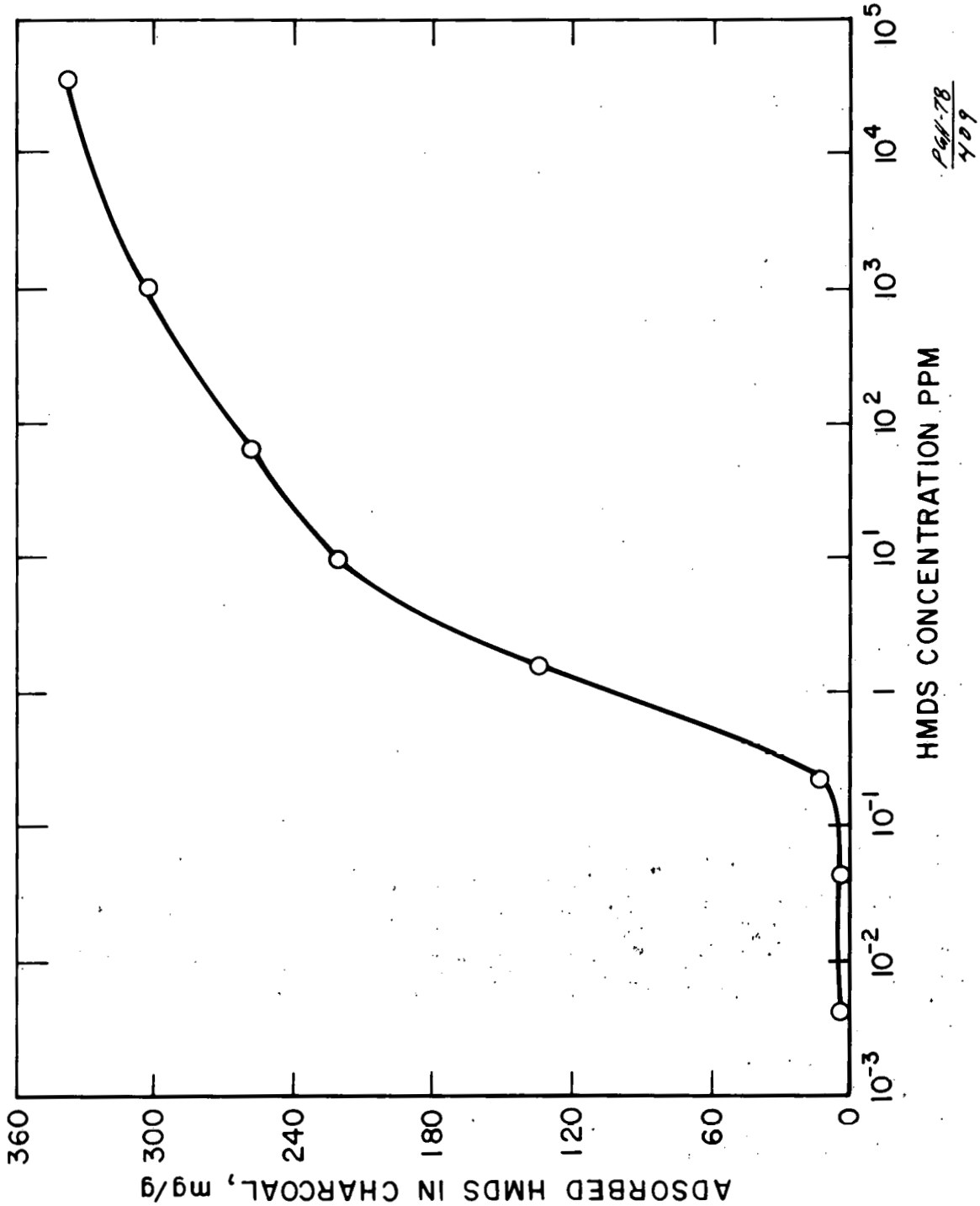


FIGURE 5.--Adsorption Isotherm Hexamethyldisiloxane. 77 °F

SIMSLIN II - A PORTABLE AIRBORNE DUST MEASURING INSTRUMENT EMPLOYING A LIGHT SCATTERING TECHNIQUE

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ABSTRACT

The paper describes SIMSLIN II (Safety in Mines Scattered Light Instrument), an instrument designed to monitor continuously the respirable fraction of airborne dust by means of a light scattering technique. The instrument is portable, intrinsically safe and rugged, and it is particularly suitable for use in industrial environments such as mines, quarries and factories. Details of the mechanical and optical construction are given, along with important features of the electronic circuits. SIMSLIN II's performance in the laboratory and in a coal mine location is described.

INTRODUCTION

The Safety in Mines Scattered Light Instrument, SIMSLIN II, is a portable unit designed to monitor continuously the respirable fraction of airborne dust by means of a light scattering technique. Light scattering methods of dust measurement provide an instantaneous indication of the airborne dust concentration and do not change the state of the dust cloud. The technique relies upon the fact that the intensity of the light scattered by airborne dust particles passing through a light beam is directly related to the instantaneous concentration of the particles. Such a measurement of intensity may be difficult to interpret, if, as is often the case, the airborne cloud consists of particles with a range of sizes, shapes and refractive indices and the composition of the cloud varies randomly with time.

In SIMSLIN II, by measuring the total intensity of the light scattered at small angles to the forward direction of the incident beam, variations in response due to particle shape and refractive index are largely eliminated and the total scatter varies in proportion to the projected surface area of the particles. For some common materials (coal, limestone, quartz, etc) the constant of proportionality approximates to the density of the material and thus the instrument may be calibrated to give an effective mass response (Seaney, 1973). Since the instrument really responds to the projected surface area of the dust (ie to particle diameter squared rather than cubed) the calibration will vary slightly from dust cloud to dust cloud especially when smoke or heavily aggregated particles are present.

It is intended that SIMSLIN should complement rather than in any way replace the gravimetric type of sampler by providing information about the short term fluctuations in dust concentration, enabling dust sources to be identified and the effectiveness of dust suppression measures to be assessed.

SIMSLIN II is a second generation instrument and is the result of extensive research and field trials with its predecessor (Seaney, Halpin and Maguire, 1972; Seaney and Hodson, 1974). The unit is particularly rugged and is eminently suitable for use in coal mines and similar environments. The design offers comprehensive facilities including digital display of the dust concentration at 1 or 15 second intervals, automatic averaging of the data over 15 minute periods, and the option of programmed operation. An interchangeable solid state memory unit (SIMSTOR 1) stores all readings for later use and thus permits a detailed analysis of the fluctuations in instantaneous dust concentration.

The instrument is certified as satisfying the safety requirements of the British Standard BS 1259:1958 Class 1 (British Standard Institution, 1958) and is therefore suitable for underground use in British coal mines.

INSTRUMENT DESCRIPTION

The instrument, which is shown diagrammatically in Fig 1, has been designed to withstand reasonable shock loads. In particular, components within the photometer are held in place by compressed O-rings and the whole photometer is held in place by sponge rubber. The stainless steel case measures 410 mm by 110 mm by 150 mm and it weighs complete with the data storage system and batteries approximately 7 kg.

PARTICLE SIZE SELECTION

SIMSLIN II has been designed with a front cross-sectional profile very similar to that of the Mining Research and Development Establishment (MRDE) 115 Multishift dust sampler, and with the same elutriation system. Its airflow characteristics are therefore the same as those already determined for the MRDE 115 which has been subjected to extensive trials by the National Coal Board to establish agreement with the MRE 113A, ie the statutory gravimetric dust sampler used in British coal mines. The elutriator is of the single parallel plate type and has a penetration of 50 per cent for 5 μm and zero penetration for 7·1 μm unit density spheres. It satisfies the penetration characteristics specified for respirable dust measurement by the British Medical Council and was recommended at the Johannesburg Conference (Orenstein, 1960).

THE AIRFLOW SYSTEM

A small rotary vane pump draws dusty air at a flow rate of 0·625 litres/minute through the parallel plate elutriator, to remove non-respirable dust, through a light beam and then through a filter, so that the respirable dust that produces the scattered light reading can be weighed and analysed later. This latter feature provides a simple and accurate means of calibration,

since the instrument also produces an average light scatter reading. For ease of handling the filter is mounted in a cassette which is accessible through a small hatch in the base of the instrument.

After the dusty air has passed through the main filter a fraction of the flow is directed through a supplementary filter and the resultant clean air is recirculated to form a sheath around the dusty air in the photometer scattering zone. This ensures that the dusty air is kept in a well defined column and prevents dust from settling out in the photometer. The diameter of the dust column is about 4 mm and is essentially defined by the diameter of the smaller entry tube. A flow meter is incorporated in the exhaust line and the air flow through the instrument can be adjusted by a control on the rear panel of the instrument.

When an instrument zero is required blocking off the exhaust aperture will cause the clean air recirculation to purge the dust from the light scattering region. Depending on the dust concentration this may take several minutes but once the reading has stabilised a true zero is produced.

THE OPTICAL SYSTEM

A small near infra-red source is mounted on one of the end plates of the photometer and the radiation from this is collected and focussed into a slightly converging beam by the lens. The beam is directed along the axis of the photometer towards the column of dusty air in the scattering region. The intersection between the beam and this region defines the scattering volume. At the intersection, the primary beam has a cross sectional area of about 3 mm^2 and the total scattering volume is about 10 mm^3. A small fraction (typically 10^{-6}) of the incident radiation in the primary beam is scattered by the dust in this volume.

The dust scattered radiation is collected over an angular range of 12° to 20° to the forward direction by a lens system and is focussed onto the central detector. The dust scattered only a tiny fraction of the incident radiation, a central light trap prevents the primary beam from reaching the central detector and swamping the scattered light signal. The angles over which the scattered light is collected are set by the outer diameter of the central light trap and the annulus. It has been empirically determined that the scatter in this region is largely independent of particle composition for common industrial dust clouds. However, the design is sufficiently flexible to allow changes in these angles for special requirements.

The unscattered primary beam enters the light trap and passes via a lossy light pipe to a second, identical, detector. The resulting reference signal is proportional to the total amount of radiation incident on the scattering volume, and is used via a control loop to compensate for any changes in the effective light output of the source, whether they be through ageing, temperature effects, or dust settling on the system optics.

An important feature of the photometer, besides its small size and weight, is the very low background radiation level. This is the scattering signal

produced in the absence of any dust and in the present design is typically equivalent to 1·5 mg/m^3. This is a stable level and is backed off electrically when the instrument is zeroed.

The light source used in the photometer is a small GaAs injection diode laser, which emits invisible radiation at a wavelength of 0·9 μm in the near infra-red region. It has a very small emitting area, typically 2 μm x 75 μm, which means that the radiation can be collected efficiently and collimated into a well defined beam by means of short focal length optics: this minimizes the length of the photometer.

The laser is operated in the pulsed mode and has a single pulse accessible emission level of 1·2 x 10^{-6} Joules. This radiation level marginally exceeds the Accessible Emission Limit (AEL) laid out in the British Standards Institution standard guide to the Protection of Personnel against Hazards from Laser Radiation (BS 4803 : 1972). However, once enclosed in the photometer no reasonable human access to the laser radiation is possible and therefore the instrument is provisionally classified and labelled as containing a "Class 1/3 Laser System" in accordance with the above draft standard specification.

THE ELECTRONIC SYSTEM

A block diagram of the electronic system is shown in Fig 2. Extensive use of a complementary MOS technology has been made to minimise power consumption.

POWER SUPPLIES

SIMSLIN II is powered by two modified Varta (DEAC) type 5/900 nickel cadmium rechargeable batteries (nominal 6 volts), which incorporate one ohm internal resistors to satisfy intrinsic safety demands. They have a capacity of 900 mAh and will allow the instrument to be used continuously for up to 25 hours. A circuit in the instrument detects when the battery voltage has fallen to a preset level, and automatically switches off the pump and all measuring functions and causes a 'Low Battery' warning light to be illuminated. A battery charging socket is provided.

The pump unit uses by far the biggest proportion of the battery's power. To ensure maximum operating time from the batteries, the pump is supplied by both batteries in series. Since the pump motor only requires three volts to provide the necessary flow rate, a switching regulator is used to make the most efficient use of the available power.

The laser diode must be operated from a voltage supply of between 20 and 30 volts. This is generated from the low voltage supply by means of a ringing choke circuit which is totally encapsulated to meet intrinsic safety requirements.

LASER SOURCE CIRCUIT

The laser diode is operated in a pulse mode with a duty cycle not exceeding 0·1 per cent. It is driven at a repetition frequency of 5 KHz by constant amplitude pulses 160 ns wide. These pulses are supplied in bursts, and the

duration of each burst is controlled by the primary beam reference signal, which keeps the mean power output of the laser, and thus the instruments sensitivity, constant. If a fault occurs so that it is impossible for the control circuit to maintain this calibration a warning light appears on the display panel and the digital display and output are disabled.

SCATTERED LIGHT DETECTION CIRCUIT

The radiation scattered by the dust is collected by a lens system and focussed onto a detector identical to that used to monitor the primary beam intensity. The output from this detector is amplified and averaged by a leaky integrator circuit with a time constant of 30 seconds, to average out any short term fluctuations. Independent zero and sensitivity controls are provided behind a tamper proof panel.

The dust concentration signal is available either as a buffered analogue output or digitally. The analogue output is intrinsically safe and is designed for connection to a high impedance chart recorder. The digital output is available serially in the form of a number of clock pulses. It is of the current sinking variety and connection to it may be made underground.

OUTPUT DISPLAYS

The analogue output is converted to a digital signal by a dual slope analogue to digital converter and is shown on a liquid crystal display located on the back panel. The reading is updated every second, and covers the ranges 0 to 199·9 or 0 to 19·99 mg/m^3.

A second liquid crystal display, which duplicates the readings, is normally located on the top of the instrument and plugged into the memory housing. It may also be sited at a distance from the instrument, connected by an extension lead. This slave display is particularly useful when dust sources are being located or when SIMSLIN is located in a zone in which the operator cannot be present eg a duct.

The digital output from the analogue to digital converter is also fed to a digital averager circuit which calculates the running mean dust concentration. This mean value is displayed on demand on a light emitting diode numeric display, and is updated every fifteen minutes. Thus at the end of the dust sampling period a mean dust concentration figure is available for that period of time.

PROGRAMMER

A programmer has been built into the instrument to allow automatic operation. A switch is located on the back panel to allow over-riding of the programmer. The programme selection switches are located on an internal circuit board and access to it is not permitted underground.

In the present instruments, the programmer has eight switches each of which corresponds to one hour of operating time. Thus during the first eight hours after switching on, the instrument can be programmed to be off

or on for each individual hour. The cycle of programming is twenty four hours. After the first programmed eight hours the instrument will be switched off for sixteen hours and then the programme will be repeated for the next eight hours and so on.

DATA STORAGE SYSTEM (SIMSTOR 1)

A solid state memory unit, manufactured in the form of a self-contained plug-in cartridge, is normally fitted to the instrument. It contains its own batteries and will retain stored data for several months. At the present time, one dust measurement reading is transferred to memory every fifteen seconds via an intrinsically safe interface. The memory has a capacity of 3000 readings which is equivalent to about 12 hours of recording time. The maximum value that may be stored is 50 mg/m^3 on one range with a resolution of ± 5% of the stored value or ± 0·1 whichever is the greater and a maximum value of 5 mg/m^3 on the other range with a resolution of ± 5% or ± 0·1. The memory units may be interchanged underground and must be taken to the surface for decoding, clearing and resetting.

CALIBRATION

The reading displayed by the instrument is dependent upon the amount of light scattered by the dust and the effective gain of the electronic system. Since the amount of light scattered is dependent to some extent upon type of material, particle size distribution and degree of aggregation, it is recommended that the instrument is calibrated with the dust cloud to be measured. Since the instrument takes a gravimetric sample and gives a mean dust concentration from the light scattered readings, the gain control on the instrument can be adjusted to make the two correspond.

LABORATORY DUST CONCENTRATION MEASUREMENTS

A series of experiments has been carried out in the laboratory to determine the relationship between the gravimetric results obtained from SIMSLIN's internal filter and the scattered light reading obtained from the averager circuit. This has been done for three types of dust: quartz, a bituminous coal and a waterproofed limestone, in the concentration range 0 to 70 mg/m^3.

The dust was dispersed into a box, approximately 500 mm cube, by a Wright generator (Wright, 1950). This type of dust disperser is capable of producing a wide range of reasonably constant airborne dust concentrations. The dust feed was located centrally on the top face of the box and a SIMSLIN instrument was connected to a rectangular slot in the side face of the box at a point two thirds down one side. The instrument was sampling under near still air conditions. Sampling times were adjusted so that at least two milligrams of dust were collected on the filter (Whatman type GFA).

After a given concentration had been sampled, the weight of dust on the filter was determined and then the dust was removed from the filter and dispersed ultrasonically in saline solution. Aliquots were then used to determine the particle size distribution by both optical and Coulter (Coulter, 1956) counting techniques. The size distribution of the dispersed dust remained reasonably constant throughout the experiments and were similar for all three types of dust.

The results of these experiments are shown in Fig 3. This shows that there is a one to one correspondence between the averaged scattered light reading and the gravimetric result for the quartz dust and for the bituminous coal, but the scattered light reading for limestone is some 25 per cent lower than the gravimetric result.

LABORATORY ENVIRONMENTAL TESTS

Eight pre-production instruments have been subjected to change of temperature and of humidity in an environmental chamber to determine the effect on instrument zero and sensitivity.

In tests to assess the effect on the instrument zero, the instruments were complete, with their pumps running but with their exhaust outlets blocked so that clean air was being circulated through the light scattering zone. The environmental chamber was controlled at 5°C and 20 per cent relative humidity for about three hours and the zero reading on both dust concentration ranges was observed for all instruments. The chamber temperature was then raised to 35°C and the humidity retained at 20 per cent. After a further three hours the zero readings were determined and then the relative humidity was raised to 90 per cent with the temperature at 35°C. Again three hours were allowed to elapse before the zero readings were observed.

The results of these experiments are shown in Table 1. On average zero drift with temperature is of the order of 0·01 mg/m^3 equivalent dust concentration per degree centigrade, on either range. The effect of changing from very dry conditions to very wet conditions is a maximum of 0·3 on the high range and 0·09 on the low range. This degree of stability is considered satisfactory for most situations.

For the second group of tests the instrument covers and elutriators were removed. Coal dust was dispersed in a 500 mm cube box and attached to the side of the box was a parallel plate elutriator designed to give the same size selection characteristic as the elutriator fitted to the SIMSLIN instrument but at a flow rate of eight times the designed flow rate of SIMSLIN. Large bore tubes were used to divide this flow to eight outlets using similar short lengths of tubing. These outlets were connected directly to the inlet tubes of the photometers of the eight instruments under test. The flow rate through each instrument was adjusted to be the standard value of 0·625 l/min. After several days of experiments the connecting tubes were examined: since they appeared clean it was concluded that little or no elutriation had taken place in them.

The instruments were located in an environmental chamber where the temperature was maintained at 5°C for several hours before the dust feed was applied to the instruments. Dust was then fed to the instruments for several hours after which the gravimetric sample filters were removed and the averaged scattered light reading observed. This procedure was repeated at 20°C and at 35°C.

The results of these tests are shown in Table 2, normalised to the reading at 20°C. From this table it can be seen that the worst performance, instrument f, gave a ± 14 per cent change for a change of temperature of 30°C. This is considered satisfactory when it is remembered that during most practical sampling periods the temperature is unlikely to change more than a few degrees and that the measurements associated with carrying out this type of test are subject to considerable error.

UNDERGROUND TRIALS

A series of underground measurements was carried out at a colliery in the north-east of England. The longwall face was retreating and the intake and return roadways were rectangular in section, 3·7 m wide and 2·1 m high and driven in the coal seam only. Three measuring instruments were used and they were located 70 m outbye the face in the return roadway. The three instruments were suspended from a roof girder about 0·3 m apart and side-by-side, so that they were near the centre of the roadway and about 0·5 m below the roof girder.

One instrument was a SIMSLIN and the other two were standard MRE 113A gravimetric samplers. All three instruments sampled the mine air for five hours on each of four consecutive days. After each sampling period the instruments were returned to the surface, where the gravimetric, glass fibre filters were removed, the instruments were cleaned and new filters placed in position ready for the next sampling period. The exception to this routine was that the SIMSLIN gravimetric filter was left in position in the instrument after the third day, so that the dust on the filter after the fourth day had been collected during the sampling periods of both day three and day four. It was hoped by this means to improve the accuracy of the result obtained from SIMSLIN's gravimetric sample.

The results of this trial are shown in Table 3 and it can be seen that excellent agreement was obtained between the SIMSLIN scattered light readings and the gravimetric readings obtained from both MRE instruments on all four days. Good agreement was also obtained between SIMSLIN's gravimetric results and the scattered light results on days one and two, but the gravimetric sample representing days three and four was about fifteen per cent lower than the scattered light result. It should be remembered that this filter would contain just over two milligrams of dust and the loss of a very small amount of filter during the handling procedure could easily account for this discrepancy.

CONCLUSIONS

SIMSLIN II is a rugged portable instrument that will perform satisfactorily in harsh locations and in a wide range of environmental conditions. The results it gives by the light scattering method show good agreement with those obtained by gravimetric means. The instrument has sufficient stability to ensure good results over a long period.

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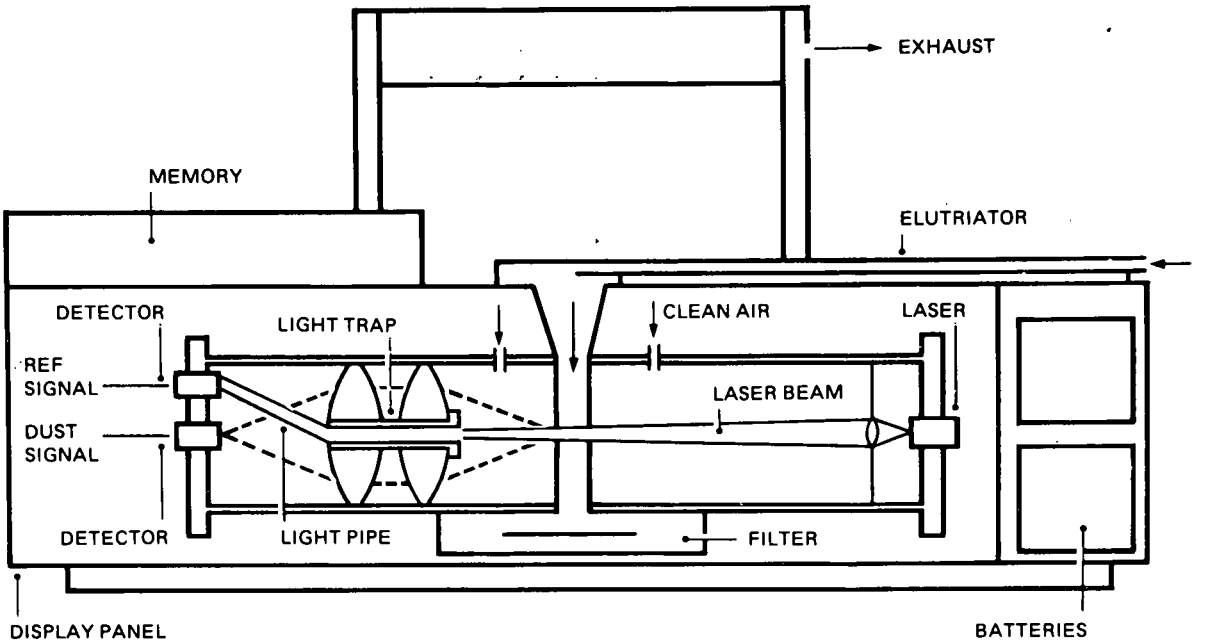

EXHAUST
MEMORY
ELUTRIATOR
DETECTOR
LIGHT TRAP
CLEAN AIR
LASER
REF SIGNAL
LASER BEAM
DUST SIGNAL
DETECTOR
LIGHT PIPE
FILTER
DISPLAY PANEL
BATTERIES

FIG.1

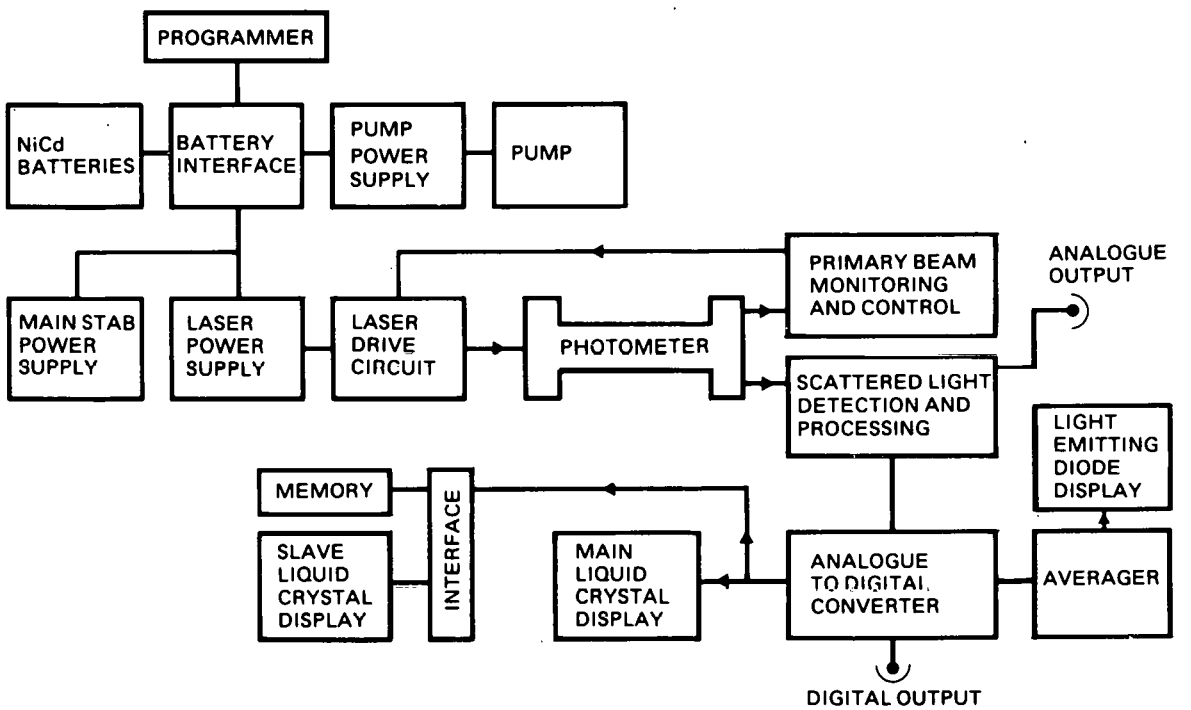

PROGRAMMER
NiCd BATTERIES
BATTERY INTERFACE
PUMP POWER SUPPLY
PUMP
MAIN STAB POWER SUPPLY
LASER POWER SUPPLY
LASER DRIVE CIRCUIT
PHOTOMETER
PRIMARY BEAM MONITORING AND CONTROL
ANALOGUE OUTPUT
SCATTERED LIGHT DETECTION AND PROCESSING
LIGHT EMITTING DIODE DISPLAY
MEMORY
SLAVE LIQUID CRYSTAL DISPLAY
INTERFACE
MAIN LIQUID CRYSTAL DISPLAY
ANALOGUE TO DIGITAL CONVERTER
AVERAGER
DIGITAL OUTPUT

FIG.2

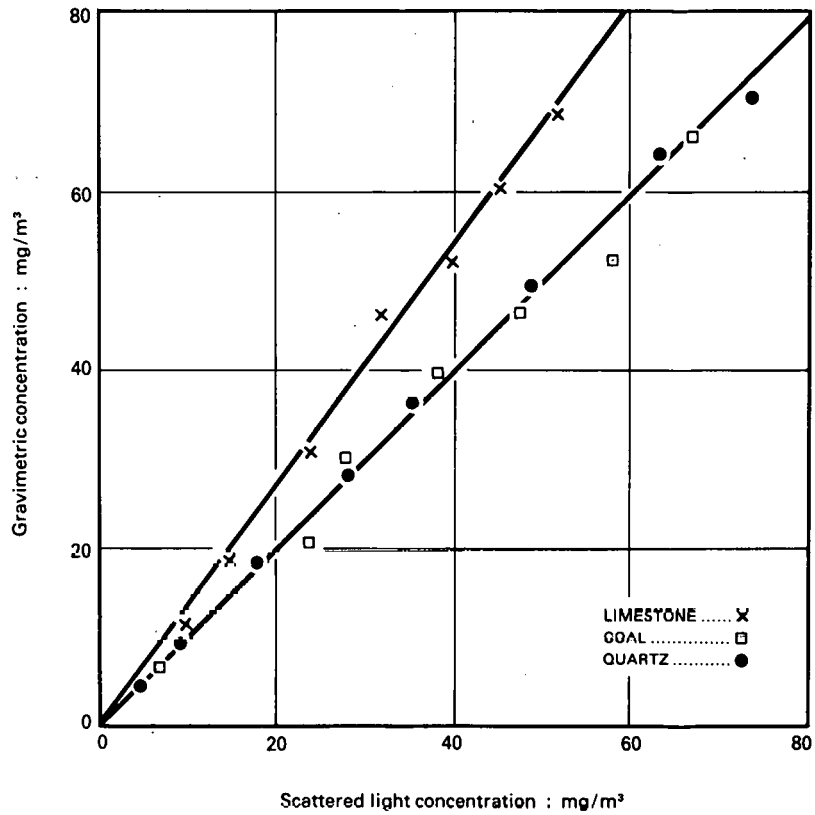


FIG.3

Table 1

The effect of change of temperature and relative humidity on the instrument's zero

	Change in zero reading (mg/m³)			
Humidity	20% R. H.		20% to 90% R. H.	
Temperature	5°C to 35°C		35°C	
Range	200 mg/m³	20 mg/m³	200 mg/m³	20 mg/m³
Instrument				
a	- 0·3	- 0·08	- 0·3	- 0·03
b	+ 0·2	+ 0·09	+ 0·3	+ 0·03
c	- 0·3	- 0·32	+ 0·1	+ 0·09
d	- 0·2	- 0·27	0·0	+ 0·02
e	- 0·2	- 0·25	+ 0·1	0·00
f	- 0·6	- 0·60	0·0	- 0·01
g	- 0·8	- 0·24	0·0	- 0·02
h	+ 0·1	+ 0·01	+ 0·2	- 0·06
Mean change	0·34	0·32	0·14	0·04

Table 2

The effect of change of temperature on the instrument's dust sensitivity

Instrument	Ratio of scattered light reading to gravimetric reading		
	5°C	20°C	35°C
a	1·10	1·00	1·04
b	0·87	1·00	1·05
c	1·09	1·00	0·91
d	1·03	1·00	1·00
e	1·00	1·00	1·11
f	1·13	1·00	0·86
g	0·97	1·00	0·91
h	1·00	1·00	1·03

Table 3

Comparison of results obtained in an underground roadway

	Concentration mg/m³			
	Day 1	Day 2	Day 3	Day 4
SIMSLIN displayed shift average	10·8	7·0	6·7	7·6
SIMSLIN gravimetric	10·5	6·6	6·3	
MRE 113A No 1 gravimetric	10·3	7·1	6·7	7·7
MRE 113A No 2 gravimetric	10·8	7·2	6·7	7·7

ARCING FAULTS ON DIRECT CURRENT TROLLEY SYSTEMS

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ABSTRACT

This paper discusses tests that have been conducted to evaluate arcing short circuit faults on direct current trolley systems. Arcing faults supplied by single and multiple power sources and limited by various impedances were studied. Data were recorded to compare various system behavior during arcing faults. The instrumentation, test procedures, electrode material and other factors affecting arcing faults are described. The results are systematically evaluated and are compared to present Federal Regulations. Areas of further study and research are also presented.

INTRODUCTION

Trolley systems are used extensively in the mining industry to transport coal and mining supplies in underground coal mines. Fires which are caused by short circuit faults in trolley systems can increase the risk of serious injury or death to those persons working underground. Since the enactment of the Federal Coal Mine Safety Act of 1952, a total of 115 fires (Figure 1) involving trolley systems were formally investigated by Health and Safety personnel of the U. S. Bureau of Mines (1952-1970) and Mining Enforcement and Safety Administration (1970-1977), now the Mine Safety and Health Administration. During this 25-year period, 38 persons were killed and 25 persons were injured by fires involving trolley systems. In the 8-year period from 1970 to 1977, a total of 22 fires occurred involving trolley systems resulting in 11 injuries and 12 fatalities. Nine fatalities were caused by one fire in 1972.

Fire hazards associated with trolley systems can be reduced by the proper installation, maintenance and protection of trolley systems. Section 75.1001, Title 30, Code of Federal Regulations, Part 75, requires that trolley wires and trolley feeder wires be provided with overcurrent protection. They are also required to be supported on insulators and not in contact with combustible materials, roof or ribs. Presently, overcurrent protection is considered adequate if the affected circuit will be properly deenergized upon the occurrence of a short circuit at any location in the trolley system. In short circuit analysis of trolley systems, a bolted fault is assumed in order to calculate the current(s) that would flow at any given location.

The mining industry has had little information on the relationship between bolted fault current and the expected fault current during an arcing fault. Arcing faults have been called high-resistance faults, and the value of arcing fault current was sometimes assumed to decrease to values which would not be detected by conventional overcurrent devices. Thus, the cause of some fires associated with trolley systems has been attributed to such high-resistance faults. Also, the cause of some trolley fires has been attributed to low-current faults due to insulation failures such as contaminated insulators. Both arcing faults and insulation failures are conditions which are undesirable and are capable of igniting coal and other combustible materials. Insulation failures are usually long-term conditions and fires resulting from such failures can be prevented by proper design and maintenance of trolley systems. Thus, they will not be treated in this paper. Arcing faults on the other hand, can occur instantaneously as a result of falls of roof, dislodged steel roof supports, haulage vehicle derailments, etc.

In order to obtain additional information on the relationship and characteristics of arcing faults, Mine Safety and Health Administration (MSHA) personnel performed various tests to simulate arcing faults on trolley systems. One of the main reasons for performing these tests was to determine the relationship between the current during bolted fault conditions and the expected current during arcing faults for known circuit parameters. Once this relationship is determined, the present short curcuit analysis method for trolley systems can be modified to include the effect of arcing faults.

DATA ON COAL MINE FIRES RELATED TO TROLLEY CIRCUITS

INVESTIGATED 1953-1977

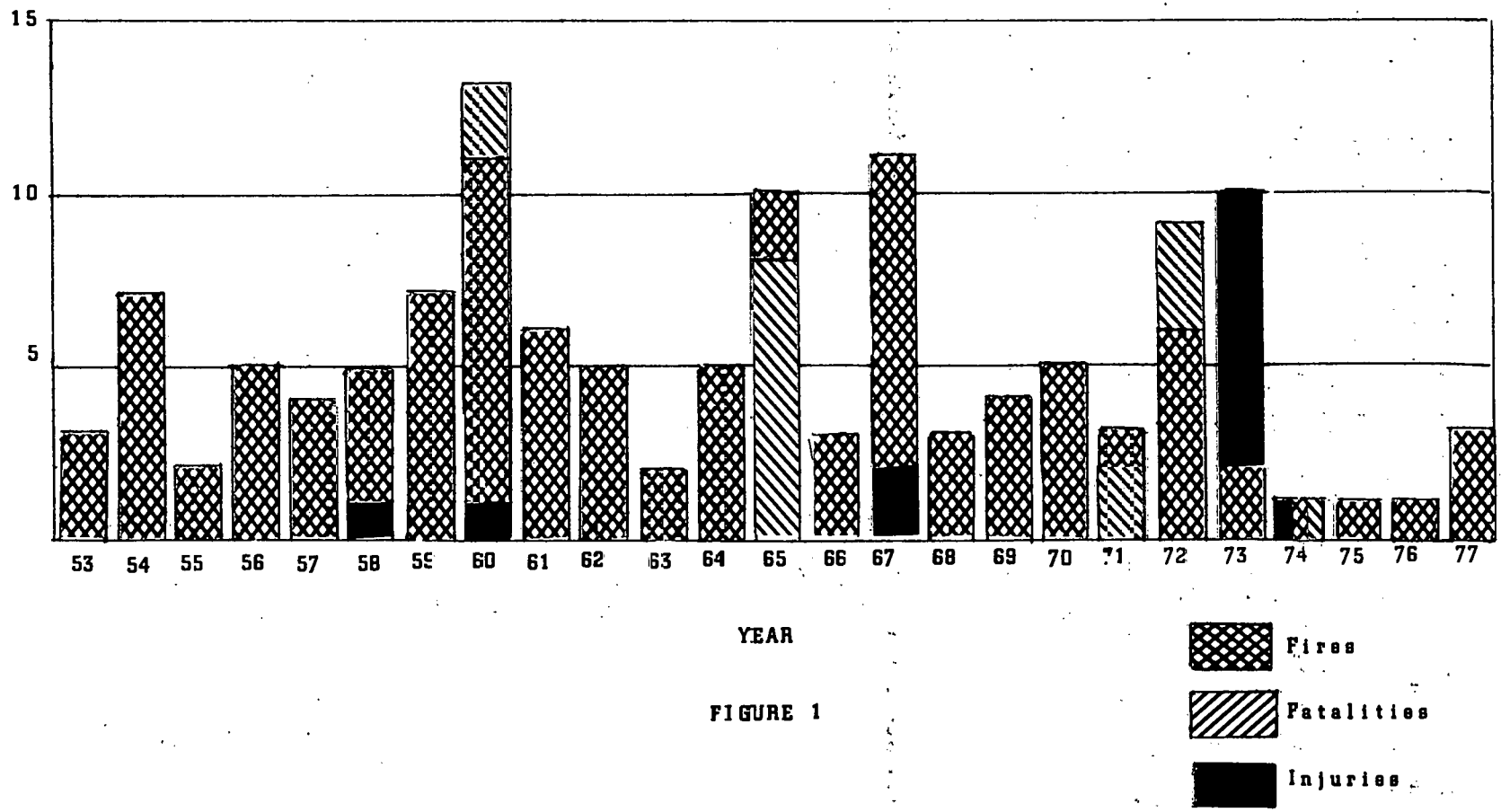


FIGURE 1

TEST PROCEDURES

Several series of arcing fault tests were conducted at a central mine shop in northern West Virginia. The shop facility utilizes a 1500 kVA bank of transformers to transform 23 kV to 2300/4160 V. An unregulated 500 kW rectifier and an induction regulated 1000 kW rectifier were used to convert 4160 Vac, three-phase power to 300 Vdc for the tests. Circuit protection was provided for the dc output by the installation of a panel mounted circuit breaker rated continuously for 4000 A. A remote-controlled contactor rated at 900 A continuous was used to initiate the arcing tests. Current was controlled by resistors rated at 1000 A for 12 sec. Circuit inductance was provided by using one of two air core inductors constructed of multiple turns of No. 4/0 and No. 2 AWG insulated wire. Figure 2 shows the setup of the testing circuit.

The arc electrodes were installed within a 2 x 2 x 4 ft. wooden box to insure safety of personnel and equipment. Standard trolley wire clamps installed on bell-type insulators were used to support the electrodes. This configuration elevated the electrodes approximately 6 in. above the bottom of the test box.

Cooper and steel electrodes were used in various configurations in order to simulate typical faults which are likely to occur in underground coal mines. Lengths of 9 section (400 MCM) trolley wire were used as the copper electrodes. Reinforcing bars of 1/2 in. diameter were used for the steel electrodes. The following configurations were tested:

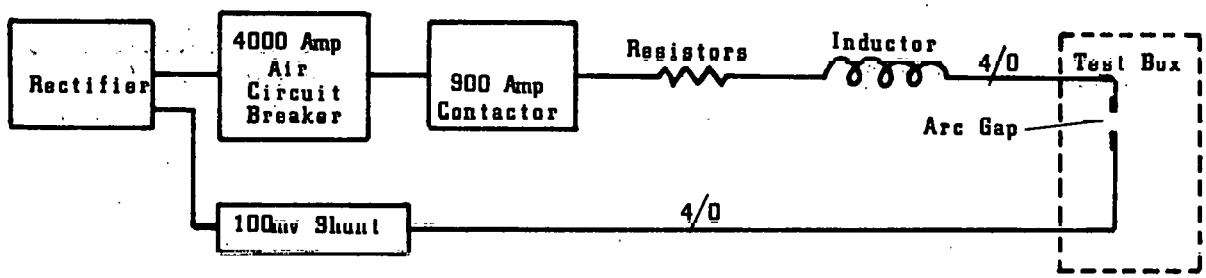
Electrode Material (+)	Polarity (-)
Copper	Copper
Copper	Steel
Steel	Copper
Steel	Steel

The initial arc gap was varied from 3/16 in. to 6 in. with the majority of the tests being performed in the 3/8 in. to 1 in. range. Arcs were initiated by bridging the arc gap with several strands of a 19 strand No. 12 AWG wire. The wire bridging the arc gap evaporated and allowed an arc to be established. The electrodes were placed end to end to prevent immediate blowout of the arc [1].

Most circuit interrupting devices are usually capable of detecting and deenergizing a fault within 0.2 sec. Therefore, successful arcs were considered to be achieved whenever an arc lasted 0.2 sec. (12 cycles at 60 Hz) or longer.

During each test the arc voltage and supply voltage were measured using high-voltage isolation amplifiers to attenuate the high-voltage signals to levels providing adequate trace deflections on the oscillograph. During the tests which contained external inductance, the voltage across the inductor was measured in a similar manner. Arc current was measured with an appropriate shunt in conjunction with an isolation amplifier which increased the low-voltage signal from the shunt to a level for adequate oscillograph deflection.

In the third series of tests which were performed during the spring of 1978, the power of the arc was measured using an analog device which multiplied the arc voltage and arc current signals to the oscillograph giving a signal which was recorded on the oscillograph.



TEST CIRCUIT

FIGURE 2

TESTS

Approximately 100 arcing tests were conducted between June 1977, and July 1978. Forty-two of these tests were conducted with an initial gap of about 3/8 in. (0.953 cm). The time constant of the test circuit was varied from approximately 0.0 ms to 16.6 ms by the addition of air core inductors. The time constant of a trolley circuit depends on the total

inductance and resistance of the circuit. When the trolley circuit is such that the inductance and resistance of the trolley conductors are much greater than the inductance and resistance of rectifier, the time constant of the circuit can be approximated by the time constant of the circuit conductors. This will usually be the maximum time constant that can be obtained for any given configuration of the circuit conductors. The minimum time constant that can be obtained will be that of the rectifier source. A typical 300 Vdc trolley circuit for a large track haulage mine can have a time constant of approximately 25 ms while the time constants of various sizes of rectifiers described in reference [2] are less than 4 ms.

Seventy-six tests were conducted with negligible inductance with the current being varied from about 350 A to 2500 A during bolted conditions. The arc gap was changed from 3/16 in. (0.48 cm) to 6 in. (15.24 cm). Of the 76 tests conducted without inductance, 23 were conducted with an initial arc gap of 3/8 in. Consistant results were obtained when the tests were conducted with this gap length.

When an inductance of 3.85 mH was used in 7 tests, the time constant of the circuit varied from 9 ms to 16.6 ms. Time constants between 5 ms and 8 ms were obtained when an inductance of 1.64 mH was used in 14 tests. An inductance of 0.17 mH was used in 4 tests to achieve time constants between 0.5 ms to 1 ms.

A list showing system parameters is given in the following chart for arc tests conducted with initial gaps of 3/8 in.

OBSERVATIONS

Over 100 arcs were investigated with various circuit parameters. Current was varied from 300 A to 2400 A with initial arc gaps ranging from 3/16 in. to 6 in. A large number of the tests were conducted with copper electrodes and an initial arc gap of 3/8 in. The following observations pertain to such arcs:

1. Both arc current and arc voltage contained high frequency components at currents below about 800 A. As inductance was introduced into the circuit, the high frequency components of the arc current were reduced substantially. However, the high frequency components of the arc voltage were not reduced. Thus, it appears that large changes of arc voltage will result from relatively small changes in arc current at low arc currents.

2. The value of arc voltage remained relatively constant for arc currents about 800 A. For arcs with gaps of 3/8 in., the value of arc voltage was approximately 50 V. This value remained essentially constant even for various combinations of copper and steel electrodes.

3. Once an arc was established, the arc gap increased sufficiently to extinguish the arc. The maximum length of the gap after the arc self-extinguished was approximately 1.5 in. When a steel cathode was used with a copper anode, the arc gap increased to a maximum of approximately 2 in. When steel was used for both electrodes, the arc gap increased to approximately 3 in.

ARCING FAULT DATA (3/8 inch Gap)

TEST	System Voltage (Volts)	System Resistance (Ohms)	System Inductance (mh)	Bolted Fault Current (Amps)	Arc Current (Amps)	Arc Voltage (Volts)
A13	325	0.90	0.0	338	225	
A14	325	No Data	0.0	No Data	No Data	
B5	325	0.60	0.0	500	375	75
B6	325	0.60	0.0	500	375	75
B12	325	0.40	0.0	725	550	No Data
C5*	325	0.30	0.0	975	750	75
C6*	325	0.30	0.0	1000	750	75
C10	325	0.20	0.0	1380	1200	58
C11	325	0.20	0.0	1380	1200	50
F3	325	0.36	1.64	850	675	55
F4	325	0.36	1.64	840	650	50
G1	325	0.31	0.17	950	800	50
G2	325	0.31	0.17	950	780	65
G3	325	0.39	3.85	770	550	75
H1	325	0.39	3.85	775	580	50
H2	325	0.23	3.85	1100	1000	60
I1	325	0.23	3.85	1100	1020	50
I2	325	0.20	1.64	1270	1200	50
I3	325	0.20	1.64	1300	1200	50
I4	325	0.16	0.17	1700	1400	50
I5	325	0.16	0.17	1700	1500	50
J1	325	0.15	0.0	1750	1400	50
J2	325	0.15	0.0	1900	1600	50
J3**	325	0.15	0.0	1800	1500	50
J4***	325	0.15	0.0	1650	No Data	No Data
K1**	325	0.15	0.0	1800	1500	50
K2**	325	0.15	0.0	1800	1500	50
K3**	325	0.20	1.64	1400	1200	50
K4***	325	0.20	1.64	1400	1150	67.5
K5**	325	0.20	1.64	1400	1200	50
L1***	325	0.20	1.64	1400	1150	50
L2****	325	0.20	1.64	1400	1200	62.5
M1****	325	0.20	1.64	1380	1240	48
M2***	325	0.15	1.64	1750	1550	50

* Are extinguished in less than 0.25 s
** Electrodes - Copper (+) and Steel (-)
*** Electrodes Copper (-) and Steel (+)
**** Both electrodes steel

4. Arcs in which steel was used for both electrodes, appeared more violent than arcs initiated with copper electrodes or copper and steel electrodes.

5. In the majority of the tests conducted the current reached bolted values before the arc was established. However, in at least one test, the arc was established before the current reached bolted value (See Figure 7). This was due to the small strands of wire bridging the gap melting before the bolted value was obtained. Thus, it is apparent that the maximum setting of overcurrent devices on trolley systems should be at values below the available bolted values.

Figure 3 shows a relationship between arc voltage and arc current for arcs having a 3/8 in. gap. The arc will usually operate at the intersection of this curve and the resistance load line of the circuit. The current will stablize at a fixed point on the curve and the arc will dissipate a relatively constant amount of power. The load line may intercept the characteristic curve in two locations as shown in Figure 4 but only one point is stable. The stable operating location is the point with the lowest arc voltage. Also, the arc voltage versus arc current characteristic curve will shift upward as the gap increases in length [3].

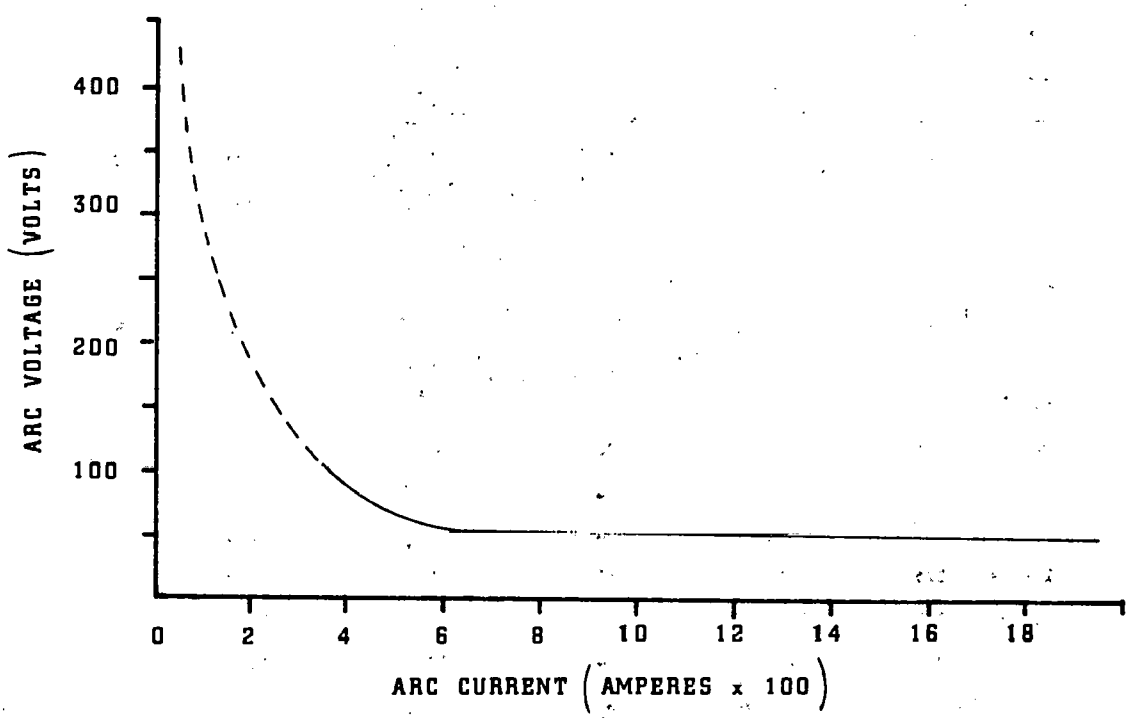


FIGURE 3

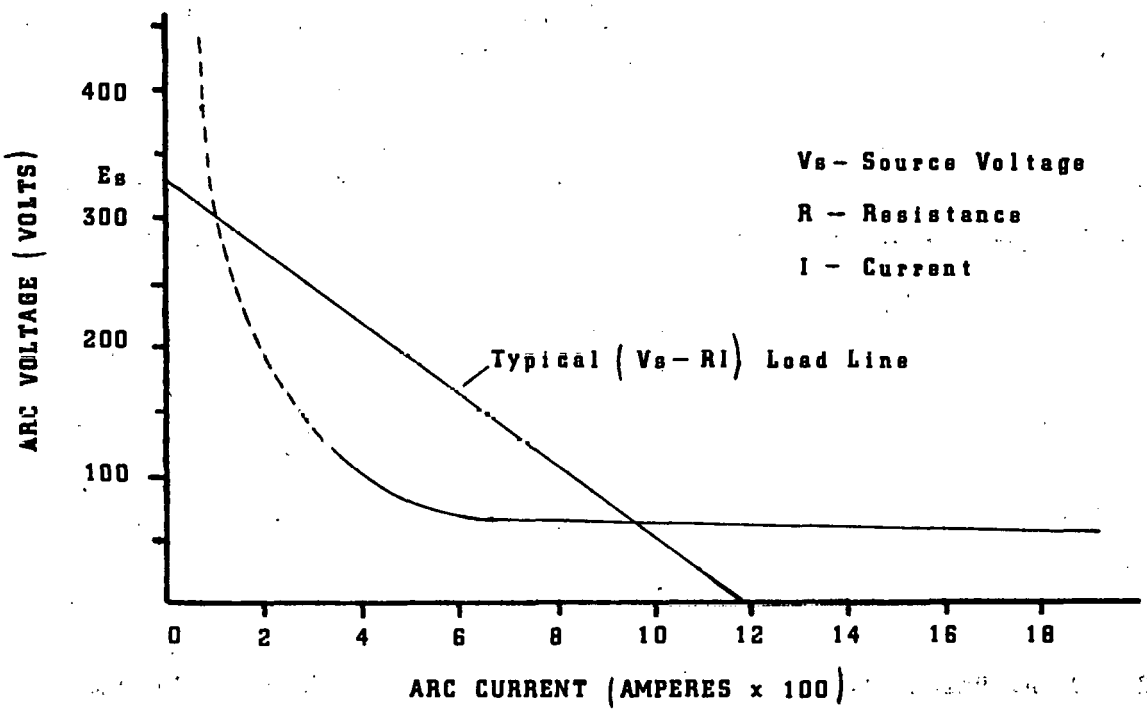


FIGURE 4

As mentioned above, the value of arc current is dependent upon the arc gap and the load line of the circuit. In circuits containing two or more sources, the arc current is dependent upon the total current flowing in the arc. Thus, as a source is disconnected from the circuit, a new load line must be obtained to determine the next value of arc current. The load line which should be used for circuits containing multiple sources can be obtained by using Thevinin's equivalent voltage and resistance of the circuit.

Figures 5 through 11 in the appendix are typical oscillographs obtained during the arcing fault tests.

FUTURE RESEARCH

The preliminary research to determine the characteristics of dc arcing faults was not sufficient to recommend specific changes in the requirements for short circuit protection for underground trolley systems. There is a need for conducting future research on dc arcing faults to determine the complete relationship between arc voltage and arc current for all faults above 100 A, including the effects of high intensity arcing as described in reference [4]. Also, surveys at various mines should be conducted to determine typical time constants and inductances of the trolley systems.

In addition, the following are areas which should be thoroughly studied to determine their effects on dc arcing faults:

1. Constant arc gaps
2. Arcing faults contaminated by coal and other combustible materials
3. Ignition of coal and other combustible materials by dc arcs
4. Various methods of initiating dc arcs
5. Methods of detecting arcing faults

REFERENCES

[1] Lawrence E. Fisher, "Resistance of Low-Voltage a. c. Arcs" IEEE Transactions on Industry and General Applications, Vol. IGA-6, No. 6 November/December 1970.

[2] D. A. Paice, A. B. Shimp, and R. P. Putkovich, "Circuit Breaker Development and Application" Phase 1 Research Report Westinghouse Electric Corporation NTIS PB 248 31 0/AS.

[3] Cobine, J. D., "Gaseous Conductors" McGraw-Hill 1941 pp. 371-378.

[4] Samuel Korman, "High'Intensity Arcs" International Science and Technology June 1964 pp. 90-98.

APPENDIX

Figures 5 through 11 show typical oscillographs of the tests performed during 1977. The oscillographs have been reduced to approximately 75 percent of their original size for reproduction purposes.

Figure 5

Test	C10
Supply Voltage	325 Vdc
Bolted Current	1380 A
Time Constant	0.0 ms
(+) Electrode	Cu
(-) Electrode	Cu
Average Current	1100 A
Time of Test	0.62755 s

Figure 6

Test	F-3
Supply Voltage	325 Vdc
Bolted Current	850 A
Time Constant	4.29 ms
(+) Electrode	Cu
(-) Electrode	Cu
Average Current	650
Time of Test	3.025 s

Note: Arc voltage contains high frequency components at lower values of current.

Figure 7

Test	H2
Supply Voltage	325 Vdc
Bolted Current	1100 A*
Time Constant	13-14 ms
(+) Electrode	Cu
(-) Electrode	Cu
Average Current	1000 A
Time of Test	2.4545 s

Note: Arc voltage fairly constant 60V
*Maximum value of available current not obtained

Figure 8

Test	I2
Supply Voltage	325 Vdc
Bolted Current	1270 A
Time Constant	6.41 ms
(+) Electrode	Cu
(-) Electrode	Cu
Average Current	1200 A
Time of Test	2.215 S

Note: Arc voltage contains high frequency components as the current drops to about 850 A.

Figure 9

Test	K4
Supply Voltage	325 Vdc
Bolted Current	1400 A
Time Constant	7.06 ms
(+) Electrode	Steel
(-) Electrode	Cu
Average Current	1150 A
Time of Test	1.7765 s

Figure 10

Test	L-2
Supply Voltage	325 Vdc
Bolted Current	1400 A
Time Constant	7.06 ms
(+) Electrode	Steel
(-) Electrode	Steel
Average Current	1200 A
Time of Test	5.325 s*

*Arc did not self extinquish - contactor opened to deenergize the test circuit.

Figure 11

Same as Figure 10. Starts at 0.4s and ends at 0.55s

Notes: (1) 470 A change in arc current in 6.25 ms (Time constant 7.06 ms).

(2) Maximum arc voltage - 495 Vdc

(3) Inductor voltage becomes negative for large changes in current.

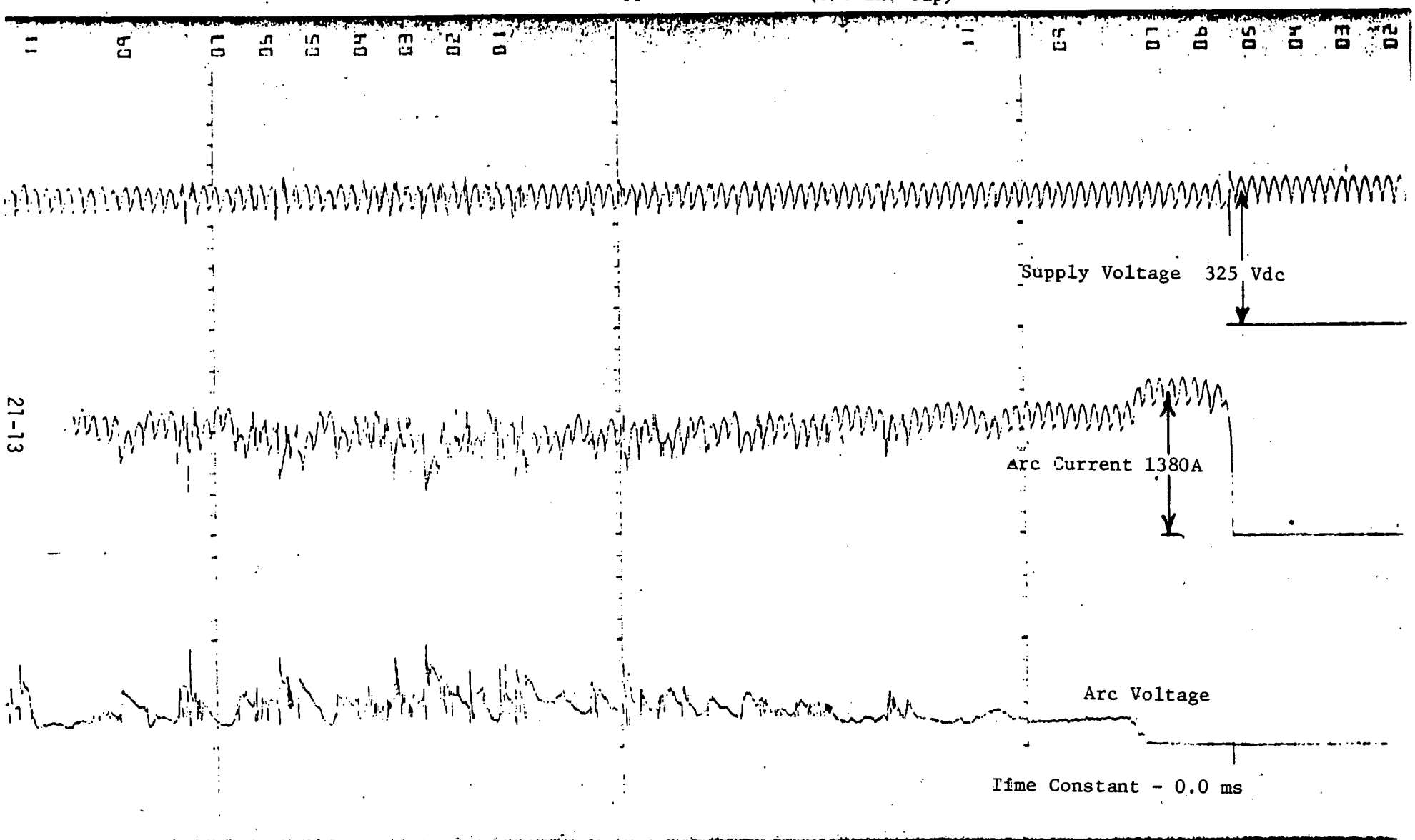

Test C10 Copper Electrodes (3/8 in. Gap)
Supply Voltage 325 Vdc
Arc Current 1380A
Arc Voltage
Time Constant - 0.0 ms
21-13

Figure 5

Test F-3 Copper Electrodes (3/8 in. Gap)

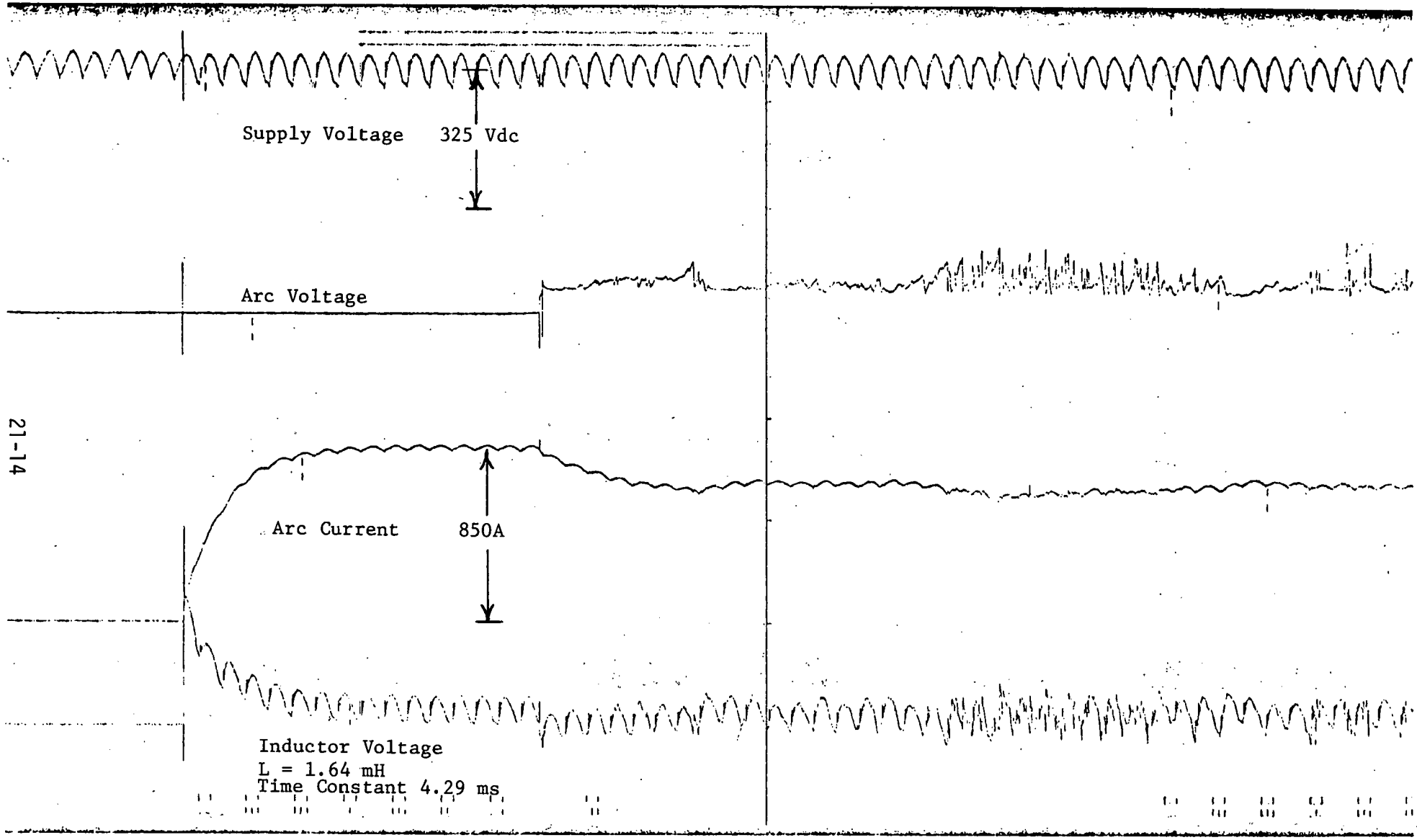


Figure 6

Test H-2 Copper Electrodes (3/8 in. Gap)

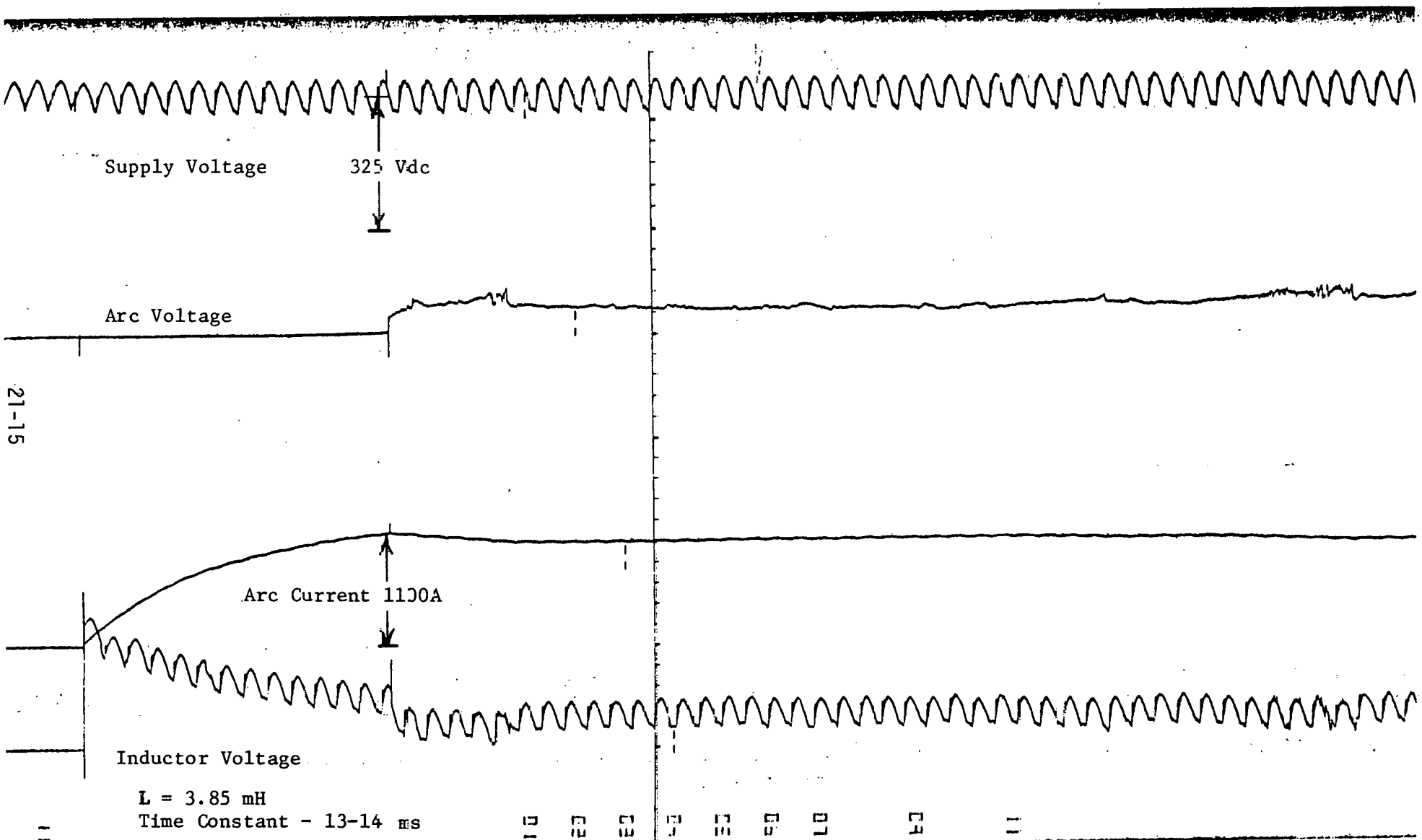


Figure 7

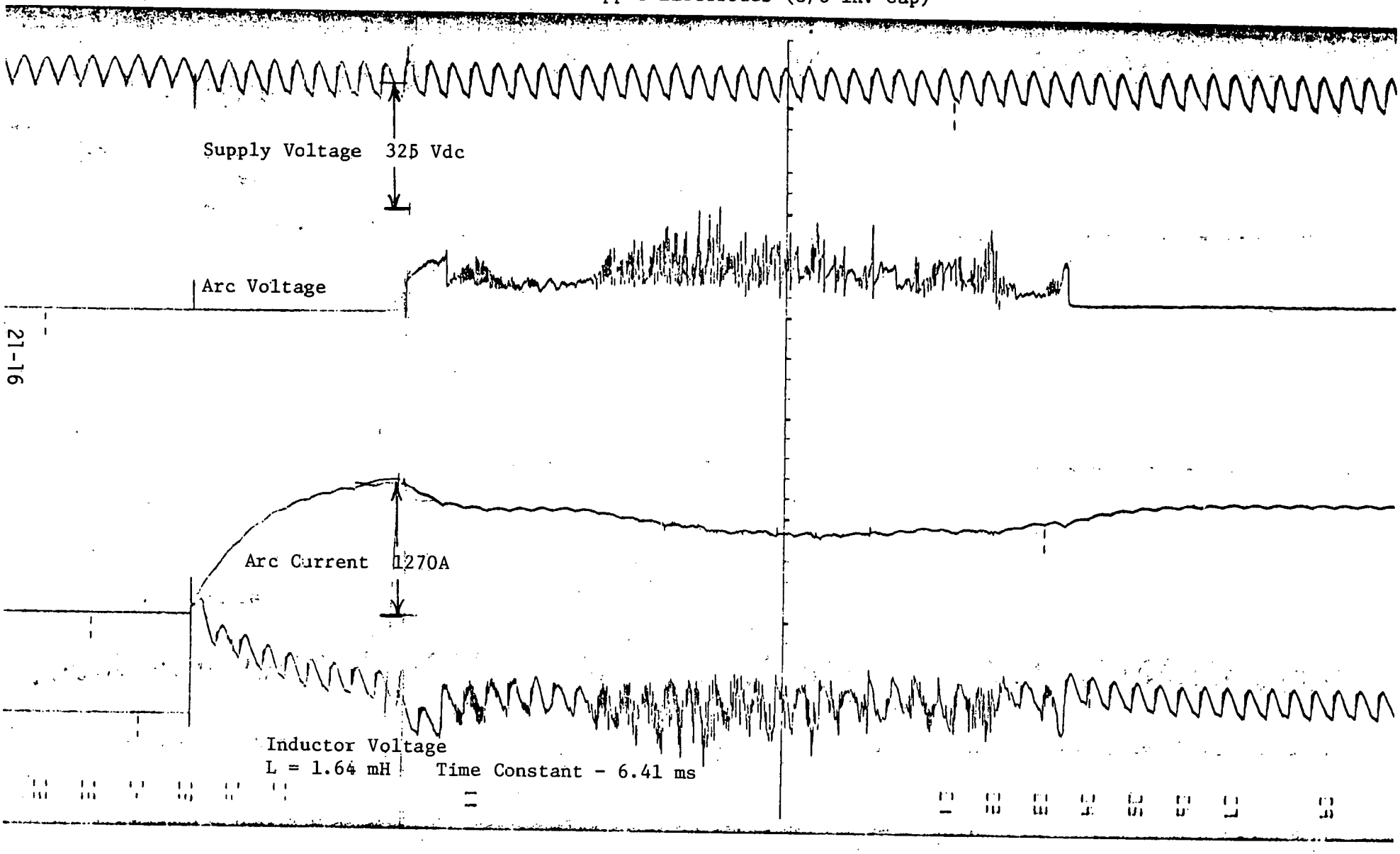

Test I2 Copper Electrodes (3/8 in. Gap)
Supply Voltage 325 Vdc
Arc Voltage
21-16
Arc Current 1270A
Inductor Voltage
L = 1.64 mH
Time Constant - 6.41 ms

Figure 8

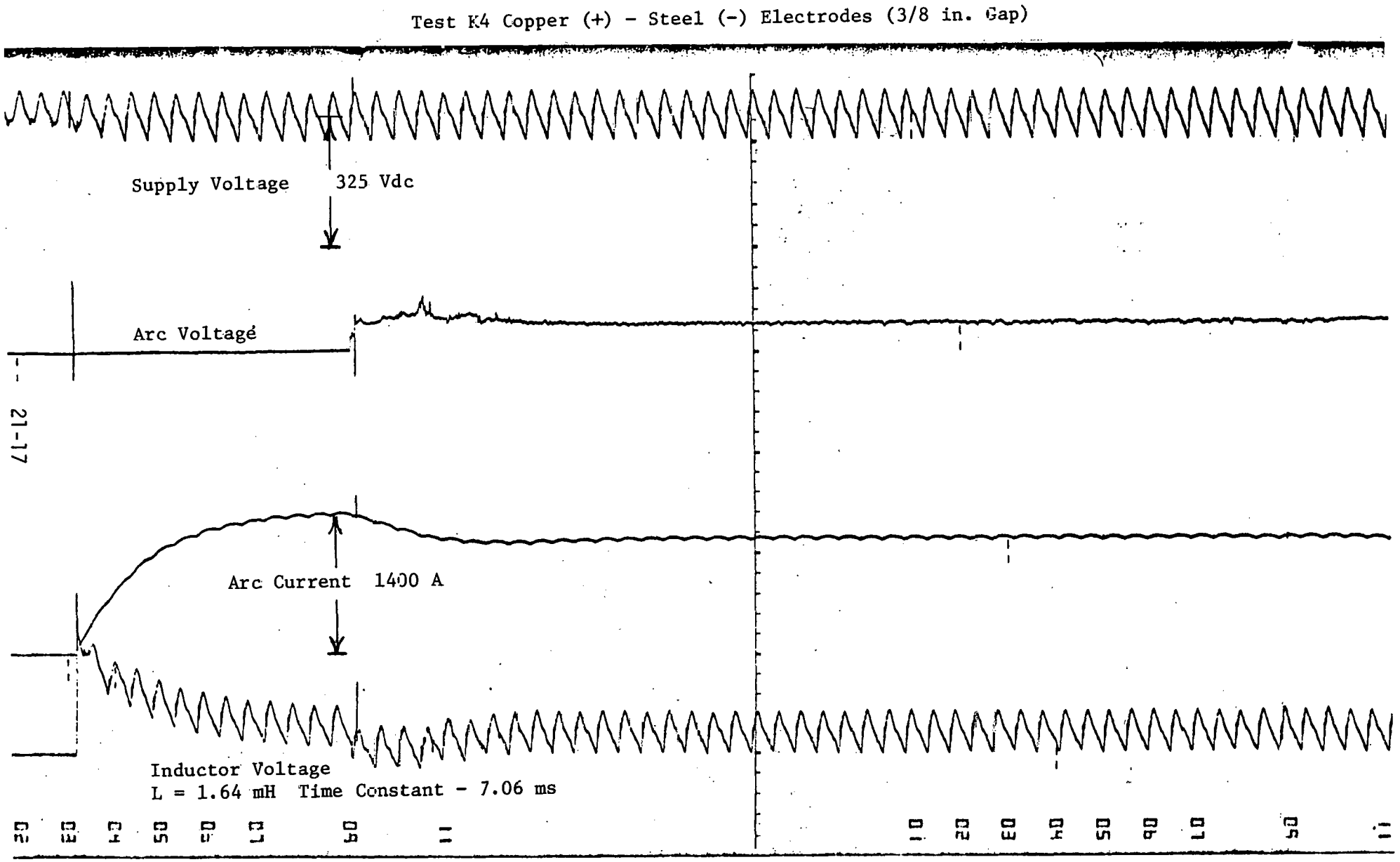
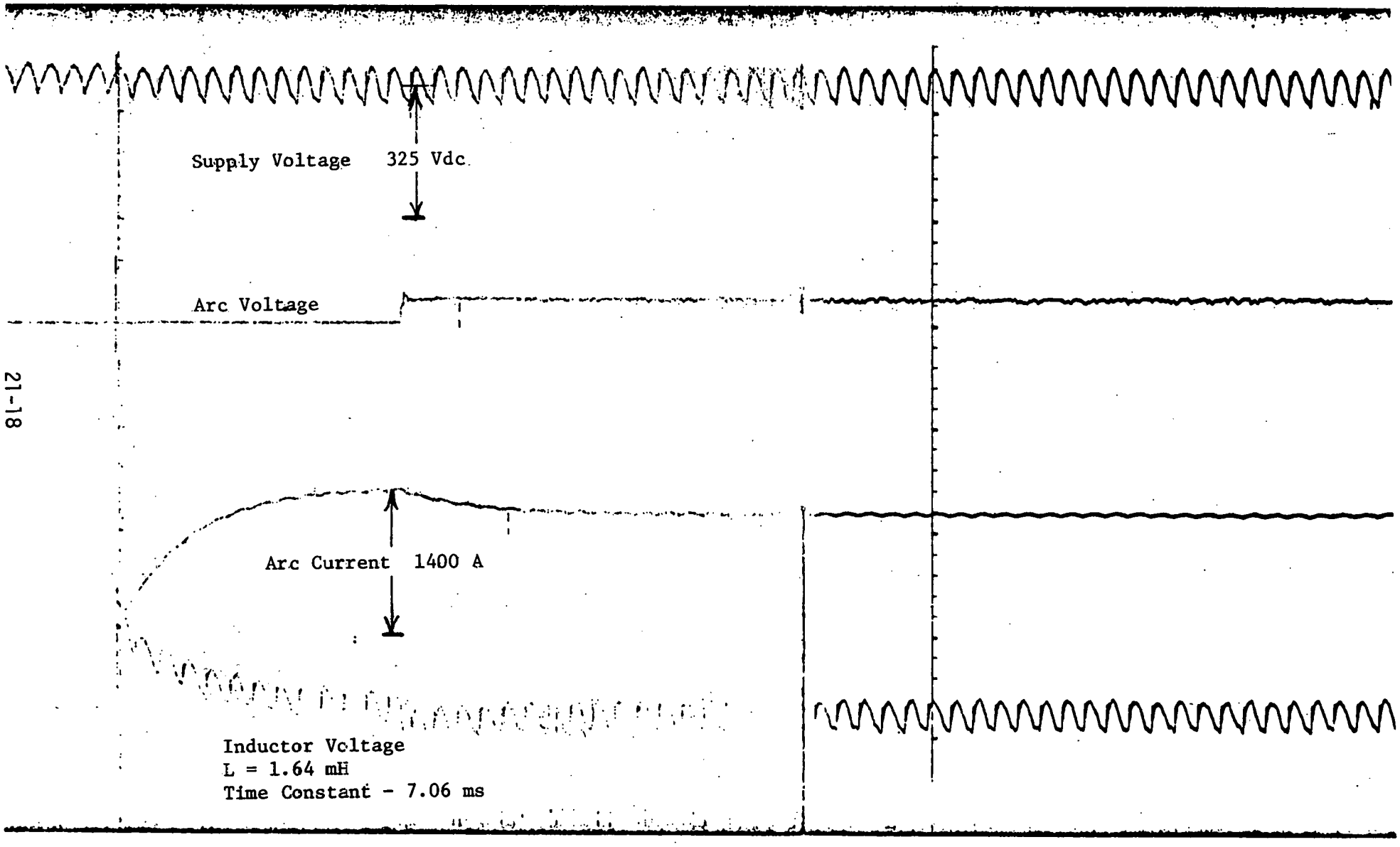

Test K4 Copper (+) - Steel (-) Electrodes (3/8 in. Gap)
Supply Voltage 325 Vdc
Arc Voltage
Arc Current 1400 A
Inductor Voltage
L = 1.64 mH Time Constant - 7.06 ms
21-17

Figure 9

Test L-2 Steel Electrodes (3/8 in. Gap)



Test L-2 Continued - Steel Electrodes

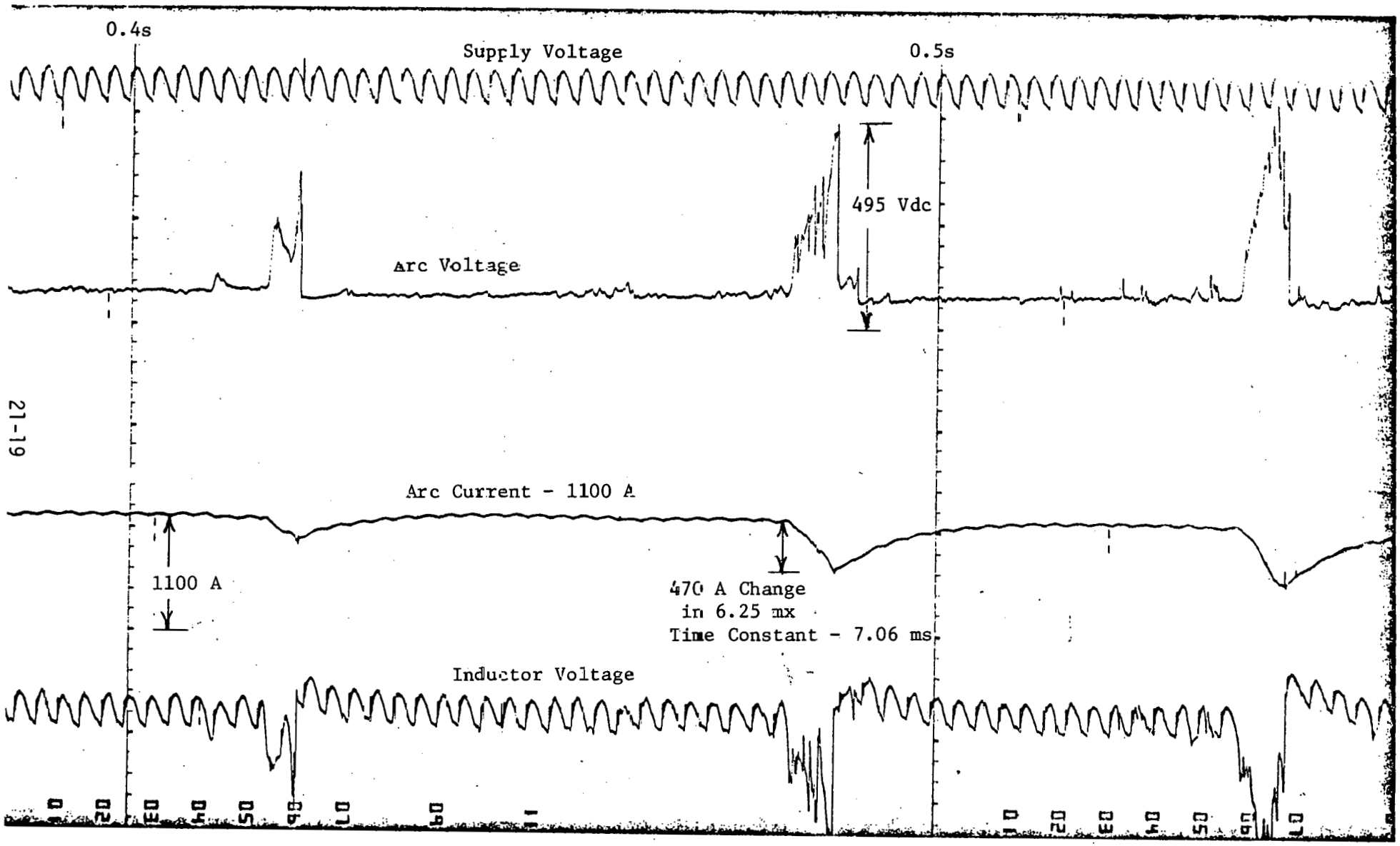


Figure 11

THE NEED FOR AND DEVELOPMENT OF

EFFECTIVE DC TROLLEY OVERCURRENT TRIPPING RELAYS

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ABSTRACT

The discussion will begin with the problems related to the unreliability of pre-1977 electromechanical and solid state d.c. overcurrent relays, and proceed through the establishment of practical design and performance guidelines to the development of reliable, fail-safe present day relays acceptable by the Mine Safety and Health Administration (MSHA) enforcement personnel. Also included will be the development of other related products directly associated with d.c. trolley circuit breakers.

INTRODUCTION

Since 1970, approximately 21 underground coal mine fires have been reported to MSHA which can be related directly to trolley wires or trolley feeder wires. As a result of these fires, 23 coal miners have sustained injuries, twelve fatally. It is also important to note that 48 percent of these fires were the result of inadequate overcurrent protection. Title 30, Code of Federal Regulations, Section 75.1001 states that, "Trolley wires and trolley feeder wires shall be provided with overcurrent protection." Section 75.1001-1 of the same code states, "(a) Automatic circuit interrupting devices that will deenergize the affected circuit upon occurrence of a short circuit at any point in the system will meet the requirements of 75.1001. (b) Automatic circuit interrupting devices described in paragraph (a) of this section shall be tested and calibrated at intervals not to exceed six months. Testing of such devices shall include passing the necessary amount of electric current through the device to cause activation. Calibration of such devices shall include adjustment of all associated relays to plus/minus 15 percent of the indicated value." The relays available to provide compliance with this section were of the electromagnetic-field-type design whose trip value was regulated by the amount of tension applied to the calibration spring. Due to this design, the six month testing and/or calibration became a laborious, time consuming, and sometimes dangerous task which mine management personnel were reluctant to adhere to.

COAL INDUSTRY REQUESTS MODIFICATION OF TESTS

The need for and development of effective trip initiation of d.c. trolley circuit interrupters was prompted by a request of the coal industry to MSHA to alleviate or modify the requirements of testing the existing relays each six month period. The direct stimulation for this request was the cumbersome calibration methods utilized in obtaining compliance with Section 75.1001-1(b), Title 30, Code of Federal Regulations. The two most common methods of calibration proved laborious, time consuming, and at times, could expose personnel to unsafe conditions due to poor procedures or practices. In both methods, the system was actually loaded by either resistors or a locomotive and the appropriate current read via a millivolt meter driven by a millivolt shunt. Each method produced acceptable results when executed by diligent and competent personnel, but the time and diligence required led the coal industry in search of devices requiring less patience and arduous methods in obtaining more dependable results. This search led to the advantegous incorporation of solid state electronics and as a result industry realized two important advantages in ease of testing and calibrating with handheld devices. The laborious testing methods which once took as much as an hour could now be accomplished in the course of a few minutes. The other even more significant advantage was an indication of an internal failure.

INFERIOR PRODUCTS

At the onset, the practical application of these units appeared so useful that the market became flooded with poorly designed units which proved to be unreliable. Since the safety of every person in the mines is dependent upon the device's ability to detect and initiate interruption of a short circuit in the trolley system, the main criteria for evaluation was reliability. Unfortunately, the units proved to be inferior rather than superior to the existing devices. The most serious discrepancy was the failure of the unit without any indication that the unit had failed. At least the electromechanical units would trip at some value, where the solid state units would fail to trip even at values approaching a bolted fault at the rectifier secondary buss. With a situation existing where no protection at all could continue for a period approaching six months, MSHA considered banning these devices as primary protective relays. At this time, further requests were made by MSHA inspection personnel, original equipment manufacturers, and coal industry maintenance personnel that some useful design guidelines be developed which would permit solid state overcurrent relays to be utilized on d.c. trolley systems.

GUIDELINES YIELD PRACTICAL AND RELIABLE RELAY

With the assistance of industry, West Virginia University, and MSHA personnel, the following guidelines were developed by MSHA's Technical Support, Beckley Electrical Testing Project (BETP).

SUBPART A — General Provisions

PURPOSE

The regulations in this part set forth the requirements to obtain acceptance for direct current overcurrent devices for protection of direct current trolley systems and trolley feeder wires. These requirements govern only those devices not requiring primary injection for testing and calibration.

Two general classes will be considered for acceptance under this part. Class category will be determined by the type of components used in manufacturing the particular device:

A. Class A — Metered Relays and Solid State Devices

B. Class B — Electromechanical Devices

SUBPART B

DESIGN PERFORMANCE REQUIREMENTS

A. Metered Relays and Solid State D.C. Overcurrent Devices

1. Any component failure shall not negate the ability of the overcurrent device to properly perform its intended function or shall open the circuit interrupting devices.

2. When redundant circuitry or components are used to comply with section (A) of this subpart, a test circuit shall be incorporated which will test each redundant circuit or component in its entirety. The test circuit shall be self-initiating and initiate a test at least every 5 seconds. If the test circuit fails to initiate a test or completely test the redundant circuitry or component, the circuit interrupting device must trip.

3. Overcurrent devices shall be capable of opening the circuit interrupting device when overcurrents are sensed in either direction, or when 50 percent of the lowest trip setting appears on the sensing leads in the reverse direction.

4. The calibration shall be maintained within plus/minus 15 percent of the indicated trip setting at any setting within the specified operating range to include the following conditions:

a. The overcurrent device shall be initially calibrated at a temperature of 18 - 27°C, unless otherwise specified.

b. Line voltage variations of plus/minus 20 percent from nominal.

c. Temperature variations from ambient (20°C) to -26°C (-14.8°F) and from ambient (20°C) to 60°C (140°F). If the calibration is not maintained during the above specified voltage and temperature variations, then the circuit interrupting device must trip.

5. The overcurrent device shall open the circuit interrupting device when the calibration is altered more than plus/minus 15 percent by line voltage variations.

6. When the sensing leads are broken or become disconnected, the circuit interrupting device shall be opened.

7. Electromechanical relays used by overcurrent devices to open the circuit interrupting device shall be energized during normal operation.

8. Overcurrent devices shall be capable of performing 10,000 operations at the equivalent amperage and inductive loading of a size 9 d.c. contactor with a d.c. operating coil.

9. Overcurrent devices shall be capable of properly performing their "intended function" while under duress of voltage transients applicable to d.c. trolley systems and trolley feeder systems.

10. Overcurrent devices shall be capable of passing an accelerated life test of 10,000 operations in which calibrated trip pulses applicable to overcurrent settings within the specified operating range will be applied to the sensing leads.

11. The entire unit, including any electromechanical relays, shall operate in 150 milliseconds or less from initiation of 115 percent trip setting and 75 milliseconds or less from initiation of 200 percent trip setting at any setting within the specified operating range.

12. The overcurrent devices shall have front panel controls insulated for the specified nominal voltage and these controls shall be easily accessible for testing and calibration.

13. The millivolt operating range shall not be adjustable beyond either end of the specified operating range.

B. Electromechanical Devices

1. The calibration of the overcurrent device shall be maintained within plus/minus 15 percent of the indicated trip setting at any setting within the specified operating range in coordination with buss bar currents or the circuit interrupting device shall trip.

2. The overcurrent device shall be initially calibrated at a temperature of 18 - 27°C, unless otherwise specified.

3. Calibration shall be maintained during temperature variations from ambient (20°C) to -26°C (-14.8°F) and from ambient (20°C) to 60°C (140°F) or the circuit interrupting device shall trip.

4. Devices shall be capable of opening the circuit interrupting device when overcurrents are sensed in either direction.

5. Devices shall operate in 150 milliseconds or less from initiation of 115 percent trip setting and 75 milliseconds or less from initiation of 200 percent trip setting at any setting within the specified operating range.

6. Devices receiving an activation signal through sensing leads shall provide a means to monitor the continuity of said leads.

7. Overcurrent devices shall be capable of passing an accelerated life test of 10,000 operations with trip pulses applicable to overcurrent settings, within the specified operating range, applied to the device.

SUBPART C

EVALUATION AND TESTS

The overcurrent device shall perform its intended function for each appropriate test.

A. Metered Relays and Solid State Direct Current Overcurrent Devices

1. Failsafe Test

a. The following components will be opened and shorted:
All semi-conductors - capacitors - transformers - inductors - surge suppression devices

b. The following components will be opened:
resistors - lamps - relay coils

c. Integrated circuits will be opened and shorted as necessary by design and application.

d. Individual power supplies will be disabled.

2. Laboratory Performance Tests

a. When incorporating a test circuit to comply with the failsafe section, the interval between test initiations will be measured with an oscilloscope and time elaspe meter. The temperature will be varied from ambient (20°C) to 26°C (-14.8°F) and from ambient (20°C) to 60°C (140°F) during this examination.

b. Overcurrent trip pulses will be applied to the sensing leads in both directions. A 50 percent reverse polarity signal will be applied to the sensing leads of unidirectional devices. The temperature will be varied as specified in section (a) of this subpart.

c. The calibration will be checked at 20 percent intervals during each appropriate examination. The temperature will be varied as specified in section (a) of this subpart.

d. The control voltage will be varied plus/minus 20 percent from nominal while calibrated trip pulses are being applied to the sensing leads. The temperature will be varied as specified in section (a) of this subpart.

e. The control voltage will be decreased until the electromechanical relay fails to operate while calibrated trip pulses are being applied to the sensing leads. The temperature will be varied as specified in section (a) of this subpart.

f. Each sensing lead will be disconnected individually.

g. Relay contacts will be opened during normal operation by removal of relay coil voltage.

h. Relay contacts will be operated at the equivalent amperage and inductive loading for opening and closing the operating coil of a size 9 d.c. contactor with a d.c. operating coil for 10,000 operations at the system voltage. The temperature will be varied as specified in section (a) of this subpart.

3. Transient Test

a. In lieu of laboratory transient applications, a representative sample of units installed at three mines acceptable by MSHA shall be monitored and evaluated by MSHA — BETP to determine the necessary suppression required. The testing period shall be sixty working days. In the future when technology becomes available to properly evaluate and reproduce transients applicable to mine power systems, the laboratory transient tests will be reestablished and applied to units manufactured after that date. The testing period shall be as specified in 5a of this subpart.

4. Accelerated Life Test

a. Each device will receive an accelerated life test in which various trip pulses will be applied at the input sensing leads ranging from 115 percent to 1,500 percent above the units nominal trip value. Additional pulses ranging from 50 - 80 percent of the units nominal trip value will also be applied. The temperature will be varied as specified in section 2a of this subpart.

b. Relay clearing time on 115 percent and 200 percent trip pulses will be measured accordingly.

5. Field Evaluation Test

a. Five units each for which application is being made shall be field tested on a track and trolley haulage system acceptable to MSHA for a period of 60 working shifts per each unit. Such field testing shall be monitored by MSHA and the device shall not be acceptable if more than one unit fails during the evaluation period.

b. Additional field evaluation may be required as specified by MSHA.

B. Electromechanical Devices

1. Laboratory Tests

a. The calibration will be checked at appropriate percentage intervals during each examination.

b. The temperature will varied from ambient (20°C) to -26°C (-14.8°F) and from ambient (20°C) to 60°C (140°F) during each appropriate examination.

c. Overcurrent trip signals will be applied in both directions.

d. Relay clearing time will be measured accordingly.

e. The sensing leads will be broken during normal operations.

2. Accelerated Life Test

a. Each device will receive an accelerated life test of 10,000 operations via trip pulses applicable to overcurrent settings of the device.

3. Field Evaluation Test

a. Five units each for which application is being made shall be field tested on a track and trolley haulage system acceptable to MSHA for a period of 60 working shifts per each unit. Such field testing shall be monitored by MSHA and the device shall not be acceptable if more than one unit fails during the evaluation period.

b. Additional field evaluation may be required as specified by MSHA.

SUBPART D

FIELD TESTING AND EVALUATION

A. Overcurrent devices and associated relays shall be tested and adjusted to within plus/minus 15 percent of the indicated name plate trip value by passing an appropriate activation signal through the device in such a manner as to test the entire sensing and deactivation circuitry.

B. The following methods are acceptable as complying with Section (A) of this subpart.

1. All electromechanical d.c. overcurrent devices.

a. Passing the necessary amount of current through the d.c. buss bar to cause activation of the overcurrent device.

2. Electromechanical d.c. overcurrent devices incorporating a calibration coil.

a. The initial installation shall be tested by using the calibration coil to set a trip point and then passing a controlled current through the buss bar causing the overcurrent relay to operate. Operation shall occur within plus/minus 15 percent of the calibrated set point.

b. After the initial installation test, the following method may be used. A test current shall be applied to the calibration coil and the overcurrent relay shall operate within plus/minus 15 percent of the indicated set point. All testing and calibration with the calibration coil shall be performed under no-load conditions.

3. Solid state overcurrent relays receiving an activation signal from a series shunt.

a. An appropriate test signal shall be applied in series with the sensing leads in both directions and the overcurrent relay shall operate within plus/minus 15 percent of the indicated set point.

b. The trolley system shall be deenergized and a sensing lead disconnected. The overcurrent relay shall not reclose when power is reapplied.

4. Solid State Units Not Using a Series Shunt

a. An appropriate signal shall be applied in a manner acceptable to the sensing characteristics of the overcurrent devices and the overcurrent relay shall operate within plus/minus 15 percent of the indicated set point.

As a result of these guidelines, the industry now has available a solid state overcurrent relay which has proven to become most reliable for the detection and initiating interruption of short circuits in trolley systems and at the same time is extremely easy to test using a millivolt source. The unit is a result of an extensive effort by Lee Engineering Division of Consolidation Coal Company[1], and is protected by patent to support its unique design. This unit carries a Letter of Test and Evaluation issued by Technical Support and is accepted by MSHA inspection personnel as the primary tripping initiator for overcurrent protection on d.c. trolley circuits.

Another unit produced by Diversified Controls, Incorporated has completed the laboratory section of these guidelines, but still requires completion of the field evaluation to receive a Letter of Test and Evaluation.

An electromechanical series overload manufactured by General Electric Company is also used extensively, and has been accepted by MSHA as providing compliance with Section 75.1001, Title 30, Code of Federal Regulations. The unit incorporates a turns ratio coil which simulates the buss bar currents and is easy to calibrate. The unit was laboratory tested by Technical Support but did not receive a Letter of Test and Evaluation due to a prior acceptance by CMS&H. The unit has good application, but is still difficult to calibrate in underground portable rectifiers due to its buss bar location. Similar in construction to the General Electric Company relay is a Line Power Manufacturing Corporation relay which requires field evaluation in completing requirements for a Letter of Test and Evaluation.

RELATED PRODUCTS

Other related products under development are Westinghouse's discriminating circuit breaker and portable calibrator for d.c. circuit breakers and Lee Engineering's trolley breaker control system.

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[1] The mention of specific brands or trade names is made to facilitate understanding and does not imply endorsement by MSHA.

The discriminating circuit breaker is being developed under a Bureau of Mines contract by Westinghouse's Research and Development Center in Pittsburgh, Pennsylvania. The system operates on a trolley network which has been tuned for 3 Khz current flowing through a current detector. A further expansion of this system has been the incorporation of a voltage detector where comparatively long segments exist between the oscillators. This combination of detectors alleviates tuning the trolley wire for 3 Khz impedances while enhancing reliability.

The portable calibrator for d.c. circuit breakers is also a Westinghouse and Bureau of Mines contract with the resulting product being aimed at easier testing and calibration of the older type electromechanical series overloads. The device produces a regulated high current pulse of short duration and is applied directly through the series overload. While still under evaluation by Technical Support, the unit appears to have good application. In contrast to the old resistor method of two or three men and as much as 40 minutes per breaker, this unit can be handled by only one man and requires as little as 15 minutes per breaker.

The trolley breaker control system is under development by Lee Engineering Division of Consolidation Coal Company and incorporates the solid state overcurrent device mentioned earlier in this paper. The unit operates over a range of 100 Khz to 500 Khz and automatically regulates the overcurrent setting of each circuit breaker within a specified block of trolley wire via continuous transmissions of coded signals from the equipment. Each piece of equipment requires a separate transmitter and transmitting antenna to transmit its signal to a receiving antenna extended the entire length of the designated trolley block, plus an overlap of 500 feet on each end. The overlap provides continuous regulation when changing trolley blocks regulated by separate receivers.

CONCLUSION

An effective and reliable solid state overcurrent trip relay for d.c. trolleys is now feasible. It can eliminate a tediously time consuming, and sometimes unsafe situation. It is also a demonstration of what can be accomplished when regulatory agencies, industry, and concerned manufacturers work together in reaching a common goal. The end result of all this effort is increased safety in operation of trolley systems in coal mines. If these units are properly installed and maintained, trolley fires in coal mines can become ancient history.

REFERENCES

Burr, John F. An Improved System for the Protection of Trolley Wires in Underground Coal Mines. (1976) Lee Engineering Division of Consolidation Coal Company.

Price, D.A., and Putkovich, R.P. Circuit Breaker Development and Application, Monthly Technical Report No. 67, January 1978, and Monthly Technical Report No. 68, February 1978, Westinghouse Research and Development Center.

MSHA, Accident and Investigation Files.

USE OF PROGRAMMABLE CALCULATORS

IN MINE POWER SYSTEMS

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ABSTRACT

The small mine operator may not feel he is economically justified in either the purchase of a computer or the purchase of a terminal and payment of a minimum monthly charge for computer services. For a small initial investment (a few hundred dollars) and a minimal continuing expense for paper tape and magnetic cards the same small operator can supply his engineering staff with a great deal of design help via programmable calculators. In a similar vein, it is possible for mine inspectors to do a more thorough and efficient analysis of the safety of a mine power system using programmable calculators. In order to satisfy these needs, two sets of software have been developed for programmable calculators: design programs, safety checking programs.

In order to make the programs of maximum benefit to the industry and inspectors they have been written for calculators with both algebraic and reverse Polish notation.

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INTRODUCTION

The electrical engineer who works for a mining company or a federal enforcement agency is generally asked to do a variety of tasks, ranging from electronics to power systems and communications. The goal of this paper is to present the results of development work aimed at easing the demands on those people with respect to power system design or inspection. At the same time the use of expensive portable terminals and the continuing costs of an account with a large computer system were avoided. This was accomplished by developing a series of programs for design and performance checking using programmable calculators. Thus, even the smallest engineering office can afford the time cost of a few hundred dollars and a minimum continuing expense for magnetic cards and possibly paper tape.

Assessment of Program Needs

It was obvious from the start that two sets of programs were needed. One set of programs would be design oriented, and would be used by mining engineers to design power systems. This set of programs

would be capable of taking system requirements and selecting cable sizes starter sizes, breaker settings, etc. The second set of programs would be more analysis oriented, since an inspector would not be interested in redesigning a system in the field. However, for a given configuration he would like the ability to check if a starter was adequate, if circuit breakers and fuses would clear faults properly, if system coordination was achieved, etc.

Based on the above logic, two lists of programs were developed. The programs that could be readily developed for available programmable hand-held calculators were written. Table 1 lists the design oriented programs while Table 2 lists the inspection or analysis oriented programs.

TABLE 1

DESIGN ORIENTED PROGRAMS

1. Selection of NEMA size contactors
2. Selection of power factor correction capacitors
3. Snubber circuit design
4. Ground bed design
   a. Driven rods in a straight line
   b. Driven rods in a circle
   c. Buried counter poises
5. Fuse coordination
6. Three phase short circuit analysis
7. Relay setting selection
8. Instantaneous trip settings for trailing cables

TABLE 2

ANALYSIS OR SYSTEM INSPECTION PROGRAMS

1. Ground bed prediction
2. Prediction of transient voltages and/or currents
3. Device coordination checking*
4. Short time equipment rating
5. Cyclic equipment loading
6. Starter size adequacy**

*Part of Relay Setting Selection Program
**Same as Selection of NEMA Size Contactors

HARDWARE LIMITATIONS

Almost any one who has used programmable calculators realizes there are two prominent manufacturers in the business who use two completely different logic systems. The two systems are Reverse Polish Notation and Algebraic Operating System. Generally, if one has used one system or the other for a prolonged period of time, it is confusing to use a calculator with the other logic. It was decided that any programs developed would be converted into both logic systems.

Two features of hand-held calculators that are important in developing programs of this type, are the number of program steps available and the number of registers available to store data. Both of these capabilities grow as newer and more sophisticated models become available. The number of program steps is hard to compare. On some calculators one can perform the four basic functions (+,-,x,÷) on a storage register in one program step. On others this may require as many as 4 program steps (SR-52). An actual example of two programs written to do the same task using the same technique required 95 steps on the HP-97-67-19c-29c while 142 steps were required for the SR-52. It should be noted these are not efficiently or compactly written programs.

Another subtlety of program storage to consider is the trade off between program steps and data storage when the same memory is used for both purposes. Eight program steps are lost for each data storage on the TI-59. When the maximum number of program steps is advertised no memory is set aside for data storage. Unfortunately, this is an unrealistic measure of program storage capability.

REVIEW OF DESIGN PROGRAMS

Due to space limitations, only the title, description of the program, and solution technique used are included. An expanded users' manual for the software can be obtained through the U. S. Bureau of Mines.

Title: Selection of NEMA Size Contactors
Description: In the design of low-voltage (460-575V) electric utilization systems, one of the routine problems encountered is the selection of the correct NEMA size contactor for a particular size motor. This process is usually done by finding the correct manufacturer's application note or referring to a table in a reference book (see Table 76, page 304 of IEEE Redbook[1]). The tabular information can be replaced by a program card which may be more accessible in the field or in an engineering office.
Solution Technique: The solution technique is quite simple. Data from the table are fit to a curve. When the curve does not provide an acceptable fit, the program branches to a default option which is basically a table look up procedure. The equation used is of the form:

$$\text{Starter size} = k \log (\text{horsepower}) - k' \qquad (1)$$

This equation is quite accurate for motor sizes between 2.1 to 600 HP. Outside this range a table look up procedure is used. Starter sizes are displayed as 0.0, 0, 1, ..., 9.

Title: Selection of Power Factor Correction Capacitors
Description: A common problem encountered in industrial power systems is the calculation of the kVAr of capacitors required to increase the power factor of a known load from a known value to a higher value. In order to solve this problem, one can use tables of multiplication factors (see Table 50 of page 252 of the IEEE Redbook[1]) or solve a power

vector diagram. The program uses an analytical approach equivalent to the table of multipliers.

Solution Technique: If one knows the original and desired power factor (assuming both are lagging) as well as the real power load, he can use the following equation to find the rating of capacitors required to change the power factor from the original to the desired level.

$$\Delta Q = P(\tan(\cos^{-1} pf_1) - \tan(\cos^{-1} pf_2)) \qquad (2)$$

where P = real power load in kW

pf_1 = original power factor (assumed lagging)

pf_2 = desired power factor (assumed lagging)

ΔQ = required capacitor rating in kVAr

This equation is exact and can be derived by observing the vector diagrams for real and reactive loads when real power is constant and the reactive power is changed by an increment, ΔQ. The equation listed above is used to derive the table of multiplication factors in reference books.

Title: Snubber Circuit Design

Description: One of the popular techniques for suppressing transients is to use a low pass filter to eliminate the high frequency transients. The filter provides a low impedance path to ground for the high frequency energy in the transient while providing a high impedance path for the system voltages and currents. The filter used is generally an RC type filter. This program provides an alternative to the nomogram technique normally used in the design of these filters.

Solution Technique[2]: The fundamental basis for the program is the equation derived from the nomogram shown in Fig. 1. The equation is of the form

$$y = a + b \ln x$$

where y is the ratio of the peak transient voltage to the system peak voltage and x is the damping factor. In this equation a and b are constants determined in the curve fitting routine.

The input data required are the following:

1. Transformer rating in volt-amperes - VA
2. Secondary line-to-neutral voltage (rms) - V_{sec}
3. Frequency - hertz - f
4. System series inductance in henries - L
5. Allowable peak transient voltage in volts - V_{pk}

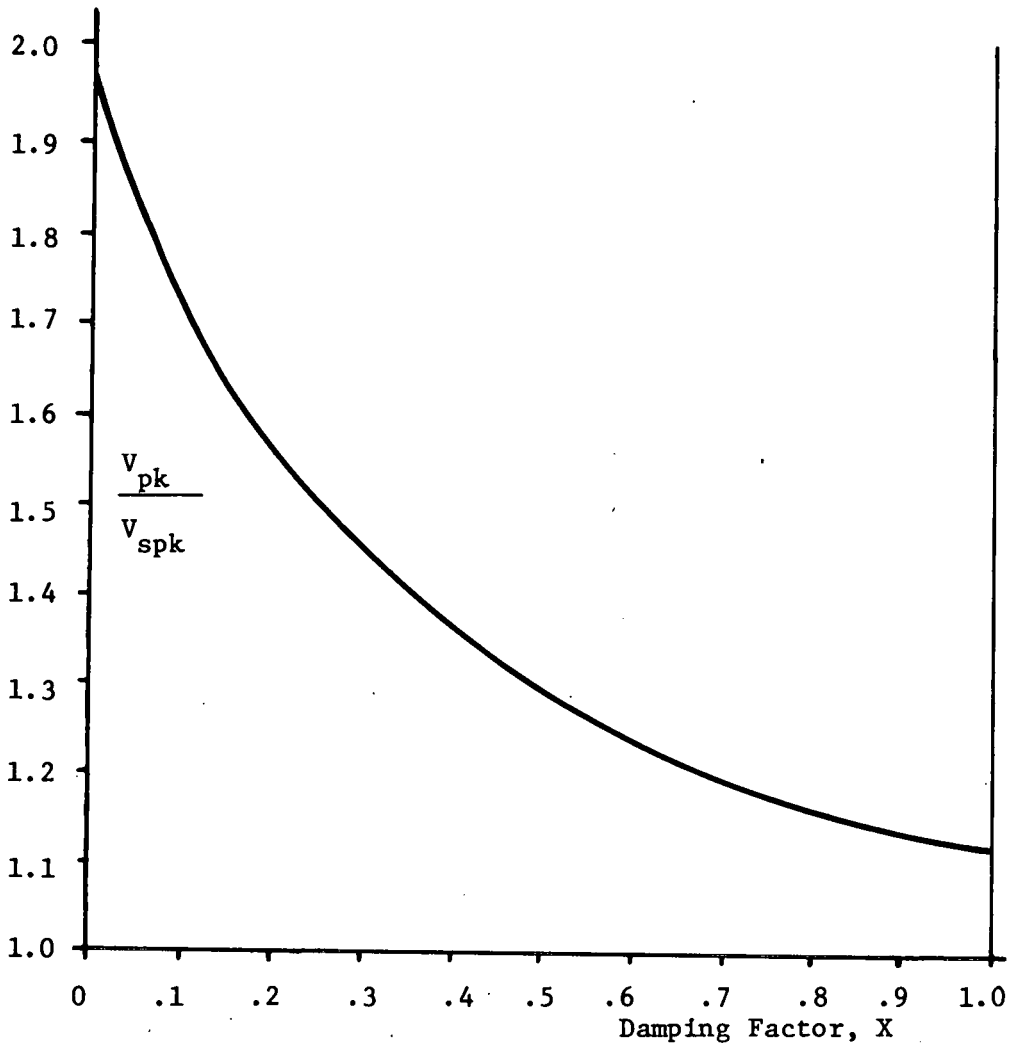


Fig. 1. Nomogram Used to Find Damping Factor.

After these data are entered, the capacitance is calculated from

$$C = 10 \frac{VA}{V^2 sec} \frac{60}{f} \text{ in farads} \quad (4)$$

Next one computes the ratio of peak voltage to peak system voltage. Then one uses equation 3 to find the damping factor, x. Finally, one computes the required resistance as

$$R = 2x \, (L/C)^{\frac{1}{2}} \quad (5)$$

The resistor rating in watts and the capacitor working voltage are then calculated based on some relatively simple relationships.

Title: Ground Bed Design (Design Circle, Design Straight Line, Design Counterpoise)

Description: This is actually a series of three programs. Each program addresses the design of separate types of ground beds to achieve a given value of ground bed resistance when several design parameters are assumed and one parameter is left to be determined. The three ground bed configurations are n driven rods along a straight line, n driven rods in a circular arrangement and n buried conductors arranged radially from a common point (see Fig. 2). In the straight line and circular arrangements for driven rods, it is assumed that the individual ground rods are connected by insulated conductors. If the rods are connected by bare conductors, the ground bed resistance will be lower than the calculated value.

Solution Technique[3]:

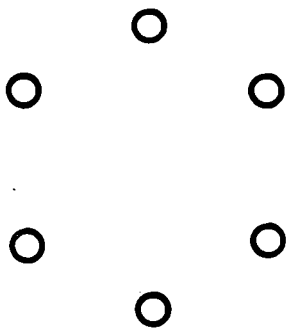
a) n-driven rods in a circle

The basic equation used in the analysis is

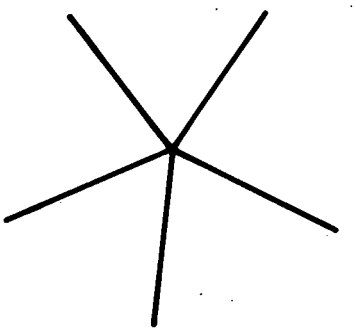
$$R = \frac{1}{n} \frac{\rho}{2\pi\ell} \left[\ln \frac{4\ell}{a} - 1 - \frac{\ell}{D} \sum_{m=1}^{n-1} \frac{1}{\sin m\pi/n} \right] \quad (6)$$

where R = the ground bed resistance in ohms
ρ = the earth resistivity in meter - ohms
n = the number of rods
ℓ = the length of rods in meters
a = the rod radius in meters
D = the diameter of the ground bed in meters

This equation can be solved for any one of the variables when all the other variables are known. In designing a ground bed it is assumed that one knows R, the desired ground bed resistance. Also, the earth



a) n - Electrodes in a Circle



b) n - Buried Counterpoises



c) n - Driven Rods in a straight line

Fig. 2. Three Electrode Configurations for Which Programs Are Developed.

resistivity, ρ, is not controllable unless the earth is salted or location is changed. The rod radius, a, does not affect the ground bed resistance greatly, nor can it be varied over a wide range. Thus, the controllable variables are rod length (ℓ), circle diameter (D), and number of rods (n). The solution may not be possible to obtain by direct means but any of these variables can be solved for iteratively.

b) n driven rods in a straight line

The basic equation used in this analysis is

$$R = \frac{1}{n}\left[R_1(a) + \frac{\rho}{\pi s}\left(\frac{1}{2} + \frac{1}{3} + \frac{1}{4} + \ldots + \frac{1}{n}\right)\right] \tag{7}$$

where

$$R_1(a) = \frac{\rho}{2\pi\ell}\left[\ln\frac{4\ell}{a} - 1\right] \tag{8}$$

s = spacing between adjacent rods in meters and R, ρ, n, ℓ, and a are defined above.

In this case the controllable parameters that can be used in the design are the rod length (ℓ), the number of rods (n), and the spacing between the adjacent rods.

c) n buried counterpoises

The basic equation in this case is

$$R = \left[R_1(a) + \sum_{m=1}^{n-1} R_0\left(\frac{2\pi m}{n}\right)\right] \tag{9}$$

where

$$R(a) = \frac{\rho}{\pi\ell}\left\{\ln\left[\frac{\ell}{a}\left(1+\sqrt{1+(a/\ell)^2}\right)\right] + \frac{a}{\ell} - \sqrt{1-(a/\ell)^2}\right\} \tag{10}$$

and

$$R_o\left(\frac{2\pi m}{n}\right) = \frac{\rho}{2\pi\ell}\ln\frac{1+\sin\frac{\pi m}{n}}{\sin\frac{m}{n}} \tag{11}$$

Other terms used in the equations are the same as those defined above.

In this case, the only controllable variables are the conductor length (ℓ) and the number of radial conductors (n).

The reader should be aware of the limitations of these techniques. All three cases assume a uniform, non-layered earth. The case of rods driven in a circle assumes $D \gg \ell$. The case of the rods in a straight line makes the approximation that all rods carry equal current. In essence, the programs will give an approximate result that should be checked by measurement.

Title: Fuse Coordination
Description: In order to properly apply fuses for overload and fault protection it is necessary that fuses be applied such that they can carry the continuous current of the system they are protecting. In

addition, when a fault occurs it is necessary that the first fuse back toward the source clear the fault before any fuse further back toward the source has a chance to be damaged. These two steps can be accomplished for common types of fuses (K, T, N) by the use of coordination tables and a table of maximum continuous currents (see Table 10 and 12) of the McGraw-Edison Distribution System Protection Manual[4]. A series of programs for use with HP calculators has been written to perform the fuse coordination design. The names of these programs are FUSES, COORD and 6-12T Data, 15-30T Data and 40-140T Data.
Solution Techniques: The solution technique is extremely simple. The programs FUSES and 6-12T Data, etc., contain the information in Tables 10 and 12 of the McGraw-Edison book. The preliminary design based on full load currents consists of reading in Table 10 data, keying in full load currents and making simple comparisons to find preliminary fuse sizes. Then one starts at the end of the radial system and uses the fuse size based on full load current. The next fuse size back toward the source is selected to coordinate with the last fuse for the maximum fault current that can flow through both fuses. This current can be found from a fault study. The fuse size can be found from Table 12 by calling the last fuse the protecting fuse and the fuse closer to the source the protected fuse. One always uses the larger of the two sizes determined by load current and coordination. One then proceeds to the third fuse from the end of the radial system, etc.

Title: Three-Phase Short Circuit (originated by George Fesak of MSHA)
Description: When a three-phase short circuit occurs on a radial system, the fault current can be found by adding up all the contributions to the series impedance (either in ohms or per unit) and dividing the system line-to-neutral voltage by the total series impedance. The contributions to impedance consists of utility equivalent impedance (usually in terms of short circuit level in kVA), transformer winding resistance and leakage reactance, and conductor resistance and reactance. As in all practical short circuit studies, shunt capacitance, loads and magnetizing currents are neglected.
Solution Technique: The solution technique is basically Ohm's Law with a little bit of help from Thevenin's Theorem. One must supply the program with the system line-to-line voltage in kV, the utility short circuit level in kVA, the transformer data (kVA, %X, %R), and conductor input data (voltage in kV, X, and R). The program then computes the magnitude and phase angle of the fault current. One can also recall the fault voltage, total X, total R and total Z.

Title: Relay Coordination
Description: The characteristics of relays are represented by a family of curves such as shown in Fig. 3, which plots the time required for the relay to operate as a function of the overcurrent for different time dial settings.

In order to design a protective system, it is necessary to find the time dial setting which will give a specified delay at a given overcurrent level. To check a system's coordination it is necessary to determine the time delay for a given setting and overcurrent level. This is the inverse problem and is also solved in this program.

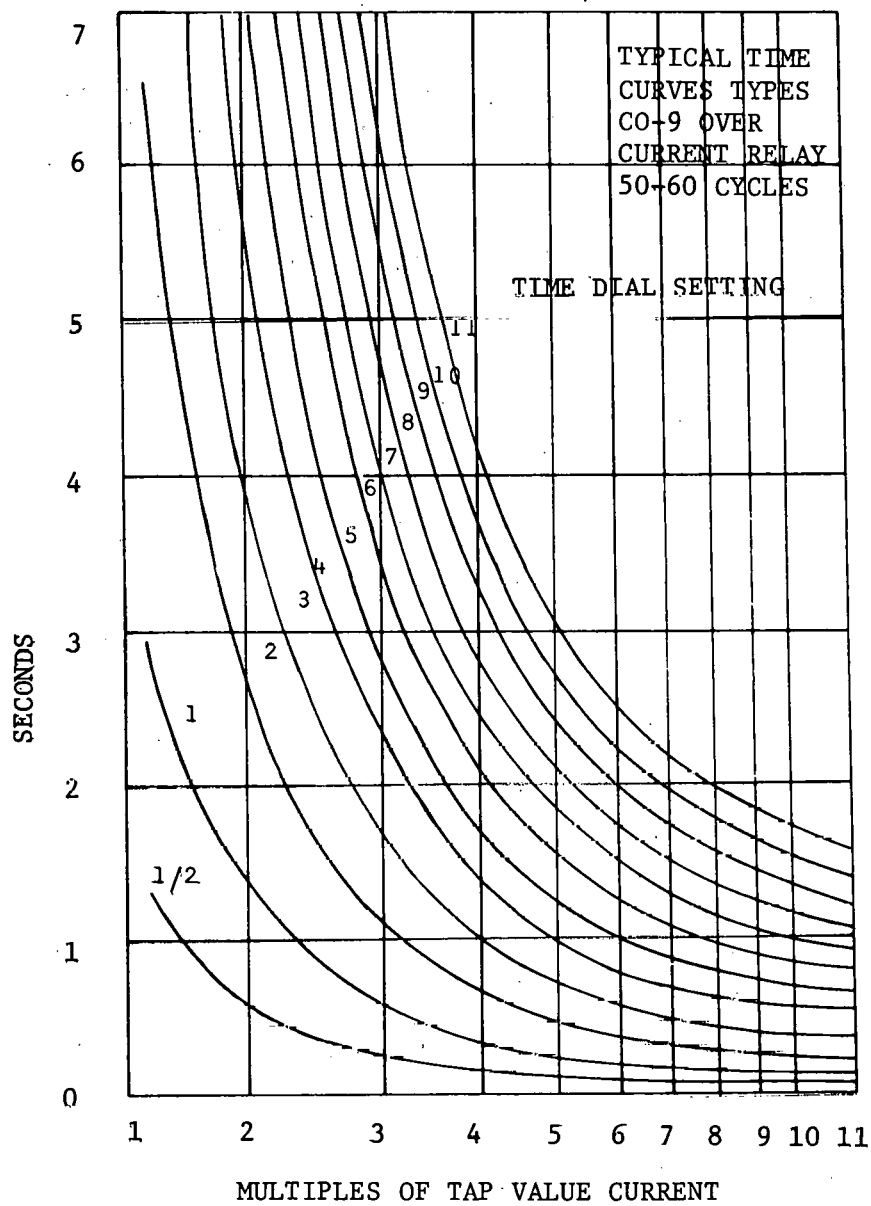


Fig. 3. Typical Time Current Characteristic of a Relay.

Solution Technique: The method of solution is based on an analysis by Hieber[5]. Hieber approximates an individual curve by the expression

$$T_d(P) = \frac{\alpha}{[P-a+bP^{-2P}]^{\beta}} + \gamma - \delta \left(\frac{P}{50}\right)\varepsilon \qquad (12)$$

The constants a and b are taken as fixed for a particular relay, while α, β, α, γ, and ε change with the time dial setting, L. Hieber proposes to evaluate these constants for two time dial settings: 0.5 and 10, and then find the value of T for other dial settings by interpolation.

Next, an equivalent time dial setting is introduced

$$L' = L + q\ (L-10)\ (L-0.5) \qquad (13)$$

where q is a constant characteristic of the relay which makes the interpolation with respect to L' as linear as possible. One then has

$$T\ (P,L) = \frac{10-L}{9.5}\ T_{0.5} + \frac{L^{X}-0.5}{9.5}\ T_{10} \qquad (14)$$

$$L' = 10 - \frac{T_{10} - T}{T_{10} - T_{0.5}} \qquad (15)$$

Equation 13 is then a quadratic for L, the solution of which can be written as

$$L = \frac{L' - 5q}{[(L' - 5q) + (1 - 10.5q)^2]^{\frac{1}{2}} + (1 - 10.5q)} \qquad (16)$$

Title: Instantaneous Trip Settings for Trailing Cables
Description: The purpose of this program is to determine the instantaneous settings for circuit breakers based on the size of trailing cables. The program has been adapted to the HP-67/97, SR-52 and TI-59 calculators. The program can be adapted to the HP 19c/29c but will not fit in the storage capability of the HP-25/25c calculators. The program uses current setting established by standard setting bodies.
Solution Technique: The program is based upon the principle of curve fitting. One simply plots the allowable current settings versus wire size and fits a curve of the form

$$I = 10^{\frac{30.4 - (\text{wire size})}{9.8}} \qquad (17)$$

to the data. This equation works for AWG wire sizes from 14 to 3/0 except for #10 and 2/0. Sizes 4/0 and over are trapped out and determined by a table look up procedure.

REVIEW OF ANALYSIS PROGRAMS

The same format used to present the design programs will be used to present the analysis programs. They will be presented in the same order as they are listed in Table 2.

Title: Prediction of Ground Bed Resistance
Description: Three commonly used ground bed configurations have been incorporated in this program. These configurations are n driven rods in a circle, n driven rods in a straight line and n radially buried conductors (counterpoises) emanating from a single point. It is assumed that the driven rods are connected by insulated conductors. If the rods are connected by bare conductors, as would be expected, the actual ground bed resistance would be lower than predicted. One must merely supply a description of the ground bed and then press a particular key and the ground bed resistance will be computed and displayed.
Solution Technique: Exact equations can be found for the driven rods in a circle and the buried counterpoises because these configurations are symmetrical and all the rods share the current equally. This is not true of the driven rods in a straight line. However, if one assumes that the current is shared equally by the driven rods in a straight line, then a fairly simple expression for ground bed resistance can be used that is quite accurate. The basic equations used in the program are the same as equations 6 to 11 described under ground bed design.

The program calculates the ground bed resistance by direct application of equations 6 to 11 and the previously discussed assumptions of uniform earth resistivity, etc. apply equally well here.

Title: Prediction of Transient Voltages (written by H. W. Hill)
Description: In doing transient studies to determine abnormal voltages on power systems, a great many problems may be reduced to a configuration that can be represented by either a series or parallel RLC circuit. In addition, the responses of series or parallel RLC circuits to common types of excitation (steps, ramps, and impulses) can be reduced to a few standard curves. These curves depend only upon the relative values of the circuit components and the type of transient. Table 3 summarizes the three circuit parameters while Table 4 lists the types of transients and corresponding standard curves numbers. The notation follows Ref. 6 as closely as possible.

TABLE 3

DEFINITION OF CIRCUIT PARAMETERS

Circuit Type	Parameter	Symbol	Description
Parallel RLC only	$2R\sqrt{C/L}$	μ	Damping factor
Series RLC only	$2/(R\sqrt{C/L}$	γ	Damping factor
Both series and parallel RLC	$t/\sqrt{LC}$	t'	Normalized time

TABLE 4

Circuit	Transient Input	Response	Standard Curve
Parallel RLC	Current Impulse	Voltage	1
Parallel RLC	Current Step	Voltage	3
Parallel RLC	Current Ramp	Voltage	4
Series RLC	Voltage Impulse	Current	1
Series RLC	Voltage Step	Current	3
Series RLC	Voltage Ramp	Current	4

Solution Technique: The solution technique is basically the solution of a second order differential equation with various driving functions. For the cases listed in Table 4, three separate solutions result. These solutions are summarized in the three attached figures which are reproduced from Ref. 6. The program has three separate options:

i) One can compute the value of a standard curve for a specific value of normalized time.
ii) One can compute the maximum value on a specific standard curve (maximum voltage or current)
iii) One can find a value of μ or γ for a given peak value of current or voltage. This option is useful in designing closing resistors on EHV systems.

Title: Device Coordination Checking
Description and Solution Technique: These are the same as those for the program entitled "Relay Setting Selection." This program combines both design and analysis options.

Title: Short Time Equipment Rating
Description: All of the directly available data on cable ampacity generally state the allowable steady-state current level for a specific application that will lead to an allowable temperature rise based on a given ambient temperature.

In reality, cable insulation does not have an allowable temperature rise but only an allowable final temperature. Thus, one is actually interested in final temperature and he can calculate the allowable temperature rise as the allowable temperature minus the ambient temperature. The concept of temperature rise comes from theoretical calculations, since if one computes the power loss in the cable he can compute the temperature rise needed to transfer that amount of power from the cable to the surrounding environment in the steady state.

One important parameter is the short time rating of a equipment that has been operated at a steady (or no) load level for a long period of time. The goal is to calculate the allowable overload factor for a specified period of overload.

In the ensuing calculations, it is necessary to know the thermal time constant for the equipment, or its reciprocal, a, and the

temperature rise resulting from one particular value of steady-state current. These quantities can be found by carrying out a very simple laboratory experiment. The purpose of this calculation is to determine the allowable overload factor, β, for any equipment that has a known initial temperature rise, θ_0, and an allowable final temperature rise, θ_F, if the overload has a duration Δt. It is assumed that the thermal time constant, τ, is known.

Solution Technique: It is shown in the literature that the temperature rise of equipment as a function of time can be expressed as

$$\theta = (\theta_F - \theta_0)(1 - e^{-at}) + \theta_0 \tag{18}$$

The final temperature rise, θ_F, is a function of the square of the steady-state current. In fact, one can state that

$$\theta_F = k\, I^2 \tag{19}$$

Thus, if one knows the temperature rise for any current level he can find k and hence the temperature rise at any current level.

For the problem at hand, one wishes to know the overload factor, β, which is allowable for a duration Δt, such that the allowable temperature rise will not be exceeded. β is the ratio of the short duration overload current to the steady-state current that causes the allowed temperature rise. In order to use Equation 18, assume an unknown current, I', flows that would cause an ultimate temperature rise of θ_F'. Assume this current is adjusted to cause the temperature rise θ_F' at the end of the overload period Δt to equal the allowed temperature rise, θ_F. Then

$$\beta = \frac{I'}{I_{rated}} = \sqrt{\frac{\theta_F'}{\theta_F}} \tag{20}$$

Title: Cyclic Equipment Loading

Description: The purpose of this program is to determine the temperature profile of a piece of equipment for a single cycle of a known load cycle from a cold start and the maximum temperature in the steady state if the load cycle occurs repetitively. This program is applicable to any equipment where the cooling effects are not load dependent. Examples of rate dependent cooling are series would motors and temperature controlled cooling systems. Therefore, this program is applicable to static colled transformers, constant or nearly constant speed machines, forced cooled equipment, where the forced cooling is not controlled by any means, etc.

The problem is idealized by assuming that the load cycle consists of five periods of constant current with known durations. The load cycle is uniquely determined by the current levels I_0, I_1, I_2, I_3 and I_4, along with the corresponding time periods, t_0, t_1, t_2, t_3 and t_4. In order to do the calculations, one must also know the equipment's thermal time constant, τ, or its reciprocal, a, the temperature rise for rated current, θ_F, and the ambient temperature θ_0.

Solution Technique: The basic for the solution is that for a given current or load, the rise in temperature is proportional to the current squared. The temperature rise is in the form of an exponential rise. The system of equations used in the program is as follows:

$$\theta_{n+1} = (k\, I_n^2 - \theta_n)(1-e^{-at_n}) + \theta_n + \theta_0 \qquad (21)$$

where

θ_0 = ambient temperature - °C
θ_n = temperature after nth period - °C
a = the reciprocal of the thermal time constant = hr^{-1}
t_n = nth time period in hr
k = heating constant in $°C/A^2$

If $t_0 \rightarrow t_4$ are in minutes or seconds then a will be in min^{-1} or sec^{-1}, respectively.

The program does basically two calculations. First, from a cold start it calculates $\theta_1 \rightarrow \theta_5$ for a single cycle of load. Then it calculates $\theta_1 \rightarrow \theta_5$ for repetitive cycles until the temperature at the end of successive load cycles converge to a steady value. The program then displays the peak temperature attained.

The program works by repetitively applying the basic equation of the form

$$\theta = (kI^2 - \theta_1)(1-e^{-at}) + \theta_1 + \theta_0 \qquad (22)$$

This equation is applied to get the final temperature from the initial temperature during a period of constant load. The final value of temperature is then used as the initial temperature for the next load period.

Title: Starter Size Adequacy
Description and Solution Technique: This program is identical to the program entitled: "Selection of NEMA Size Contactors".

CONCLUSIONS

More than a dozen programs useful in design and safety checking of mine electrical power systems have been described along with the solution techniques used. Due to space limitations, instructions for program use, examples and listings available in USBM reports were not included.

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A PROCEDURE FOR STRAY-VOLTAGE SURVEYS IN UNDERGROUND COAL MINES

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ABSTRACT

Stray voltage between grounded mining equipment and local earth is a common problem, especially in mines employing direct-current rail haulage. This paper describes a general procedure for identifying stray-voltage sources for those cases not resolved by normal maintenance activities.

Under the procedure, a survey of stray-voltage waveforms, observed with a portable oscilloscope, is made throughout the mine. The waveforms are broken down into their component parts, and these parts are correlated with mining activity and equipment operation. The probable causes of these components are then given a priority ranking on the basis of their contribution to the total stray voltage and the cost of eliminating them as an aid to planning corrective action. A series of electrical practices for avoiding or reducing stray voltage problems is presented.

INTRODUCTION

The external structure (frame) of every piece of mining equipment receiving power from the underground electrical distribution system is connected by the "frame ground system" to a ground bed buried in the earth outside the mine. The frame ground electrical conductors are carried in the underground high-voltage distribution cables to equipment such as haulage power supplies, switches, and section power centers. The frame ground conductors are carried in medium- or low-voltage trailing cables from section power centers to mining machines such as roof drills and continuous miners.

Stray voltage is a voltage between the equipment frame and the roof, rib, or floor (Figure 1b) of the mine, resulting from unintended circuits or extraneous electric currents in the earth. Stray voltage occurs even though all the mine electrical equipment may be in good working order and all mining equipment and operations comply with Mining Enforcement and Safety Administration regulations. Stray voltage must be controlled to an acceptable level because

1. Stray voltage can be large enough to shock workers. This presents a personnel hazard, primarily through startling persons working with or repairing mining equipment.

2. Stray voltage can reduce productivity by causing machine operators to distrust their equipment or by distracting their attention.

3. Stray earth currents associated with stray voltages accelerate corrosion of rails, water lines, and electrical equipment contacting the mine floor or roof.

4. Stray currents associated with stray voltage can cause nuisance tripping of ground-fault monitors.

Sensitivity to electric shock depends on an individual's physical characteristics and working environment. Furthermore, people are sensitive to the instantaneous peak voltage rather than an average as would be indicated on an ordinary multimeter or a blaster's meter. No Federal safety regulations deal directly with stray voltage. However, an intention to limit stray voltage to 100 volts for high-voltage equipment and 40 volts for medium- and low-voltage equipment can be inferred.[1)]* Experience and experiment have shown that limiting the peak stray voltage to 15 volts is advisable in mines where workers are using frame-grounded equipment in a wet environment.

INITIAL PROCEDURES

If objectionable stray voltage is encountered, the following procedures are likely to disclose its principal cause and are good electrical practice in any case.

1. In mines using rail haulage, the track bonds should be inspected and restored where needed, and any inadvertent contact between the trolley line and the roof should be eliminated.

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* See References.

2. The frame-ground system should be inspected, and accidental contact with the mine floor, ribs, and roof should be eliminated. Such contacts frequently occur at portable substations and along messenger wires supporting underground high-voltage cables, Figure 1c.

3. Recent changes in the electrical system equipment and mining operations should be reviewed.

If these procedures fail to locate and/or eliminate the source of the persisting stray voltage, a survey of stray voltage throughout the mine should be made.

SURVEY AND ANALYSIS OF STRAY VOLTAGE

A portable oscilloscope (Figure 1a) is used to make a stray-voltage survey of the mine. The oscilloscope is connected between the surface of any piece of grounded mine equipment and a nearby roof bolt or hanger (Figures 1b, 1c). The observed waveforms (Figure 2) are correlated with ongoing activities throughout the mine. Although stray-voltage measurements should be concentrated along the frame ground path leading to the problem site, it is advisable to survey the entire mine (Figure 3) in an effort to associate certain components of the stray voltage with geographic areas of the mine.

The stray-voltage waveform can be analyzed, or broken into its components, as an aid to separating the causes of the stray voltage. The following voltage-component classifications have proven useful, and the accompanying examples, while not exhaustive, indicate how such voltage components might arise.

An average voltage component (Figure 4) results from the flow of direct current. Thus, the average voltage must be associated with the DC haulage system, DC power centers, direct currents flowing in the earth, or a combination of these. Figure 5 shows how a mine-wide stray average voltage can result from direct current leaking into the frame ground system and causing a voltage in the ground bed. Measurement of the currents in the frame ground branches feeding into the ground bed made with a portable direct-current clamp-on ammeter (Figure 6) can indicate which mine area needs maintenance. A simply constructed test lamp (Figure 7) can be used to quickly test trolley-line insulators that may be leaking current into the roof.

A changing average voltage, varying during several seconds to a minute, is usually associated with the movement of locomotives on the rail haulage system. Figure 8 shows how a voltage drop in the rails can be carried to the working face. The resultant stray current flowing in the frame ground system causes a voltage drop in the ground bed, and a stray voltage appears throughout the mine.

Constant-amplitude alternating voltage is further divided into a 60-hertz component (Figure 9) and higher frequency components. A steady 60-hertz voltage component, often accompanied by nonrepetitive high-frequency components, indicates a low-current ground fault. A large 360-hertz voltage component (Figure 10) is usually associated with rectification "ripple" on the trolley line. Distributed inductance along the rail and frame ground lines can exaggerate the 360-hertz component compared with the average voltage component of a leakage current.

Varying amplitude alternating voltage is also divided into a 60-hertz component and higher frequency components. Alternating voltages that increase as more current is drawn by underground machinery are usually inductively coupled into the frame ground system. Starting the larger AC motors on continuous mining machines, for example, induces a large 60-hertz component of stray voltage, responsible for intermachine arcing problems.[2] Starting a locomotive often induces stray-voltage "spikes" (Figure 11) in the frame ground system as the current switches from rectifier to rectifier within the haulage power supply. The spikes increase in amplitude along the high-voltage feeder cable until the loaded power supply is reached (Figure 12). Conductor transpositions along the feeder cables usually reduce inductively coupled stray voltage.

Fast transient voltages, nonrepetitive stray-voltage variations lasting less than a second, may be due to opening or closing switches, intermittent failure of an electrical component, or an electrical fault. These infrequently recurring events require long-time monitoring for detection, but once observed, they are usually easy to correlate with the operation of specific mine equipment. Figure 13 shows a stray-voltage transient induced into the frame ground system as a result of the misfiring of an ignitron in an old haulage rectifier unit.

These examples show that stray-voltage problems often include a major portion if not all of the frame ground system. The stray-voltage survey and analysis of the stray-voltage waveforms indicate probable causes of the voltage. Identification of the causes is confirmed by successful corrective action that reduces the stray voltage. The following practices are helpful in preventing or reducing stray-voltage problems.

PRACTICES THAT REDUCE STRAY VOLTAGE

The most obvious and important practices are to

1. Keep the trolley-line and high-voltage cable messengers clear of the roof and ribs.

2. Maintain the electrical bonds on the haulage rails.

To inhibit current leakage into the frame ground system at the working faces,

3. Eliminate metallic conduction paths between the water line and mining machinery.

4. Insert nonconducting sections into the water line near wet areas of the mine to increase its electrical resistance.

5. Prevent electrical contact between the shuttle cars and the main haulage equipment.

To inhibit magnetic coupling between the high-voltage conductors and the frame ground systems,

6. Transpose the high-voltage cable conductors every 1000 feet (300 m) as the cables are laid.

7. Do not run high-voltage cables or main power cables in steel conduits or troughs.

To keep the DC and frame ground lines separated at DC power supplies for rail haulage,[3)]

8. Disconnect frames of haulage power supplies from conductors feeding the rails.

9. Place the haulage power supply a minimum of 25 feet (8 m) from the grounded rail.

10. Insulate the positive and negative power conductors from the power supply to the rail and trolley connections.

11. Frame ground the haulage power supply.

12. Remove metallic objects that could provide leakage paths between the power supply and the grounded rail from the power supply site.

13. Elevate all water pipe a radial distance of 50 feet (15 m) around the power supply, placing it on concrete or cinder blocks with a piece of substation matting between the pipe and the block.

14. Remove spare parts, such as coils and fuses, from the enclosures of the haulage power supply to minimize the

possibility of elevating the frame ground to full potential above the rail.

CONCLUSIONS

Stray-voltage problems are more common in mines employing DC rail haulage, simply because of the current leakage paths between the DC haulage system and the frame ground system. These paths include the rail-floor distributed contact and the inevitable contacts between the trolley line and the mine roof. In most cases, normal maintenance of the rail system, frame ground system, and electrical equipment will control stray voltage to an acceptable level.

In the case of a persistent problem, a stray-voltage survey of the mine and analysis of the stray-voltage waveforms can provide helpful clues in locating the causes. The stray voltage is often the sum of components arising from several causes. In planning corrective action to reduce the voltage, it is useful to give a priority ranking to the probable causes, considering both the magnitude of their contribution to the stray voltage and the cost of each correction.

ACKNOWLEDGMENT

The authors acknowledge the courtesy and cooperation of A. A. Moran, General Superintendent, Gary District - U. S. Steel; J. Caffrey, Chief Engineer; F. C. Bacho, Superintendent, No. 2 Mine; and J. Elswick, General Maintenance Foreman, who provided the facilities and manpower to carry out this investigation. The authors also acknowledge the efforts and cooperation of V. L. Kitts, Sr., Electrical Engineer; and R. E. Puckett, Management Trainee; who assisted in performing the analyses and underground measurements in an operating mine to verify the models and results presented in this paper.

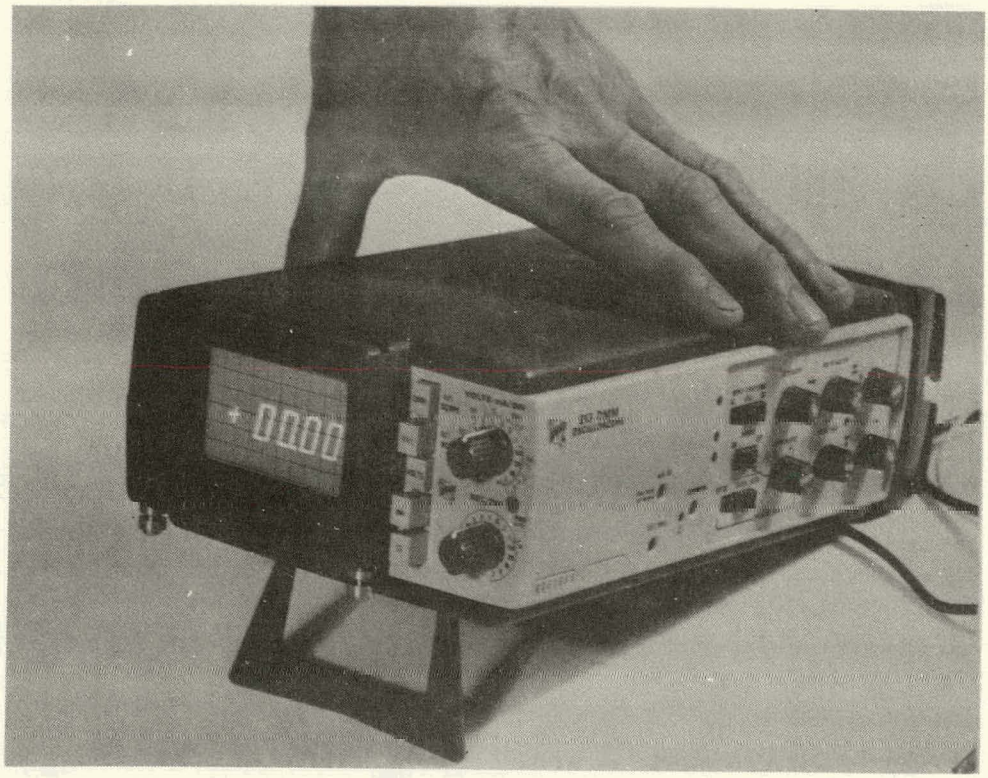
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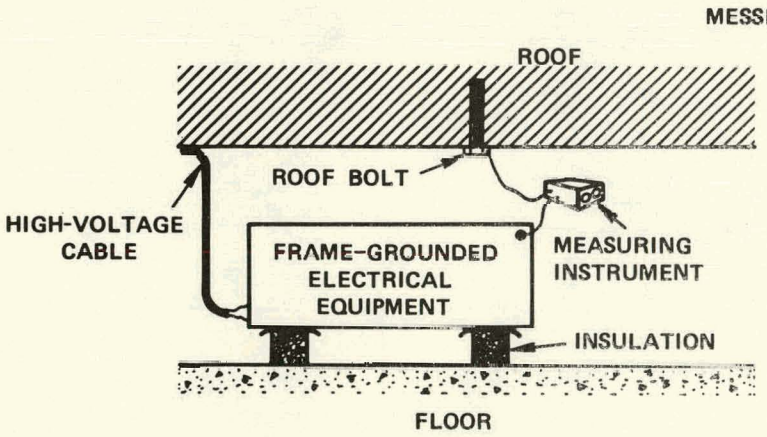
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It is understood that the material in this paper is intended for general information only and should not be used in relation to any specific application without independent examination and verification of its applicability and suitability by professionally qualified personnel. Those making use thereof or relying thereon assume all risk and liability arising from such use or reliance.

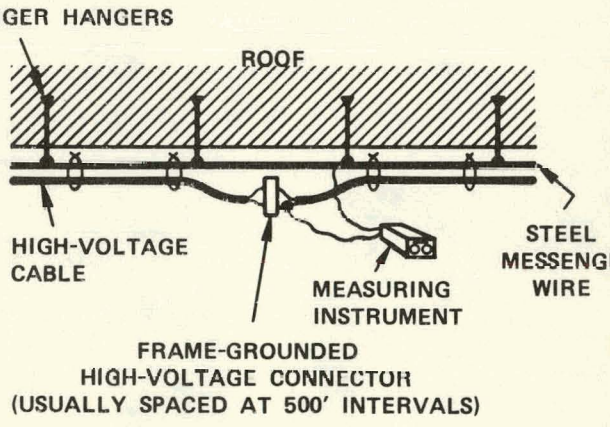


A. PORTABLE OSCILLOSCOPE FEATURING AN INTEGRAL DIGITAL MULTIMETER.

1A. PORTABLE OSCILLOSCOPE FEATURING IN INTEGRAL DIGITAL MULTIMETER



1B. MEASUREMENT AT UNDERGROUND ELECTRICAL EQUIPMENT

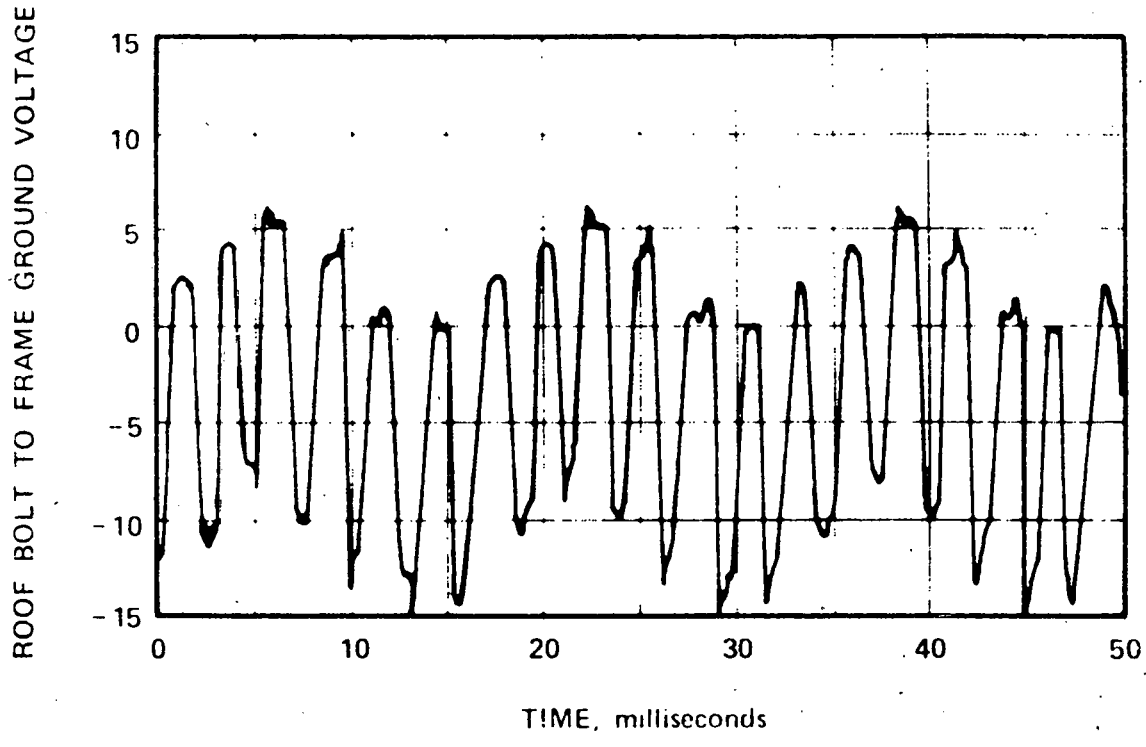


1C. MEASUREMENT AT A HIGH-VOLTAGE CABLE CONNECTOR

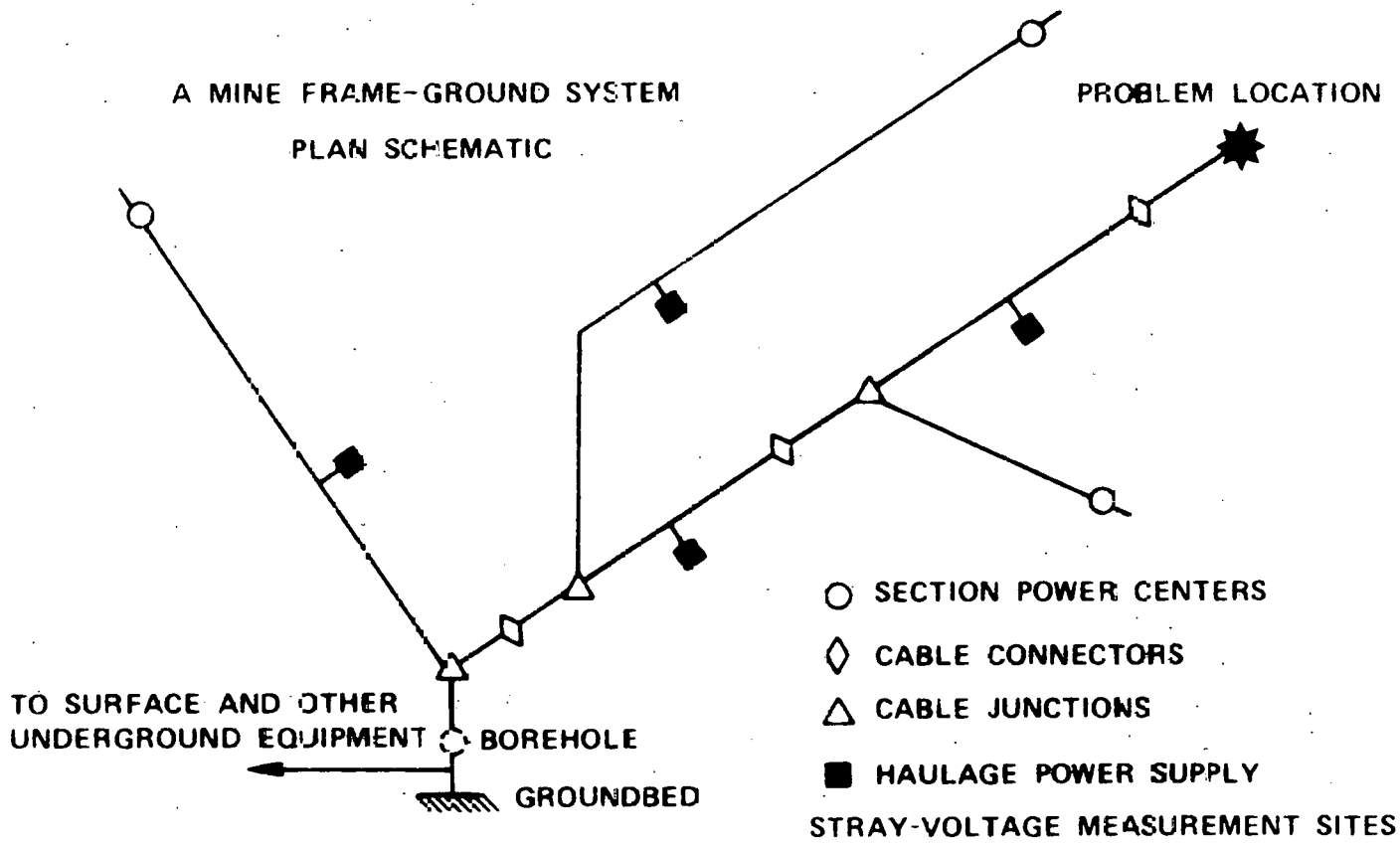
MEASUREMENT OF STRAY VOLTAGE

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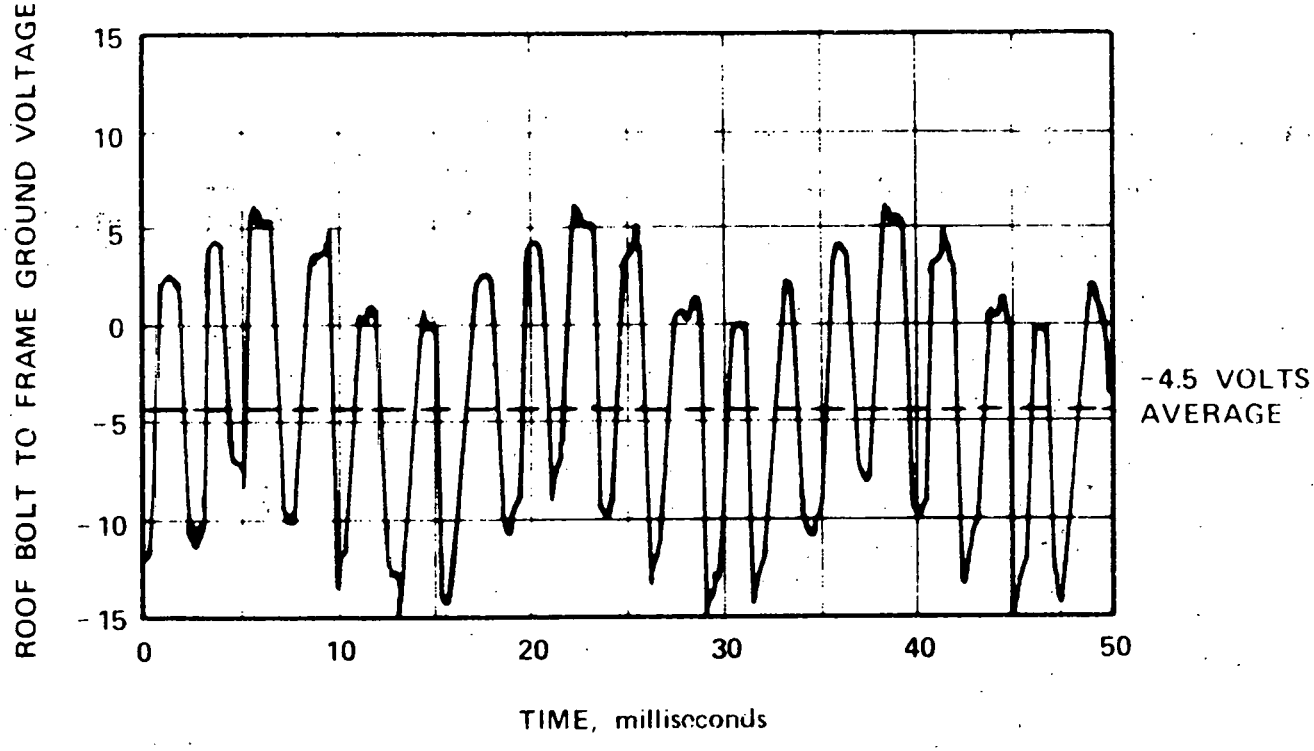
Figure 1



OSCILLOGRAPH OF THE ROOF BOLT TO FRAME GROUND STRAY VOLTAGE IN AN OPERATING MINE

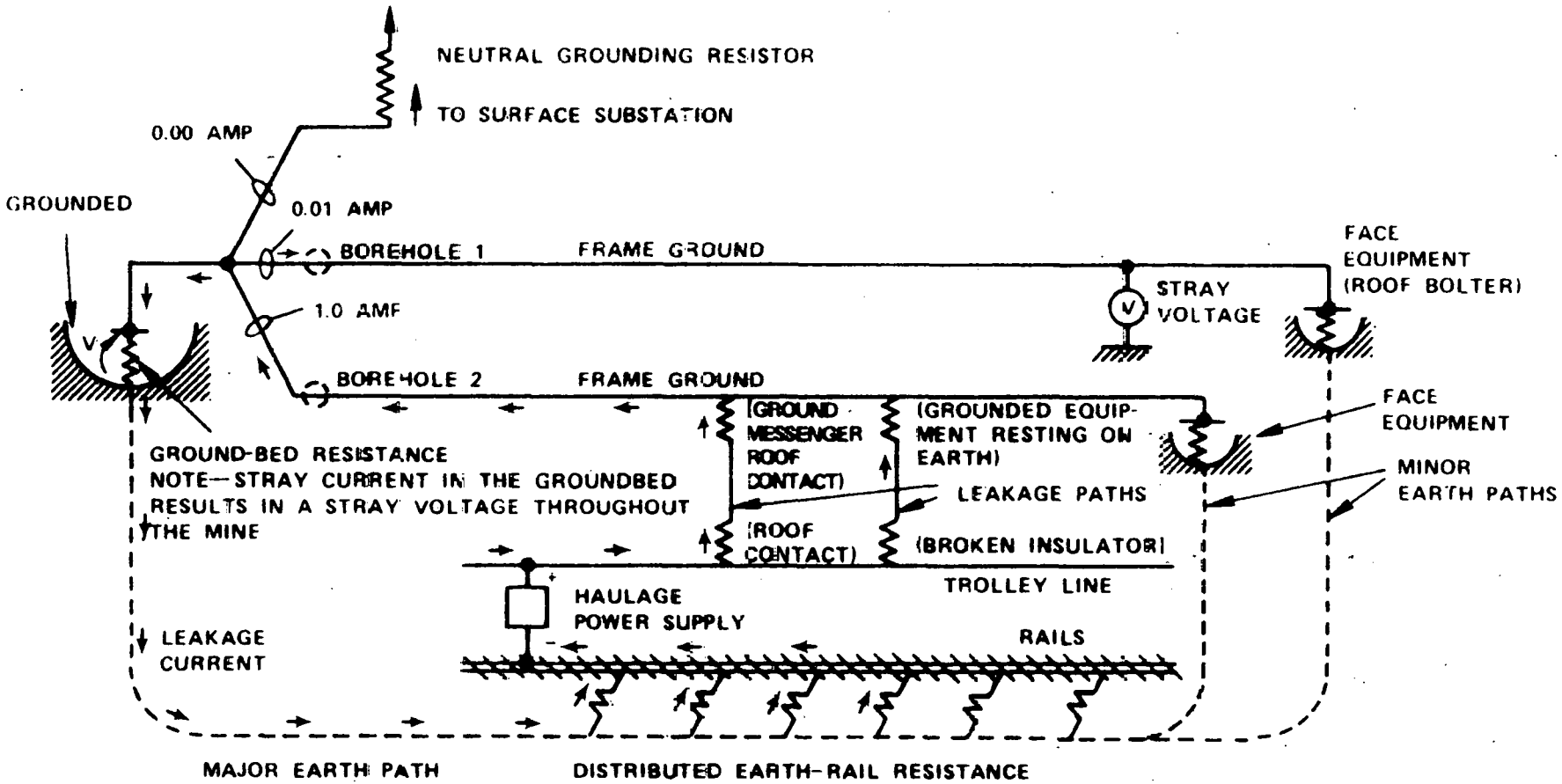


AN INITIAL STRAY-VOLTAGE SURVEY OF A MINE



AN OSCILLOGRAPH OF STRAY VOLTAGE
SHOWING THE AVERAGE VOLTAGE

Figure 4



NOTE THAT CLIP-ON DC AMMETER MEASUREMENTS INDICATE BY MAGNITUDE AND CURRENT DIRECTION WHICH BRANCH OF THE FRAME GROUND SYSTEM REQUIRES MAINTENANCE

STRAY VOLTAGE CAUSED BY A DIRECT LEAKAGE CURRENT FLOWING THROUGH THE GROUND BED

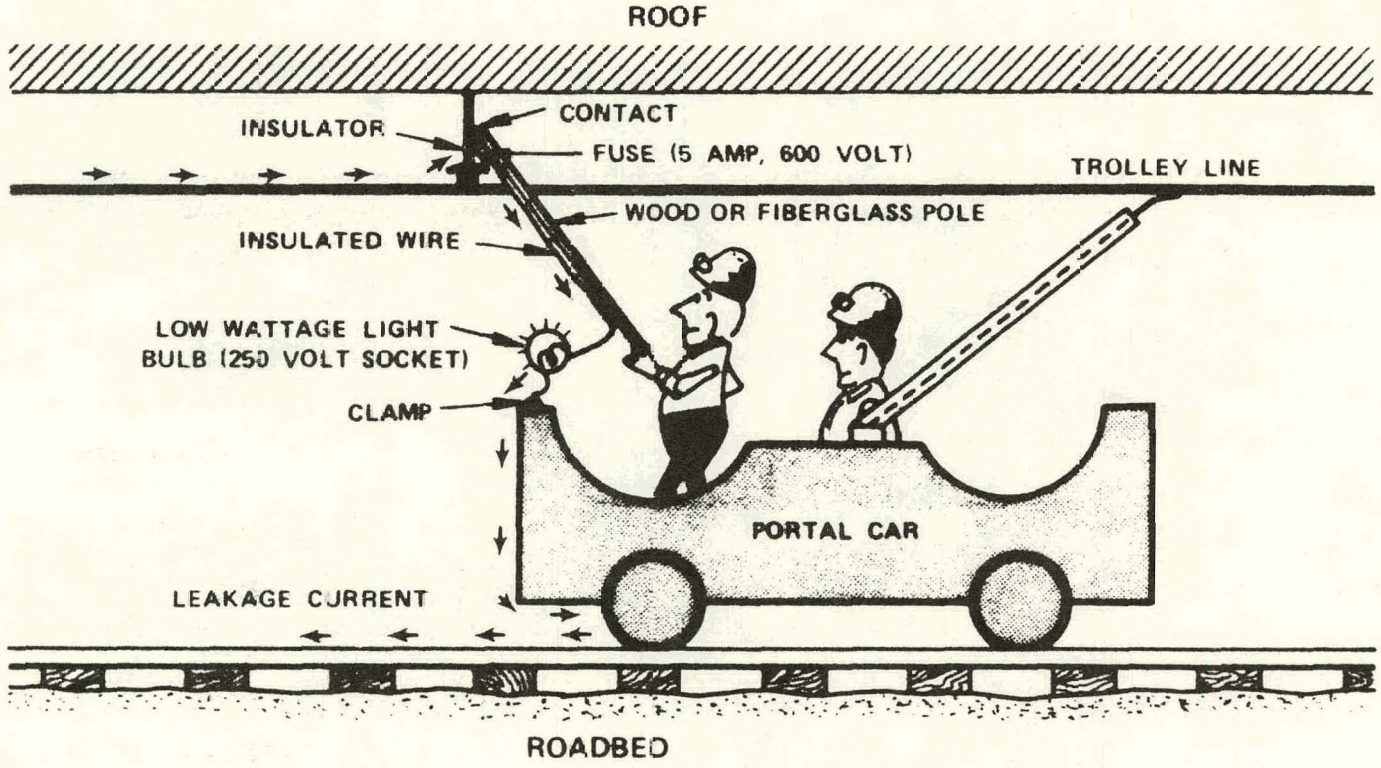


A CLAMP-ON AC-DC AMMETER.

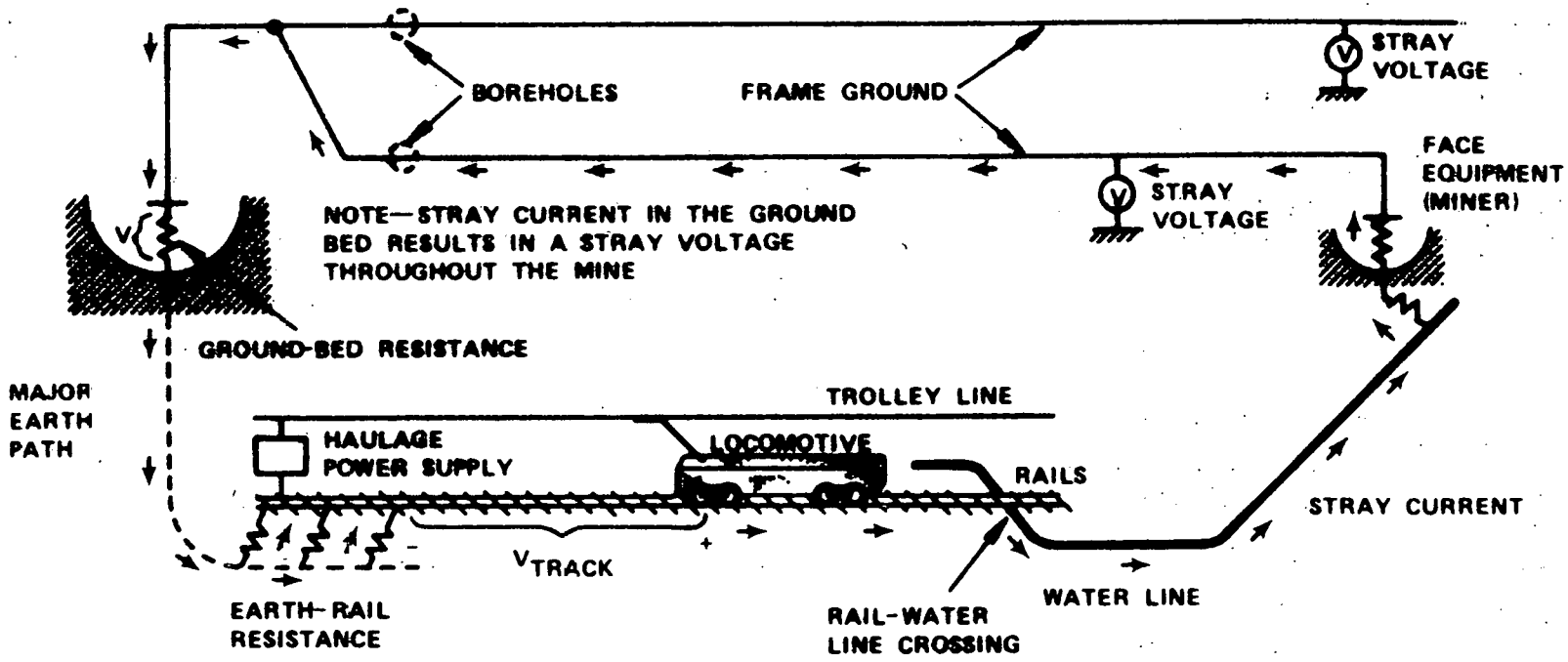
THIS PORTABLE INSTRUMENT MEASURES CURRENTS FROM 0.01 TO 2000 AMPS WITHOUT HAVING TO BREAK THE CURRENT-CARRYING CONDUCTOR

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Figure 6



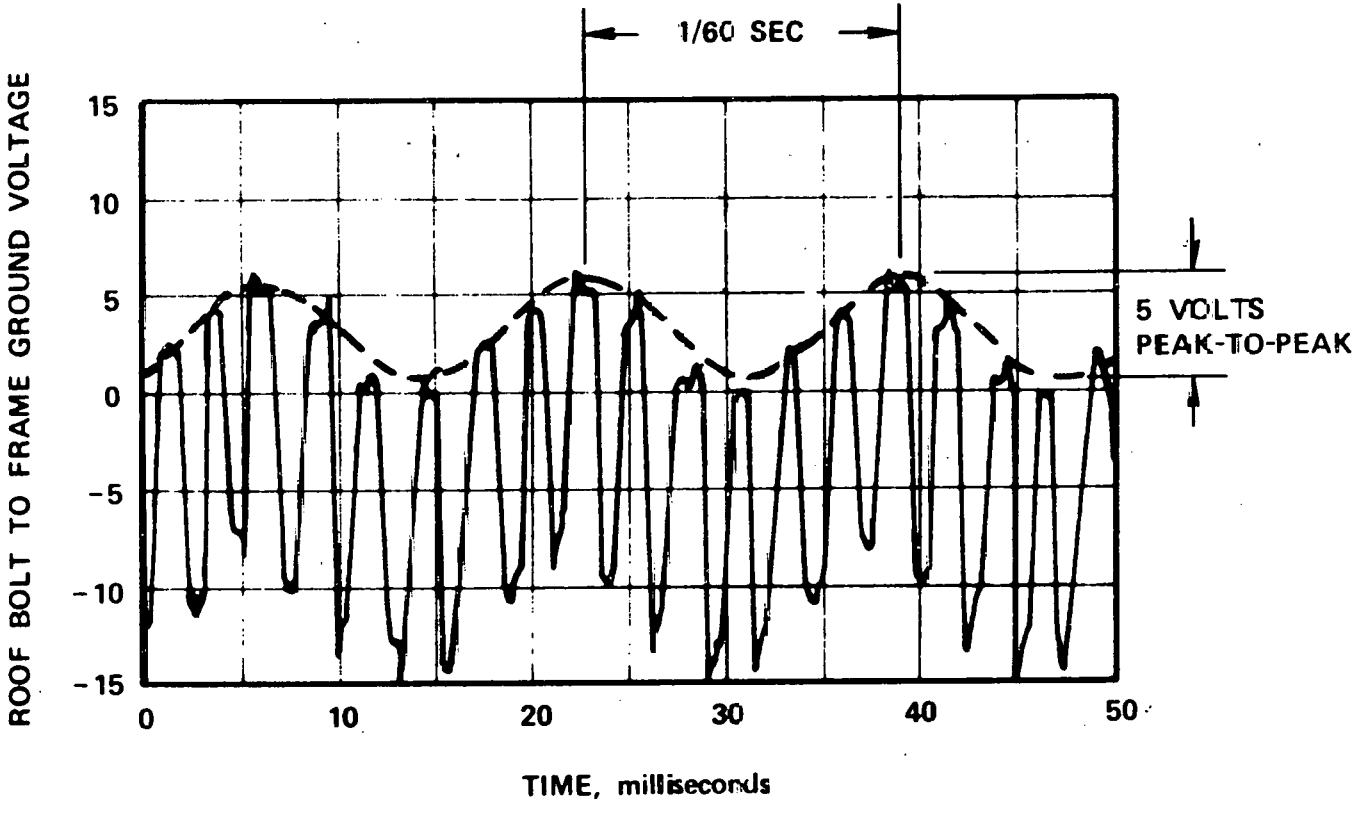
LOCATING FAULTY TROLLEY-LINE INSULATORS



STRAY VOLTAGE CAUSED BY A VOLTAGE DROP IN THE HAULAGE RAILS
THE RAIL VOLTAGE DROP IS CARRIED TO THE WORKING FACE BY A
WATERLINE, CAUSING A STRAY DIRECT CURRENT IN THE GROUND BED

RL-10-2857

Figure 8

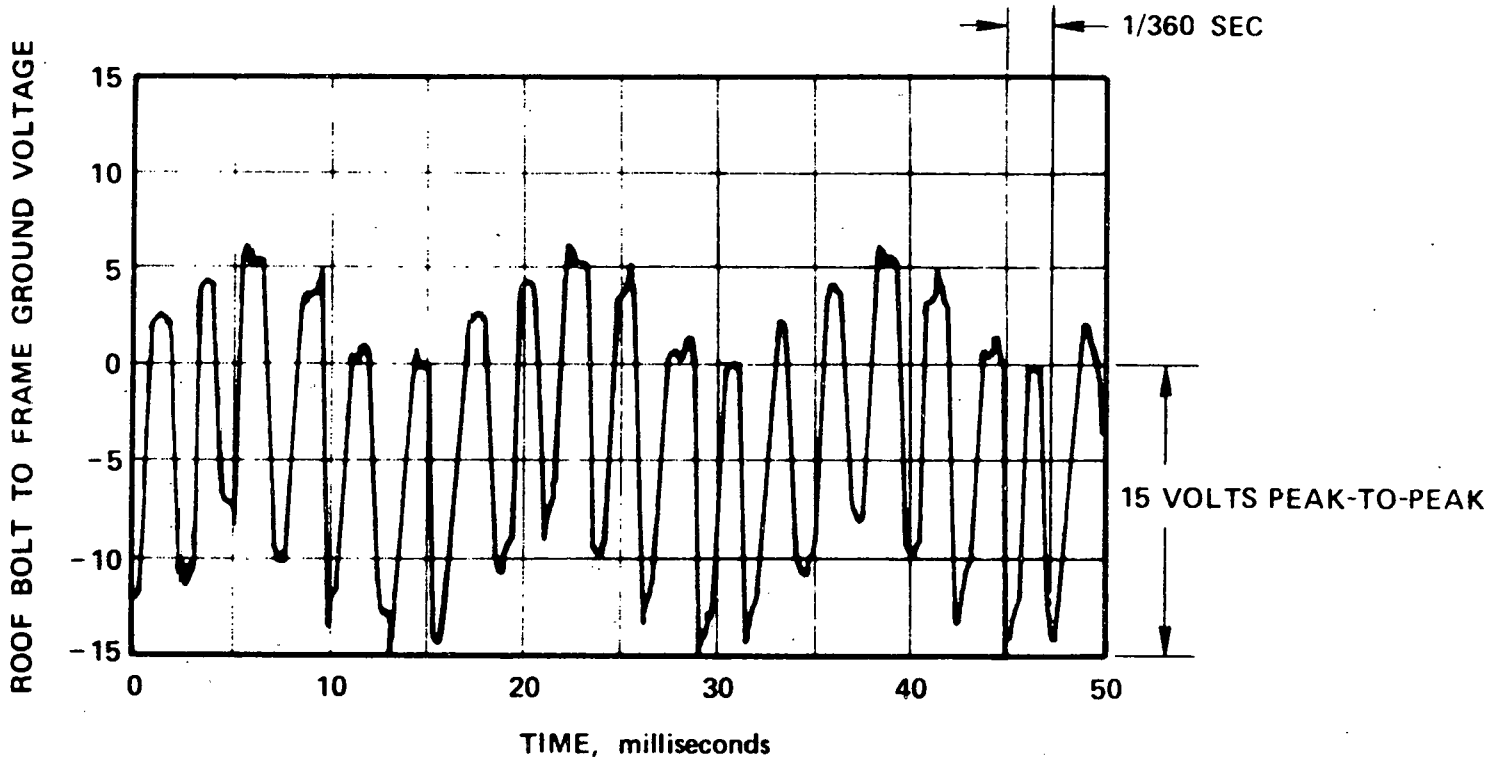


$$\text{PEAK VOLTAGE} = \frac{\text{PEAK-TO-PEAK VOLTAGE}}{2}$$

PEAK VOLTAGE = 2.5 VOLTS

AN OSCILLOGRAPH OF STRAY VOLTAGE SHOWING THE 60-HERTZ COMPONENT

Figure 9



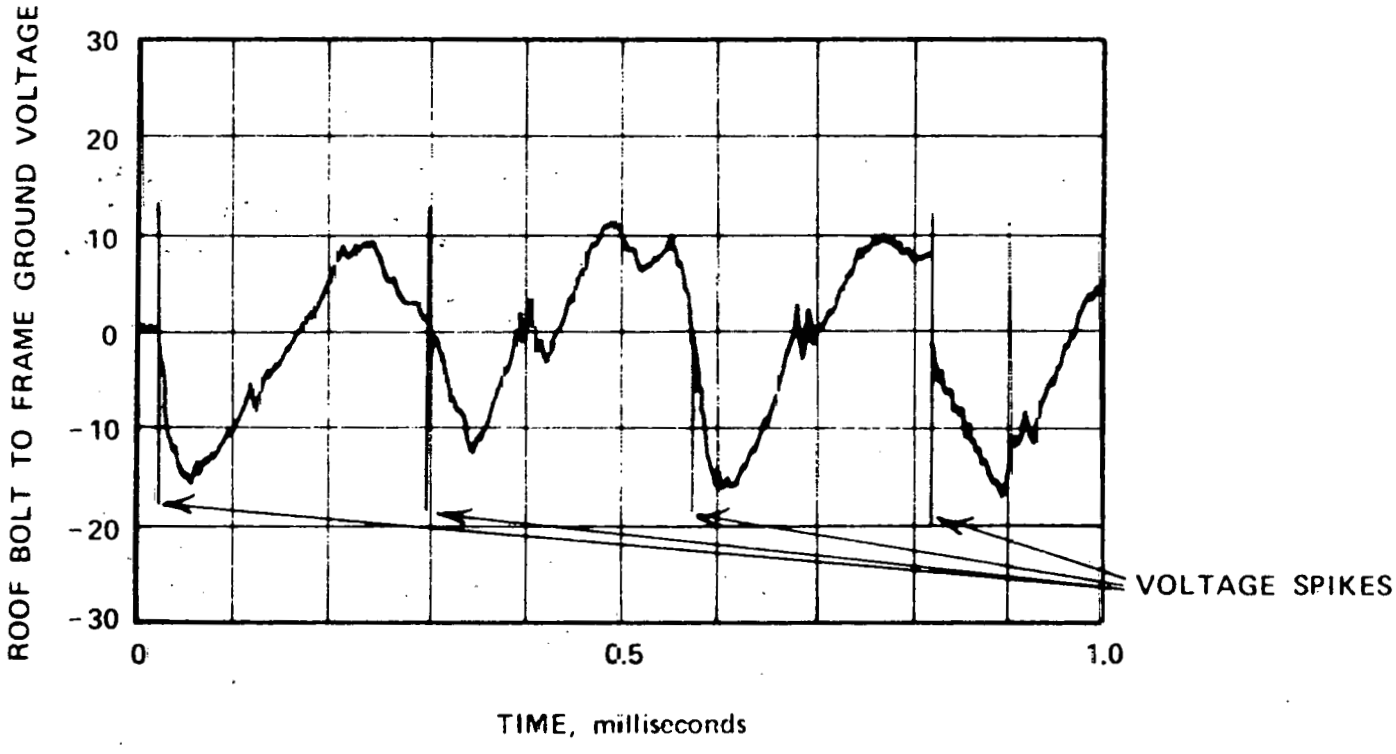
AN OSCILLOGRAPH OF STRAY VOLTAGE SHOWING A STRONG 360-HERTZ COMPONENT

$$\text{PEAK VOLTAGE} = \frac{\text{PEAK-TO-PEAK VOLTAGE}}{2}$$

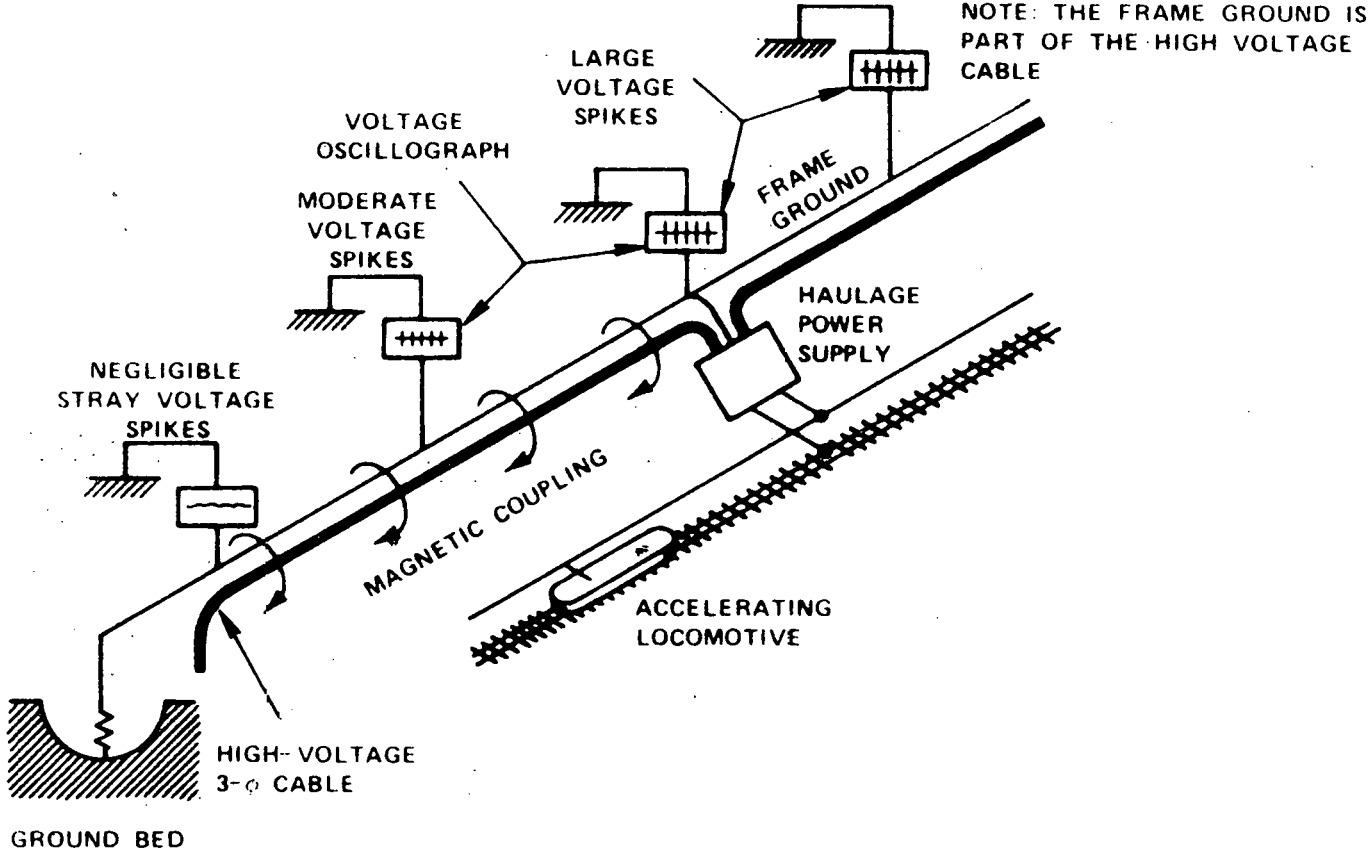
PEAK VOLTAGE = 7.5 VOLTS

RL-10-2859

Figure 10



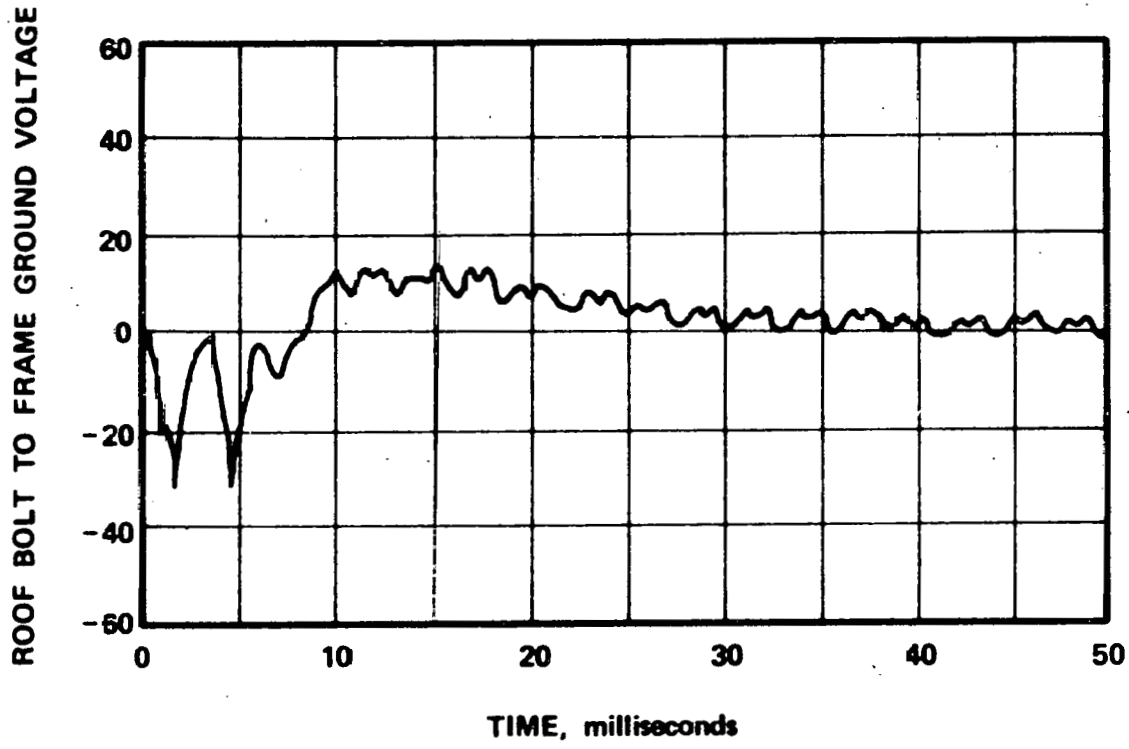
AN OSCILLOGRAPH SHOWING VOLTAGE SPIKES INDUCED INTO THE FRAME GROUND SYSTEM DURING THE ACCELERATION OF A LOCOMOTIVE



MAGNETIC INDUCTION OF STRAY VOLTAGE INTO THE FRAME GROUND SYSTEM

RL-10-2861

Figure 12



OSCILLOGRAPH OF A FAST TRANSIENT STRAY VOLTAGE INDUCED INTO THE FRAME GROUND SYSTEM

A NEW PNEUMATIC DEVICE FOR TESTING

MOLDED-CASE CIRCUIT BREAKERS

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ABSTRACT

One of the ways the reliability of circuit breakers could be assessed is by conducting suitable life tests on them. A pneumatic device required for this purpose has been recently built. This paper discusses the operational functions, the capabilities of this device and types of data to be collected. Briefly, the device has been designed to test three molded-case circuit breakers automatically in an accelerated manner.

INTRODUCTION

West Virginia University (WVU) has been conducting research since 1974 on the reliability assessment of molded-case circuit breakers and other protective devices used in coal mine power systems. This research has been funded by United States Bureau of Mines (USBM), through Grant No. G0144137, with George Conroy as the technical project officer. A three-prong approach has been adopted in the research. These approaches are theoretical, field and laboratory assessments. A number of reports[1-2] and papers[3-6] have been written on this research work.

The aim of the laboratory approach is to conduct life tests on the circuit breakers. It is the understanding of the authors that the molded-case circuit breakers used in the coal mining industries are not required to conform to Underwriters Laboratory (UL) Specifications[7]. Thus, in order to perform meaningful laboratory tests, the UL guides have been taken into account to conduct the life tests. In order to obtain results within a reasonable time, the life tests will be accelerated as explained later.

A pneumatic device required for this purpose has recently been built for conducting the tests in the WVU laboratory. It was jointly designed by the USBM personnel and WVU along with Tool-O-Matic, Inc. (Zelionople, Pennsylvania) personnel. The aim of this paper is to expose the functions and capabilities of the device to all those concerned with the coal mine power systems.

PNEUMATIC DEVICE

Functions:

The pneumatic device is capable of performing the following important functions:

1. Three molded-case circuit breakers of the same type and size can be tested simultaneously and automatically.

2. It is possible to test most of the ac (<1000 V) and dc (<500 V) molded-case circuit breakers in the 100-1200 A range, used in the coal mining industry.

3. In the event any breaker does not operate successfully, the entire testing process can be shut down. Then that particular breaker which failed to operate successfully can be identified along with the cause of failure.

4. It is possible to vary the cycle time of the test in order to accelerate the testing process. This also implies applying different levels of stress during the testing of the breakers.

Description:

The pneumatic device has the following dimensions:

Height - 5 ½ feet
Width - 4 feet
Depth - 2½ feet.

The necessary circuit and parts' diagrams along with the instructional manual are available with Dr. S. S. Venkata, Department of Electrical Engineering, West Virginia University, Morgantown, West Virginia.

The approximate cost of the device is about $5,660.00.

The pneumatic device consists of a control unit, a pneumatic drive unit and a power unit. Figures 1, 2 and 3 are the photographs of the control, pneumatic drive and power units, respectively. Figure 4 is the block diagram of the device showing the arrangement of the three units.

The control unit basically consists of the following sub-units:

Input Power Controller
Test Voltage Selector
Input Indicator
Undervoltage Release (UVR) Controller
Cycle Start-Stop Unit
Breaker Energizer
Output Indicator
Cycle Time Controller
Malfunction Detector
Electromechanical Counters

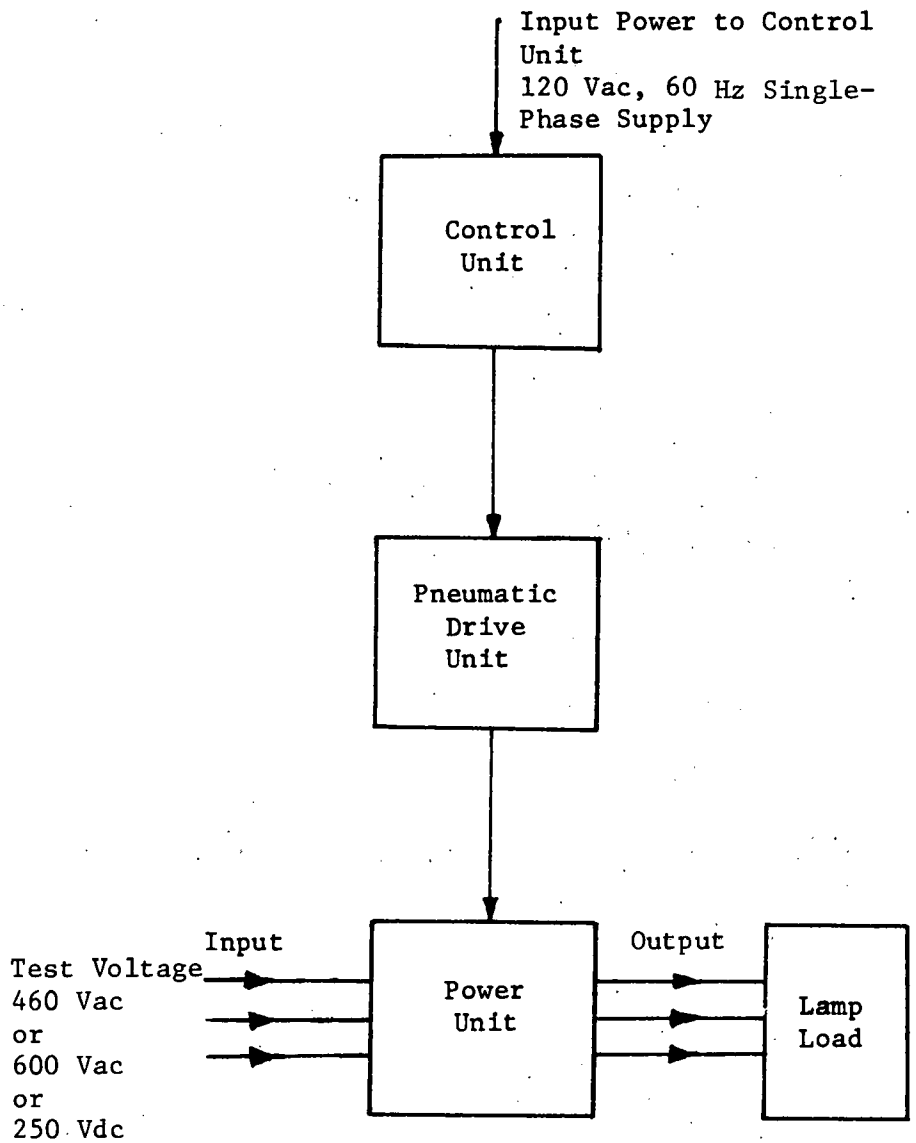
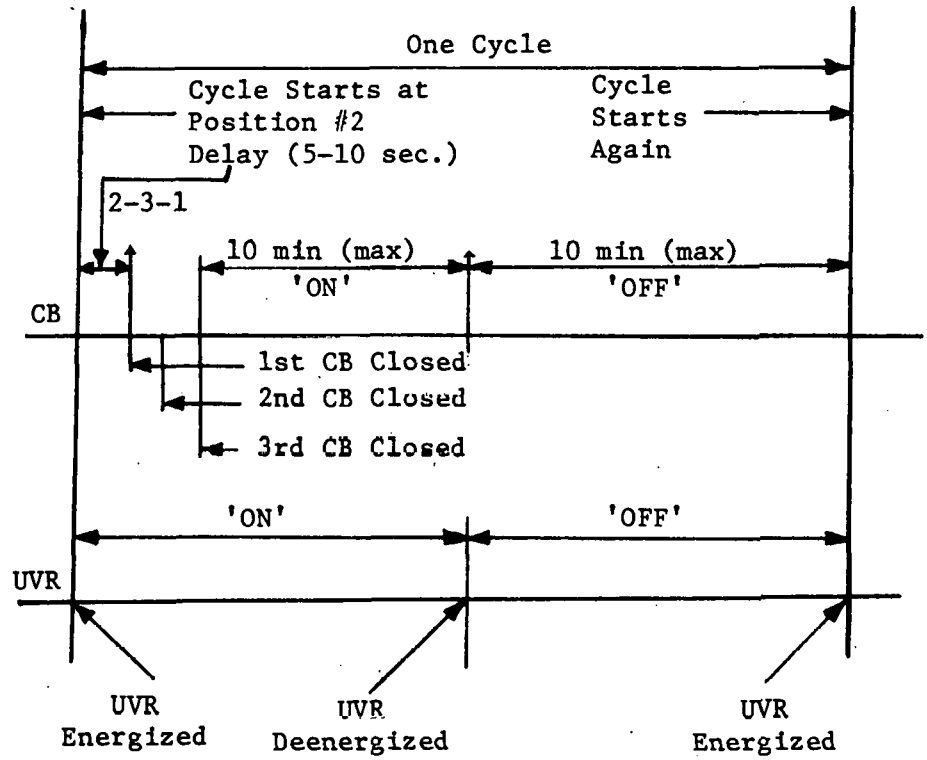
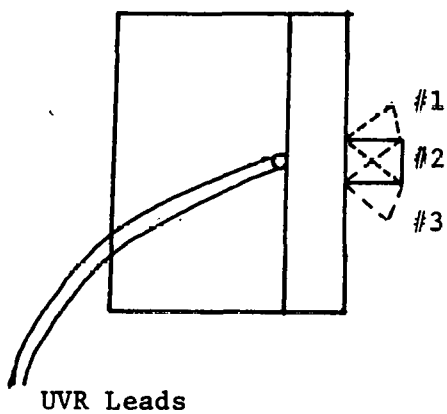


Figure 4: Block Diagram of Pneumatic Device



CB = Circuit Breaker, UVR = Undervoltage Release

Figure 5a. Sequence of Operation of Molded-Case Circuit Breaker Along with UVR.



#1: ON (Closed Position)
#2: OFF (Open Position)
#3: RESET (Reset Position)

Figure 5b. Side View of Molded-Case Circuit Breaker Showing Different Handle Positions.

The pneumatic drive unit as shown in Figure 2 comprises of solenoid valves, pressure regulators and air cylinders.

Operation:

The entire system operates in the following manner: Figure 5a shows the sequence of operations during the testing of the breakers, while Figure 5b shows the different handle positions of the breakers during the test. A manual depression of control switch in the input-power control unit, provides power to the remaining sub-units of the control unit. The test voltage is selected by operating two manual switches. At this point the test voltage is impressed at the input terminals of the test breakers, and a pilot lamp lights up indicating that the action has been completed. At the same time the UVR control energizes the UVRs in all the three test breakers.

After the UVRs have been energized the breaker energizer sends signal to the pneumatic drive to start energizing the breakers one at a time. When the last of the three breakers has been energized, the cycle time control unit starts the 'on' time delay period. At the end of this 'on' period, this same unit sends a signal to the UVR control to de-energize the UVRs, thus de-energizing the breakers. The 'off' delay time starts at this point. When the breakers have been de-energized successfully, the counters register counts indicating the completion of a cycle.

When the 'off' time period is completed the entire cycle starts again. In the case of malfunction of a breaker the entire system is shut down and can only be started again manually.

TEST PROCEDURES

Two types of test procedures will be used in conducting the laboratory test on molded-case circuit breakers. These test procedures[8] are sudden death and magnified loading.

Sudden Death Test:

In the sudden death procedure a set of three breakers will be tested simultaneously until at least one of them fails. At this point all these first set breakers will be removed and a second set of three breakers will be mounted on the device. This procedure will be followed for the remaining sets until all the breakers in a group are tested. The factor of acceleration is the rate at which the breakers are opened and closed. This rate will be higher than the normal rate at which the breakers are normally operated in mines.

The type of data to be collected are:

1. Ordered time-to-failure of the breakers—t_1, t_2, ...t_n

   where n is the number of breakers failed.

2. Number of cycles of operation to failure.

Using the total number of breakers that failed, the cumulative failure percentage will be calculated for each failed breaker. A plot of the cumulative failure percentage against time-to-failure on a Weibull probability paper would provide the following information:

1. The actual distribution of time-to-failure of the breakers.

2. The estimated hazard rate of the breakers.

3. The estimated mean-time-to-failure of the breakers.

The estimated hazard rate obtained from the above procedure is based on the accelerated test. Thus the hazard rate based on normal operation of the breakers have to be found. Assume that strength denisty of the breakers while on test is Raleigh distributed[9] as

$$f(y) = kye^{-(ky^2)/2} \tag{1}$$

where y is the stress on the breakers caused by the test, and k is a constant. It can be shown[9] that if x_1, and x_2 are the rates of operation of the breakers during the test in the laboratory and during the normal operation in the mines, respectively, then,

$$\lambda_2 = (x_2/x_1)^2 \lambda_1 \tag{2}$$

where λ_1 and λ_2 are the test and normal operation hazard rates of the breakers, respectively. The hazard rate λ_1, as already mentioned, incorporates the effect of acceleration in the test.

Magnified Loading Test:

In the magnified loading test procedure, the breakers are tested one at a time. Each breaker is operated at a different rate and the lowest rate selected is usually higher than the normal rate at which the breakers are operated in the mines.

The types of data to be collected in this test procedure are:

1. Time-tofailure of the breakers, T_1, T_2, T_3 ... T_m, where m is the number of breakers failed.

2. Rate of operation of the breakers, N_1, N_2, N_3 ... N_m, cycles/minute.

3. Number of cycles of operation to failure of each breaker, M_1, M_2, M_3 ... M_m.

The rate of operation if plotted against the time-to-failure of the breakers on logarithmic scales should produce a straight line. From this plot, it would be possible to obtain the time-to-failure and hazard rate of the breakers operated in the mine, using their rate of operation in the mine.

RESULTS AND CONCLUSIONS

Twenty molded-case circuit breakers are undergoing the two types of life tests described above in the WVU power laboratory. Their specifications are: 3-pole, 600-Vac, 400-A frame, 130-A continuous rating, 24-Vdc UVR, Special thermal setting of 225-A, and Magnetic trip setting of 400-800-A.

Each of these breakers is being tested at 480-V and at 150 A. This procedure conforms to the UL specification that requires molded-case circuit breakers be tested at rated voltage and current. In this case, the test current is greater than the rated current, thus providing some kind of accelerating stress in the test. Also, the rate of operation will be around 4 cycles per minute as required by UL specifications. This rate of operation provides the main accelerating stress in the tests.

Effort had been made to simulate coal mine environmental conditions on the circuit breakers for about a month in the United States Bureau of Mines facilities at Bruceton, Pennsylvania. Humidity was varied between 25% and 85%, while the temperature was varied between 40°F and 70°F.

Some initial results have been obtained in the past two months. However, they are not adequate enough to process statistically and report them in this paper. The final results will be reported at a future meeting dealing with coal mine power systems.

The pneumatic device is useful for the laboratory testing of molded-case circuit breakers for their reliability evaluation. The results of this evaluation could be used in the optimum design and operation of the circuit breakers from safety view point.

ACKNOWLEDGEMENTS

The authors appreciate the funding of this research by the U.S. Bureau of Mines, in the form of Grant No. G0144137. Mr. George Conroy of U. S. Bureau of Mines Research Laboratory, Bruceton, Pennsylvania is the Technical Project Officer.

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800 MHz RADIO COMMUNICATIONS FOR UNDERGROUND MINING

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ABSTRACT

For many years, we have speculated as to the effectiveness of UHF radio communications in underground mining. Theoretical studies have indicated that the optimum frequency range is around 1000 MHz. Several years ago, the Federal Communications Commission authorized land-mobile use of UHF television Channels 70-83. This allows use of 806-866 MHz for conventional systems.

With commercial land-mobile equipment available in the "800 MHz band", comparison testing has been done with 450 MHz in coal and limestone mines. This paper details tests done in late 1977 and early 1978 and compares actual results to theoretical studies. The conclusion recommends the most practical radio band for several typical mine types based upon actual testing.

For years we have speculated as to the usefulness of radio frequencies in the 800-900 MHz spectrum for land-mobile communications. 800 MHz mobile testing started in 1952 when W. R. Young and Bell Labs made tests in New York City on several bands up to 3700 MHz using war surplus radar equipment. Motorola began testing in this frequency range in 1955 in Chicago to see if 800 MHz was practical.

These tests used modified 450 MHz tube equipment with a transmitter power of eight watts and a receiver sensitivity of 0.6 microvolts (20dBQ). Comparisons were made using 150 MHz, 450 MHz, and 800 MHz. Quoting from a paper presented at the Southwestern I.R.E. conference in 1957 by Curt Schultz,[1] "tests were conducted in every conceivable environment, ranging from a test vehicle parked 40 feet underground in a parking garage (communications was possible with the base station), to inside railroad stations".

Conclusions of these tests were:

1. 800 MHz is very immune to man-made noises.

2. 800 MHz has the ability to penetrate into tunnels.

3. 800 MHz is extremely readable, even in standing wave conditions because the signal drop-outs below receiver threshold are so brief that they detract very little from the voice modulation being received as compared to the other bands in use.

This 800 MHz - 900 MHz band has long been viewed as one holding high potential for underground systems. Immunity to noise, ability to penetrate tunnels, and outstanding readability in a multi-path environment provide the elements necessary for a good underground radio system. In other testing done for the United States Bureau of Mines in 1975, Arthur D. Little concluded "that frequencies about 1000 MHz are optimum for long straight paths".[2] (See Figure 1 attached.)

It was not until several years ago that the Federal Communications Commission authorized land-mobile use of UHF television Channels 70-83 in Docket 18262. This allows use of 806-866 MHz for conventional systems. With this allocation, standard off-the-shelf equipment is now becoming available.

With this background of F.C.C. allocated frequencies, of encouraging theoretical studies, of successful tunnel testing, and of standard equipment availability, we organized several tests in operating mines. Tests were made in high coal and in large open mines primarily due to the ease of transporting equipment and accomplishing testing in the mine. The Blizzard of 1978 made the scheduled test in a low coal mine impossible. Testing will be done in low coal soon.

Our tests were made with the help of Valley Camp Coal near Charleston West Virginia (6'H x 16'W), CBS Limestone near Princeton, Kentucky (35'H x 50'W), and Blackriver Limestone near Corrington, Kentucky (35'H x 40'W). Motorola and USBM test results at Magma Copper near Tuscon, Arizona, that resembled high coal, was also used in preparing the results of this report.

The philosophy guiding our testing was to investigate different system components in operational tests. We wanted to see the effect of mine geometry on propagation of 800 MHz. We wanted to compare it to 450 MHz that has a good track record of success in underground systems currently operating.[3] We wanted to test different antenna

types and polarizations so the optimum combination could be recommended. We also desired to develop numbers for straight line attenuation (db/100 ft.) and compare our field results to theoretical studies.

To accomplish these objectives, we used an 800 MHz repeater coupling only its exciter into an antenna. Power output was 16 watts at the base station for these tests. To facilitate testing, a mobile was attached to a cart with antenna, control accessories, and batteries. This "portable" was also delivering 15 watts of power. We hope to continue testing using standard portable units when they become available. Power is not important to these tests; however, because it was as far as we could de-tune the equipment and still meet specs, the 15 watt level was used. An attenuation at the receiver simulated a portable transmitter power of 1.5 watts, which is the basis for all of our testing.

An outstanding property of all production 800 MHz equipment is the receiver sensitivity and fidelity. Our tests used several different units of the same base station model (Motorola C75RCB) with a 20dB Quieting of 0.33 microvolts. This low sensitivity adds to the total useable system gain resulting in increased range. As the 1955 tests pointed out and our tests confirmed, the readability of 800 MHz at low signal levels is absolutely excellent compared to any land-mobile radio band in use today. This is due to the small wavelength that produces extremely short multi-path fading. The net result is a much cleaner signal as perceived by the listener. This is especially apparent at low signal levels under one microvolt.

Signal strengths were measured at the base station by an engineer, using test point "W" in the receiver IF stages. This test point produces a DC voltage with an RF probe that is linear with signal strength. With this method, it was easy to determine the average signal levels from various locations throughout the mine.

Much practical antenna testing was done using Yagi antennas, DB Products hardhat and teardrop antennas, and quarter-wave whip antennas. The 800 MHz band again offers an advantage underground because the small wavelength provides for smaller physical antenna sizes. This means mounting antennas in tunnels and mounting antennas on vehicles is much easier. Portable antennas will also be small as the quarter wavelength of 800 MHz is about three (3) inches. A discussion of the results of our testing follows.

Figure 2 summarizes polarization information taken in high coal (6'H x 16'W). These numbers show the significant effect that tunnel geometry has on the propagation mode. Horizontal polarization is so

dominant that a vertically launched wave will soon be observed to be horizontal. Any system design in high or low coal must consider thi to obtain maximum range. Difficulty in designing a mobile or portable antenna for horizontal polarization may result in a horizontal repeater antenna talking to a vertical mobile antenna which is an 8dB compromise.

Figure 3 shows what happens when the tunnel height to width ratio approaches one and/or the tunnel dimensions are many times that of the wavelength being used. 800 MHz shows a 6dB average preference for the vertical while no preference was seen on 450 MHz. Both bands had the polarization rotate in an unpredictable manner around corners suggesting that a circularly launched wave from the repeater antenna might fill in better.

Waveguide coupling devices (antennas) are compared in Figure 4. The Yagi is superior, but the cost, installation difficulty and vulnerability to damage may outweigh the 6dB gain. Future testing will be done in coal with these antennas and a simple dipole.

The straight line attenuation test results from 800 MHz is an exciting number. In high coal, the attenuation rate is one-half that for 450 MHz. In 35'H x 50'W limestone, 800 MHz and 450 MHz attenuation rates are about equal (Figure 5). These results were confirmed in a second limestone mine and in a deep copper mine that resembles high coal. Propagation of Radio Waves in Coal Mines by Arthur D. Little, Inc.[2] predicted total straight line attenuations in high and low coal at 450 MHz and 1000 MHz. Their predicted values correlate well with experimental values considering that the frequency is slightly lower than 1000 MHz and that the coal mine tunnel undulated with the coal seam.

These repeatable results and their agreement to predicted values encourage further study of the ADL report. ADL goes to predict straight line attenuation with a loaded trip. Attenuation predictions for both bands are included with varying train to wall clearance. Using the numbers in Figure 5, a realistic situation can be constructed using 800 MHz that predicts worse case repeater spacing in high coal.

Let's assume the following system parameters:

A) Direct radiation (no leaky feeder)
B) 800 MHz
C) 2 watts of power (system gain is 140dB)
D) 1000 ft. loaded trip between repeaters
E) 3 ft. side clearance
F) 72" coal
G) No curves
H) 20dB train to tunnel coupling loss

It can be predicted from these assumptions that a repeater spacing of one mile is possible.

SUMMARY

POLARIZATION

Polarization is a function of tunnel shape. In a 6 ft. high by 16 ft. wide tunnel, the signal will propagate horizontally regardless of the injection polarization. Using horizontal polarization in a tunnel with this dimension, ratio significantly increases range. In very large tunnels or tunnels with a width to height ratio of one, no polarization is dominant. Circular polarization may help decrease dead spots in these mines.

ANTENNAS

Although antenna gains do affect range, more important considerations are space for mounting, polarization desired, and durability. For base station antennas, a single Yagi is preferred. If space is limited or bi-directionality is required, a single dipole can be used. The prime considerations on mobile antennas are polarization and ruggedness. For 800 MHz the DB teardrop is the most rugged antenna we have found and has some elements of horizontal polarization.

ATTENUATION

Straight line attenuation in large tunnels is about the same at 800 MHz as at 450 MHz. In high coal, 800 MHz propagates twice as far as 450 MHz and in low coal, the ratio goes to three to one. Calculations involving tunnel blockage from long trains indicate 800 MHz can still provide continuous coverage with one mile repeater spacing in high coal.

SYSTEM DESIGNS

From the information summarized in Figure 6, the most appropriate band and antenna system can be selected to match the mine type and communication need.

800 MHz holds the promise of simplifying underground radio systems from the installation, expansion and maintenance viewpoints. Whether it be low or high coal, section, haulage, or hoist communications, 800 MHz is a valuable new tool in the effort to upgrade deep mine communications.

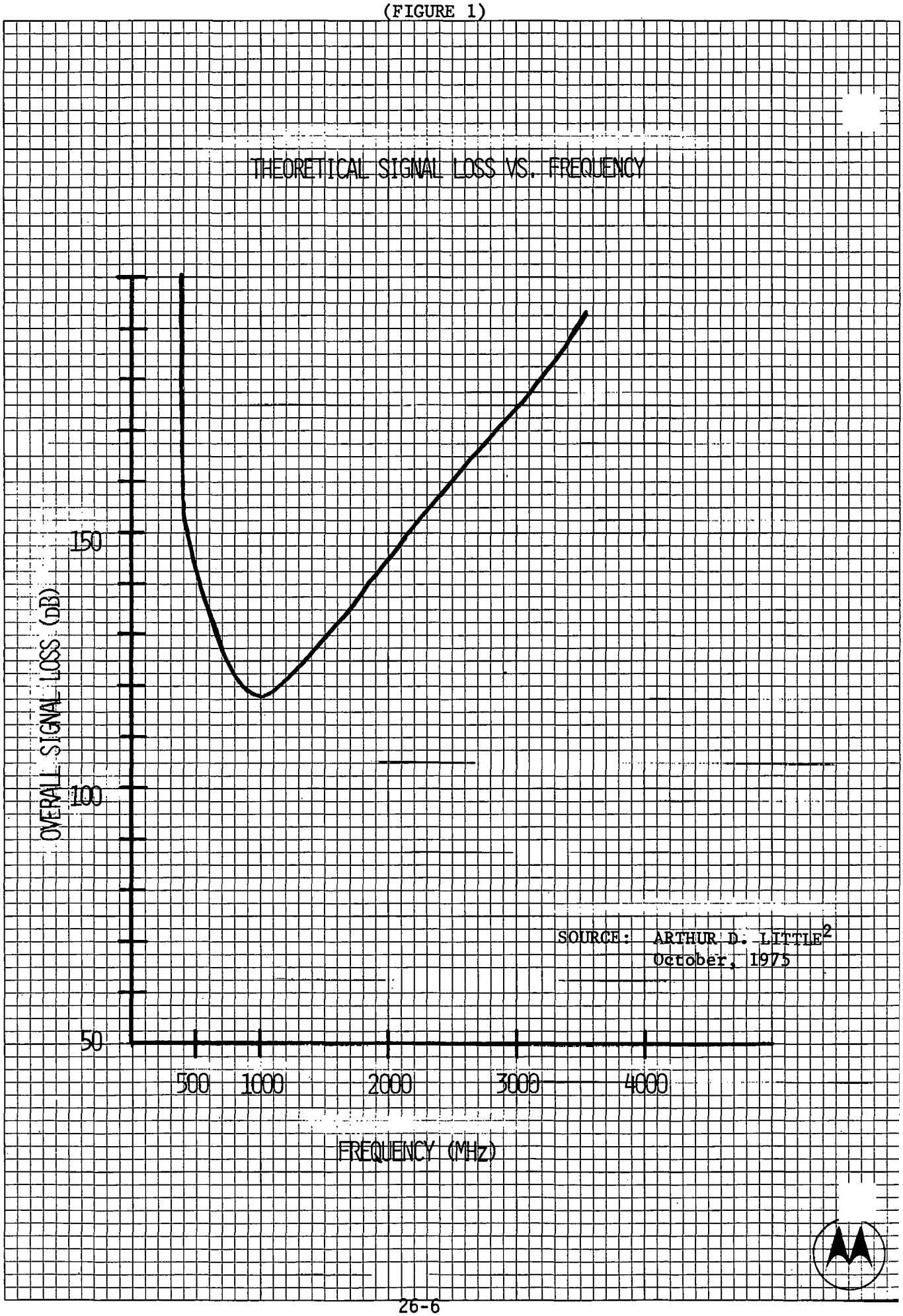
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(FIGURE 1)


THEORETICAL SIGNAL LOSS VS. FREQUENCY
OVERALL SIGNAL LOSS (dB)
150
100
50
500
1000
2000
3000
4000
FREQUENCY (MHz)
SOURCE: ARTHUR D. LITTLE[2]
October, 1975

BASE ANTENNA	MOBILE ANTENNA	RELATIVE DB
V	V	0DB
H	V	+21DB
V	H	+21DB
H	H	+29DB

H - HORIZONTAL POLARIZATION

V - VERTICAL POLARIZATION

- 6 FT. HIGH BY 16 FT. WIDE COAL
- FIVE POINT TEST AVERAGE
- 450 MHz SHOWS SIMILAR POLARIZATION CHARACTERISTICS

MOTOROLA INC.
12955 SNOW ROAD
PARMA, OHIO 44130

TITLE
POLARIZATION TESTS - 800 MHZ COAL

SCALE	DR. BY	DRAWING NO.
DATE 7/78	CK'D. BY	2

BASE ANTENNA	MOBILE ANTENNA	RELATIVE DB
V	V	+6
H	V	+3
V	H	+4
H	H	0

H - HORIZONTAL POLARIZATION

V - VERTICAL POLARIZATION

- 35 FT. HIGH BY 50 FT. WIDE LIMESTONE
- THREE TEST POINT AVERAGE
- NO DOMINANT POLARIZATION WAS NOTED ON 450 MHz

MOTOROLA INC.
12955 SNOW ROAD
PARMA, OHIO 44130

TITLE		
POLARIZATION TESTS - 800 MHZ LIMESTONE		
SCALE	DR. BY	DRAWING NO.
DATE 7/78	CK'D. BY	3

GROUND PLANE	0DB
DB PRODUCTS TEARDROP	+3DB
SINGLE YAGI	+6DB
DUAL YAGIS	+8DB

-TESTING DONE IN 35' HIGH BY 50' WIDE LIMESTONE

MOTOROLA INC.
12955 SNOW ROAD
PARMA, OHIO 44130

TITLE		
800 MHZ ANTENNAS		
SCALE	DR. BY	DRAWING NO.
DATE 7/78	CK'D. BY	4

	FREQUENCY		
	450 MHz	800 MHz	
HIGH COAL	6.2	3.2	MEASURED
LIMESTONE	3	3	ATTENUATION
HIGH COAL	5	2.1	PREDICTED[2]
LOW COAL	26	8.1	ATTENUATION
HIGH COAL	36	10	PREDICTED[2] ATTENUATION WITH TRAIN PRESENT (3 FT. SIDE CLEARANCE)
LOW COAL	BEYOND EXTRAPOLATION	16	

- VALUES ARE IN DB PER 100 FT.
- HIGH COAL - 72" H BY 16' W
  LOW COAL - 42" H BY 16' W
  LIMESTONE - 35' H BY 50' W

MOTOROLA INC.
12955 SNOW ROAD
PARMA, OHIO 44130

TITLE		
STRAIGHT LINE PROPAGATION		
SCALE	DR. BY	DRAWING NO.
DATE 7/78	CK'D. BY	5

	450 MHz		800 MHz	
	LEAKY FEEDER	DIRECT PROPAGATION	DIRECT PROPAGATION	
LOW COAL			✔ H	SECTION COMMUNICATIONS
HIGH COAL		✔ H	✔ H	
LARGE TUNNELS		✔ O	✔ O	
LOW COAL	✔ O			HAULAGE COMMUNICATIONS
HIGH COAL	✔ O		✔ H	
LARGE TUNNELS		✔ O	✔ O	
	✔	✔	✔	HOIST COMMUNICATIONS

H - HORIZONTAL POLARIZATION

O - VERTICAL, HORIZONTAL OR CIRCULAR POLARIZATION

MOTOROLA INC.
12955 SNOW ROAD
PARMA, OHIO 44130

TITLE		
SYSTEM DESIGNS		
SCALE	DR. BY	DRAWING NO.
DATE 7/78	CK'D. BY	6

MEDIUM-FREQUENCY PROPAGATION IN COAL MINES

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ABSTRACT

Medium-frequency (MF) waves can propagate in a coal seam, bounded above and below by conducting rock, in an approximate transverse electromagnetic (TEM) transmission-line mode with the electric field vertical and the magnetic field horizontal. The theory of this coal seam mode is applied to recent MF data for a number of mines in order to classify the propagation characteristic of the mines in a simple way.

When conductors such as rails, trolleylines, or power cables are present in the vicinity of the measurement path, it is found that the magnetic field versus range plots have two distinct regions with very different slopes. This effect is attributed to coupling of the cylindrically spreading coal seam mode to a low-attenuation conductor guided mode. As a result, voice communications range between mobile transmitters can extend several thousand meters. The use of MF for mine communications is discussed.

INTRODUCTION

MF wireless radio satisfies communication needs similar to those met by ultra-high-frequency (UHF)/leaky feeder systems.(1)* However, MF systems make use of existing mine wiring for range extension and do not require the installation of leaky feeder cable in the extensive complex of mine tunnels.

It is known that UHF radio suffers from the severe disadvantages of high losses when communicating around corners and through barricades.(2) UHF signals are confined within the boundaries of mine tunnels. MF signals, on the other hand, travel relatively well through the coal and rock.

* Numbers in parenthesis refer to references at the end of this paper.

MF wireless radio is intended to satisfy the communication needs of roving miners as well as the haulage communication requirements of trackless/trolleyless vehicles. The miners involved are typically section foremen, maintenance workers, ventilation crews, and machine operators. An effective MF radio system would answer the following communication needs:

1. Miners working in basic room-and-pillar sections and in sections with special needs, such as longwalls and sections having full-dimension haulage systems.

2. Miners operating vehicles in mines with trackless and trolleyless haulage systems in which traditional trolley wire carrier systems are not useful.

3. Members of rescue teams during mine rescue operations, thereby removing the need for inconvenient trailing wires.

The underground mining industry's extensive experience with trolley-wire carrier-phone systems has shown that a mine's electrical wiring, tracks, trolley wires, phone lines, and air lines can be used to carry low-frequency signals throughout underground mines, even to locations some distrance from the trolley wire-rail haulage system. Early in 1975 the Bureau of Mines bought and tested underground some prototype 335-kHz portable transceivers that were manufactured in South Africa. The wireless communication ranges obtained with these units were quite good and, in fact, much greater than originally anticipated from theoretical predictions. However, the tests left some doubt as to whether 335 kHz was the most favorable operating frequency for U.S. mines.

To better define MF communications technology, the Bureau of Mines initiated a measurements and hardware development program. Its output would be a MF unit to meet the portable wireless communication needs that remained unsatisfied by UHF radio. To be successful it is necessary to determine which operating frequencies are best suited to wireless applications in U.S. room-and-pillar coal operations and to design associated portable equipment rugged and small enough for use by the mining industry.

PROPAGATION MEASUREMENTS AND EXPERIENCE IN COAL MINES

The initial set of comprehensive measurements were made in a conductor-free area of Consolidation Coal's Ireland mine in the Pittsburgh seam. A 60- to 2,000-kHz frequency range was covered. An analysis of the data showed the following:

1. Longer communication ranges than anticipated were achieved (up to 500 m) in conductor-free areas.

2. Maximum ranges occurred at higher frequencies than expected (between 300 and 900 kHz).

3. Propagation was not influenced by the network of tunnels in the coal seam.

4. The best performance occurred when the planes of the loop antennas were oriented vertically and parallel to each other.

Subsequent measurements at other mines have shown that propagation in conductor-free areas is strongly dependent on coal seam properties. Figure 1 shows the results of measurements in six coal seams and clearly demonstrates the wide variation in propagation loss from seam to seam.

Original work indicating the existence of a low attenuation mode in a coal seam was published in 1976 by J. R. Wait.(3) Further analysis has been recently done by A. G. Emslie and R. L. Lagace(4) in which the coal seam is treated as a waveguide propagating a TEM wave. The model assumes the mine can be represented as three layers consisting of coal bounded by rock overburden and underburden and has shown good agreement between theory and experiment in the frequency range of 100 to 1,000 kHz.

OPERATING FREQUENCIES IN CONDUCTOR-FREE AREAS

The data analysis and theoretical modeling of coal seam waveguide radio wave propagation presented above provides a firmer foundation than previously available on which to base the selection of operating frequencies for wireless mine communication in the MF band. Large variations in signal attenuation rate have been found between the three coal seams investigated which are widely separated geographically. The Pittsburgh seam measured in northern West Virginia was found to be the most favorable, while the Herrin No. 6 seam measured in southern Illinois was found to exhibit extremely high loss.

Figures 2 and 3 show theoretical calculations for received field strength at 200 and 400 m, respectively. The parameter values are the average values deduced from the mine data shown in Figure 1 for the Pittsburgh, Pocahontas, and Herrin seams.

Also plotted on these figures for comparison are magnetic field noise levels versus frequency. These noise levels represent the upper bound envelope of average rms values found in active areas of coal mines and a lower bound noise level representing the intrinsic receiver noise for a narrow band frequency modulated (FM) radio receiver having an intermediate frequency (IF) bandwidth (B) = 12 kHz, noise figure (F) = 6 db, and loop antenna effective turns area (NA) = 1 m^2. The active mine noise levels were taken from data measured at spot frequencies in three coal mines by Bensema, Kanda and Adams(5)(6)(7) of the National Bureau of Standards.

Examination of Figures 2 and 3 reveals that performance is best for the combination of lowest coal conductivity and highest rock conductivity. Increasing coal conductivity increases the shunt loss in the coal seam waveguide, while decreasing the rock conductivity increases the series loss, both of which increase the signal attenuation rate. Furthermore, as this attenuation rate increases, the frequency behavior changes from a broad bandpass filter type of characteristic centered around 1 MHz for low attenuation rates like those found in the Pittsburgh seam, to an increasingly attenuated low pass filter characteristic having a steadily decreasing cutoff frequency as the attenuation rate increases to values like those found in the Herrin No. 6 seam. This trend toward the attenuated low pass filter behavior is, of course, more accentuated for the longer communication ranges, i.e., 40 m versus 200 m.

The most favorable frequencies for communicating these ranges can be estimated by comparing the signal level curves with the noise level curves plotted on each figure. The most favorable operating bands are seen to be those where the slopes of the signal and noise curves are equal or nearly equal. Since the signal curves also represent levels for the largest, intrinsically safe transmit moment, an absolute measure of performance can also be estimated by noting the number of db that the signal level curve lies above the noise level curve at the frequency of interest. For example, moderate performance (occasional message repetitions required) requires that the average rms carrier-to-noise ratio at a FM receiver be at least 10 db or better for the mine noise condition prevailing during the message transmission.

The equivalent receiver noise curve is probably the most representative for communication in conductor-free areas located at least one or two entries (i.e., separated by one or two pillar widths) away from mine electrical conductors carrying high noise currents. On the other hand, the active mine upper bound noise curves most likely give an overly pessimistic view. In this latter instance, these levels occur in the immediate vicinity of high-power machinery and electrical conductors, locations where the signal will also be enhanced by the presence of the conductors. Thus an active area noise curve approximately half way between the two noise extremes is probably more indicative of worst case "conductor-free area" noise levels. These levels are likely to occur in transition regions between conductor-free and conductor-present locations.

Figure 2 reveals that conductor-free area portable-to-portable radio ranges on the order of 400 m will be attainable only in highly favorable, low-attenuation seams like the Pittsburgh seam. Furthermore, the performance will occur only at operating frequencies between about 900 kHz and 2 MHz and under noise conditions approaching the receiver-noise-limited case. In addition, Figure 3 illustrates that conductor-free ranges of 200 m will not be attainable in high-loss seams such as the Herrin No. 6

seam, even at frequencies below 100 kHz and under receiver-noise-limited conditions. The outlook is somewhat improved, but not much, for moderate-to-high-loss seams like the Pocahontas No. 3 seam, where 200 m ranges should be possible over a broad band of operating frequencies between about 150 kHz and 1.5 MHz under receiver noise limited conditions.

On a more positive note, the in-mine measurements of Cory [8] and others have also shown that greatly improved radio communication ranges are obtainable in coal mines when the portable units are located in a tunnel having electrical conductors such as a power cable, pagerphone line, or trolley wire/rail line. Significant range improvements have even been experienced with one and sometimes both of the units located in a tunnel separated by a coal-pillar width from the tunnel with the conductors. In some cases improvements have also occurred with one of the units located in a tunnel separated by two coal pillars from the tunnel having the electrical conductors. Since these conductors are generally located in haulageways and working sections where miners requiring communications are also located, this propagation condition is an important one. Moreover, it is one that may allow the desired radio communication ranges to be achieved and exceeded even in coal seams having high signal propagation attenuation rates. Figure 4 shows data taken in three mines near conductors that illustrate the range-enhancing effect of conductors. Additional data are presently being taken in the vicinity of mine electrical conductors and analyzed to quantitatively model and assess the impact of such conductors on the attainable radio communication range.

HARDWARE DEVELOPMENT

The Bureau is currently developing a MF system for mobile and portable communication capable of covering a small mine using belt haulage and battery-powered vehicles. The overall system is indicated in Figure 5 in which a repeater is used to provide whole-mine coverage from portable to portable and from portable to base. Tentative specifications are shown in Table 1.

An operating system is expected to be completed by the fall of 1979.

SUMMARY

Prototype MF wireless radios have been tested in underground mines representing coal seams of several different conductivities. These tests were conducted as part of the Bureau's program to investigate the propagation characteristics of mines. The test results in the 60- to 2,000-kHz range indicate a conductor-free area propagation range of 150 to 500 m and a range of over 1,000 m in the presence of mine conductors. The range at a specific frequency in conductor-free areas is largely dependent on the conductive properties of the coal and, to a lesser extent, on seam thickness and conductive properties of the surrounding rock; in areas with conductors these properties are much less important.

Further underground evaluation of prototype MF wireless radios for both personnel and vehicular use as well as evaluation of various repeater schemes will be performed in conjunction with propagation measurements.

The results of these equipment tests and propagation loss measurements will be used to improve the design and performance and to determine system design guidelines for future mine wireless radio equipment.

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TABLE 1.--Portable transceiver specifications

Frequency range	300-1000- to 10-kHz steps
Channels	2 transmit, 1 receiver for repeater operation and talk around
Modulation	SSB
Antenna	Detachable loop, bandolier
Power out	1 watt PEP
Squelch	Self-adjusting relative to noise background
Weight	Less than 1 kg

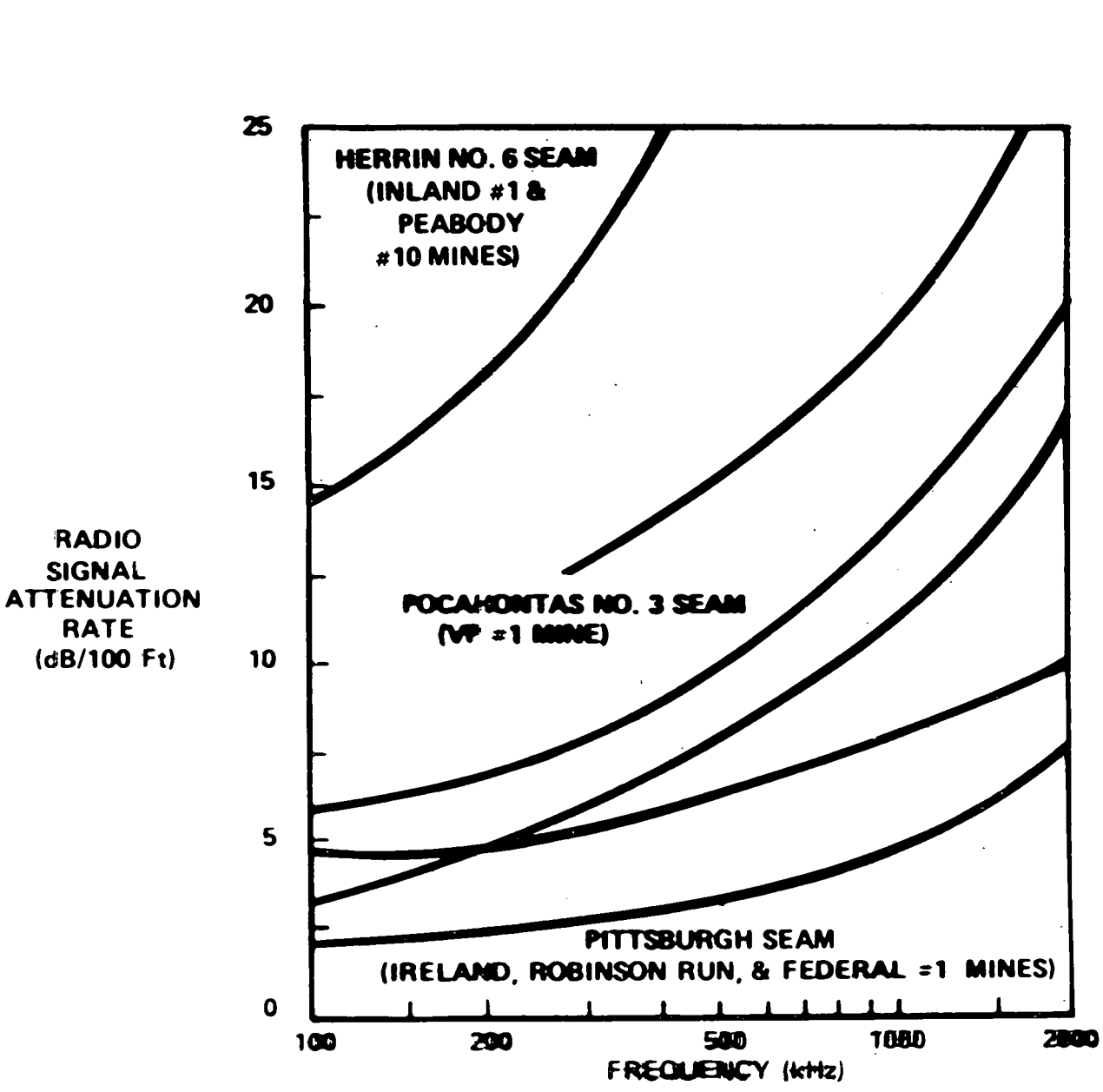


FIGURE 1.--Composite plot of signal attenuation rates in db/100 ft for six mines in three different coal seams

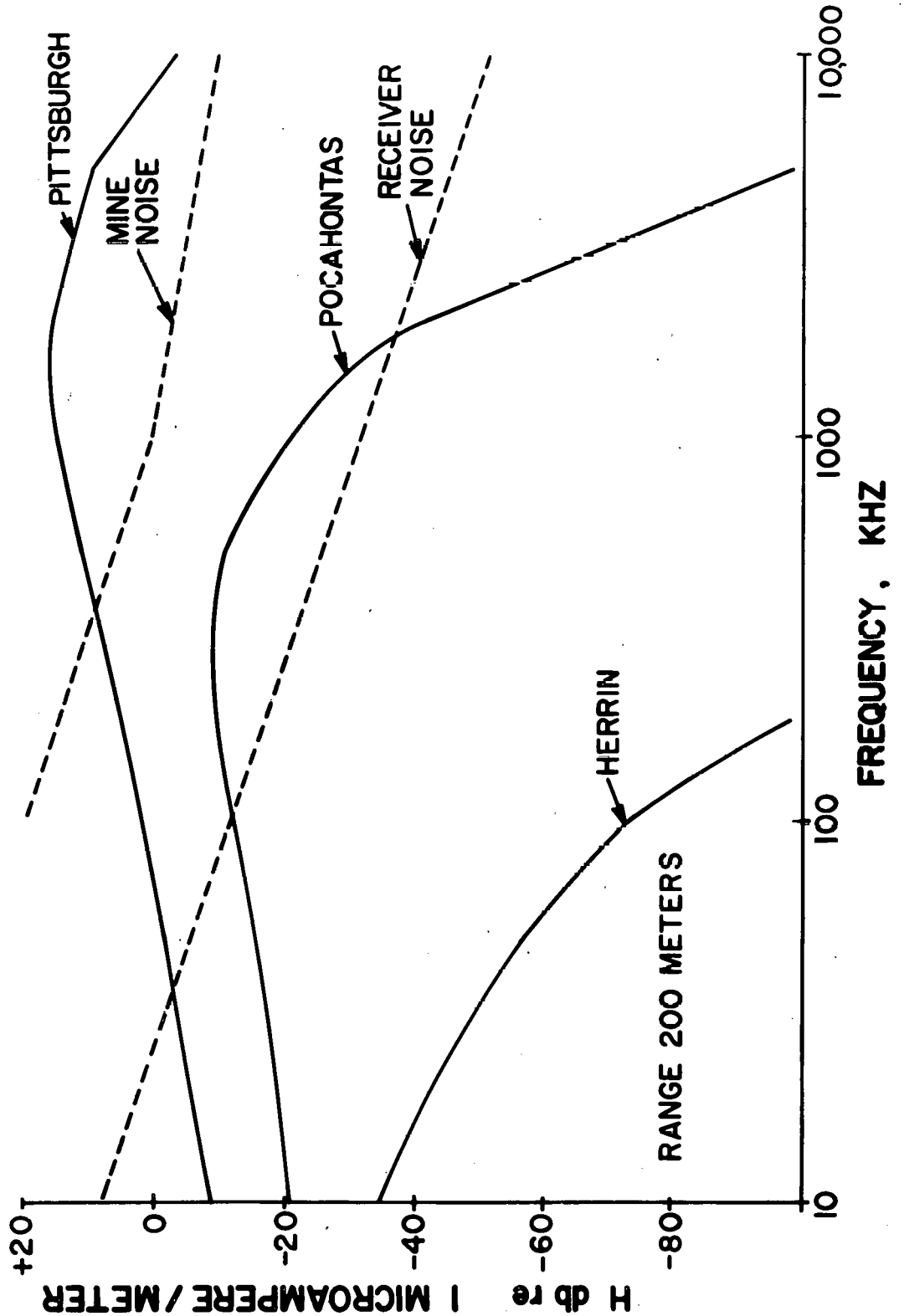


FIGURE 2.--Theoretical values of H as frequency for Pittsburgh, Pocahontas, and Herrin coal seams at a range of 200 m

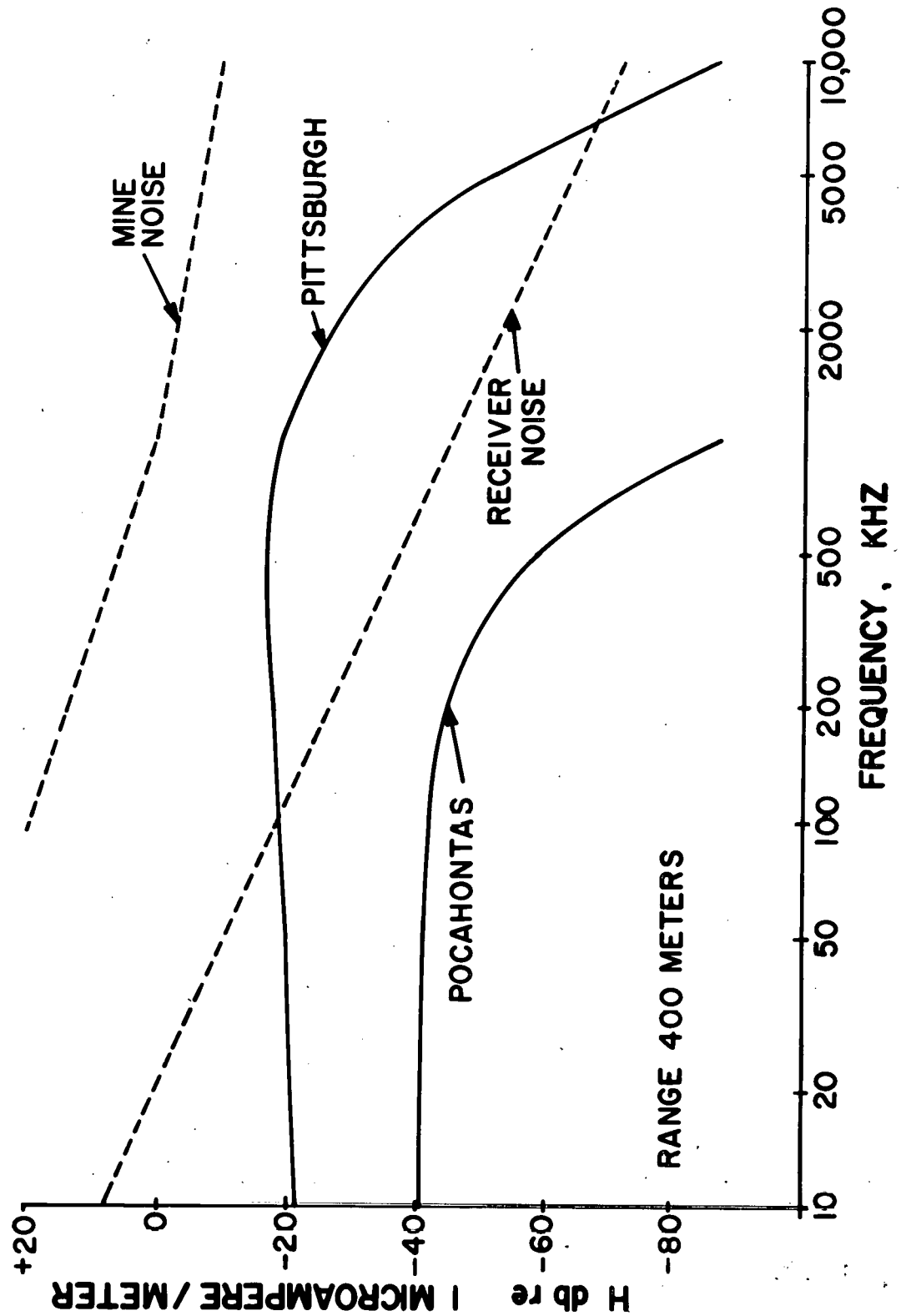


FIGURE 3.--Theoretical values of H vs frequency for Pittsburgh, Pocahontas, and Herrin coal seams at a range of 400 m.

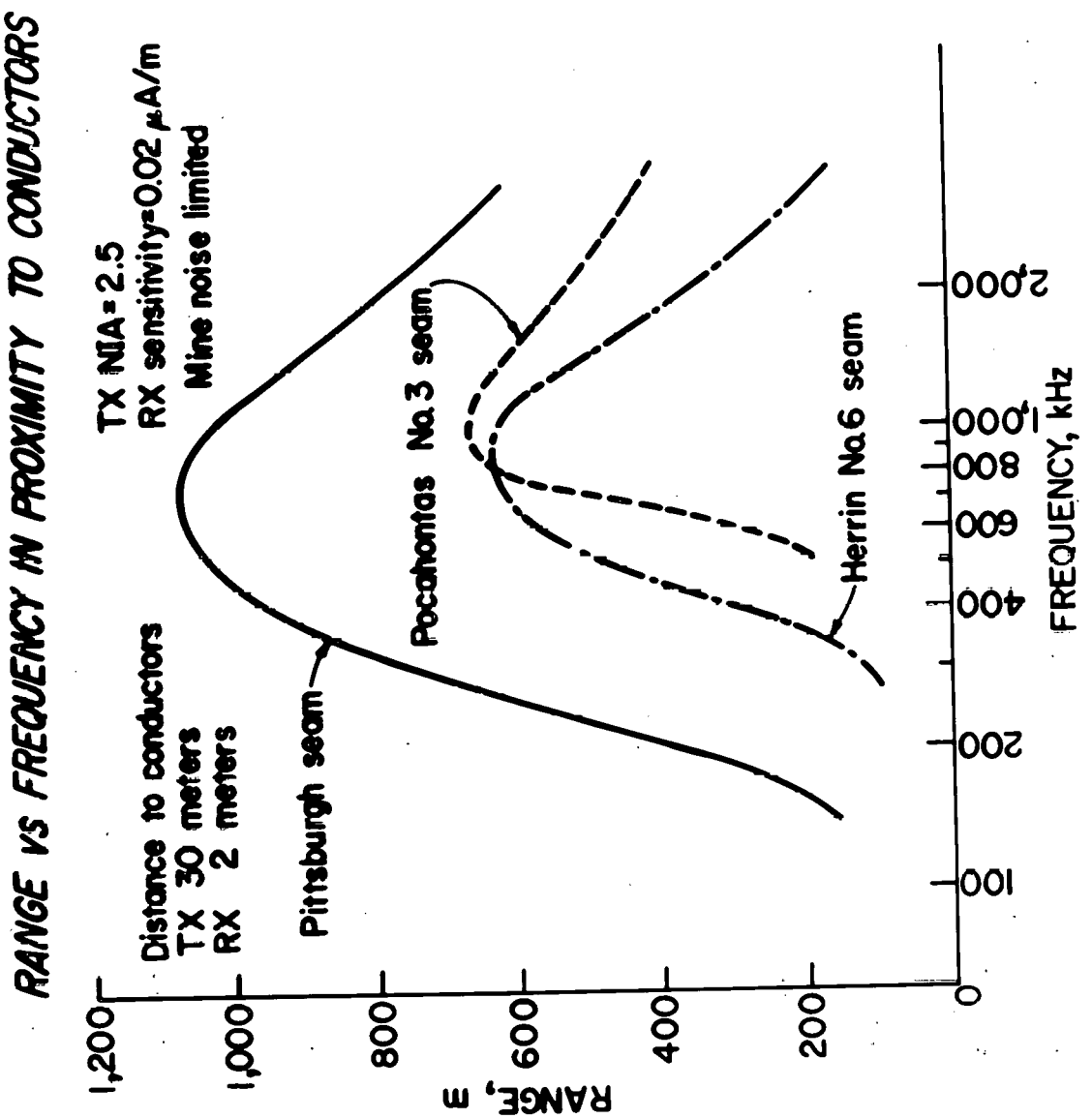


FIGURE 4.--Communication range in three seams in areas with conductors

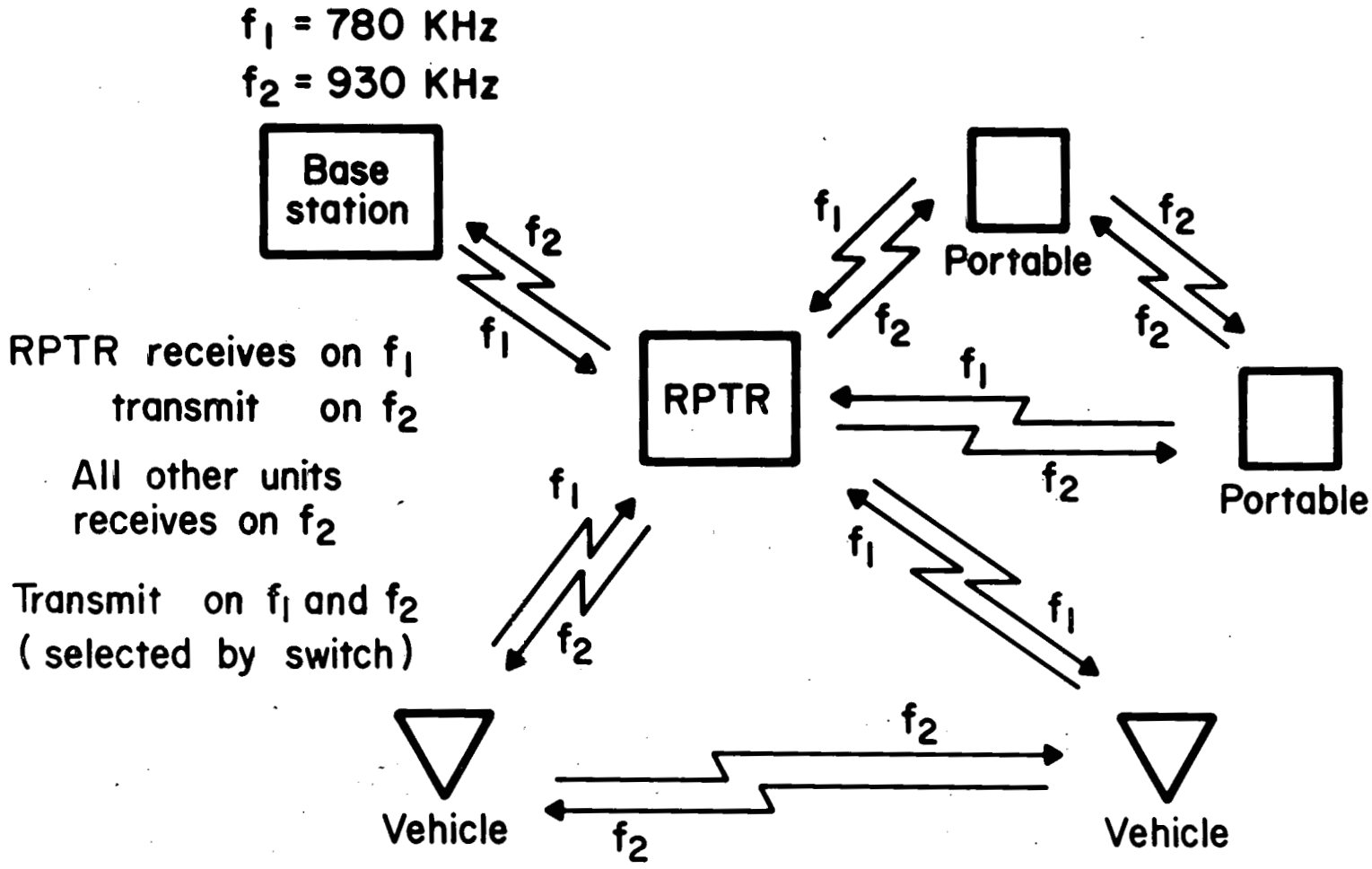


FIGURE 5. Medium-frequency communications using a repeater

TRAPPED MINER THROUGH THE EARTH

COMMUNICATIONS IN COAL MINES

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ABSTRACT

The U.S. Bureau of Mines has developed an electromagnetic location system for locating trapped miners in coal mines. This development program has led to the fabrication of a low power light-weight transceiver that can be attached to miners belts and a surface receiver and transmitter that can communicate through the earth with the underground transceivers. The equipment and through the earth communications concepts are being field tested at numerous U.S. coal mines. Performance of the trapped miner location equipment has been successfully demonstrated at mine depths up to 1400 feet.

I. INTRODUCTION

In recent years, much attention has been given to the development and use of low frequency through-the-earth radio systems for mine rescue operations. Murphy and Parkinson, in their review paper of 1978 (1), have discussed the desirability of using wireless systems for both communications and trapped miner location.

A wireless electromagnetic location and communications system has been developed for locating trapped miners in coal mines. This system consists of a light-weight transmitter and receiver that can be attached to miners belts.

The underground transmitter is a compact unit, designed to attach to the top of a miners cap lamp battery for power. Each transmitter operates on one preselected discrete frequency, tone pulsed at a slow rate. The frequencies used are from 510 Hz to 3030 Hz, spaced in 60 Hz increments between the 60 Hz power line harmonics. The transmitter is designed to be used with any available wire loop antenna. The nominal loop is standard #12 wire deployed around one or two coal pillars. This configuration results in a transmitting moment of typically 1000 amp t m^2. Deployment of this system is shown in Figure 1.

The surface receiver is a larger multichannel unit approximately the size of a self rescuer. It also attaches to a miners belt and has its

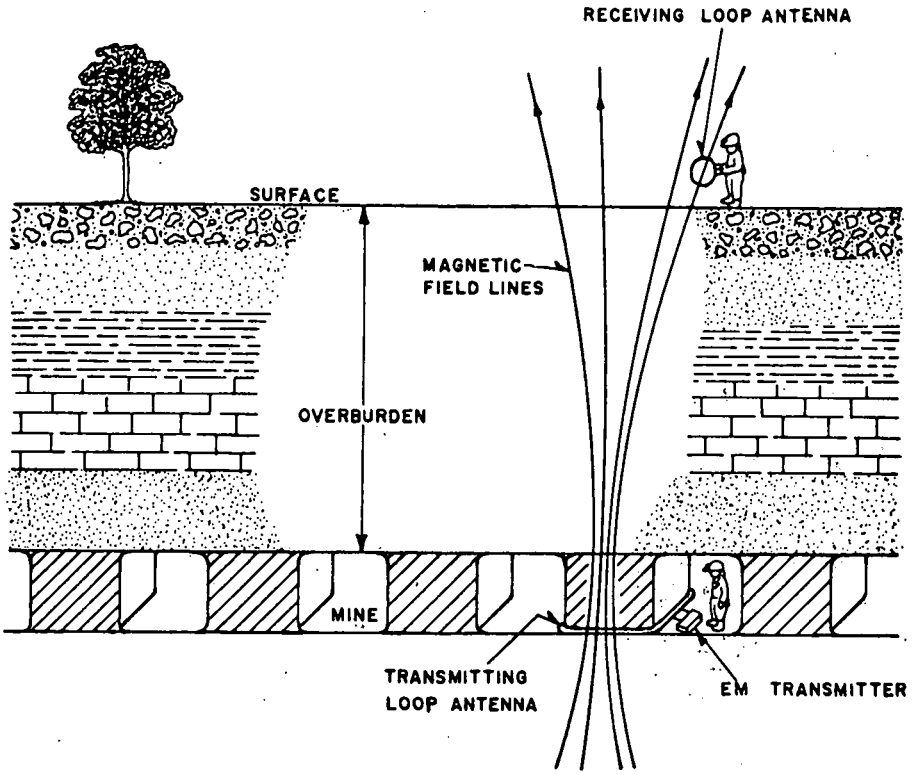


Figure 1 Deployment of Trapped Miner EM Location System

own battery pack inside. Each receiver has multichannel capability so that any transmitter can be received on one receiver. The receiver uses a 500 turn loop antenna fifteen (15) inches in diameter. With this portable antenna, the receiver can be moved to locate the point of minimum horizontal field strength, and thus enable the user to accurately locate the position of the transmitter.

Under the USBM, "Reliability and Effectiveness Analysis of Electromagnetic Location Systems," Contract #J0166060, the performance of four of these transmitters and several receivers is being evaluated for both uplink and downlink propagation. In addition, a high power transmitter, capable of producing a 20,000 Amp t m^2 transmitting moment, is being used to transmit downlink baseband voice. A compatible voice receiver is used in the mine to receive the downlink transmissions. The test program developed under this contract has several aims:

1. Evaluate the performance of the trapped miner location devices in a variety of coal mines.

2. Establish a data base of transmission loss as a function of depth.

3. Compare uplink transmission loss characteristics with downlink loss characteristics.

4. Characterize the mine propagation environment in terms of EM noise and effective overburden conductivity.

II. THROUGH-THE-EARTH COMMUNICATIONS CONCEPTS

The effects of a finitely conducting earth on electromagnetic waves excited by oscillating dipoles were first analyzed by Sommerfeld in 1909 (2). This and subsequent work (3) form the basis for modern electromagnetic through-the-earth communications theory.

Formal expressions for the electric and magnetic fields excited by horizontal and vertical magnetic dipole antennas have been given by Wait and Campbell (4) and Sinha and Bhattacharya (5) respectively. These theoretical expressions were used to compute attenuation factors for through-the-earth propagation as a function of propagation distance in skin depths (see Figure 2). (One skin depth is defined as the distance a plane wave has to travel in a conducting medium before it is attenuated to 37% of its initial amplitude).

Skin depth (δ) can be computed as follows:

$$\delta = \left(\frac{2}{\omega\mu\sigma}\right)^{\frac{1}{2}}$$

where ω = the radian frequency ($2\pi f$).
μ = the magnetic permeability of the medium in henries/m.
σ = the electrical conductivity of the medium in mhos/m.

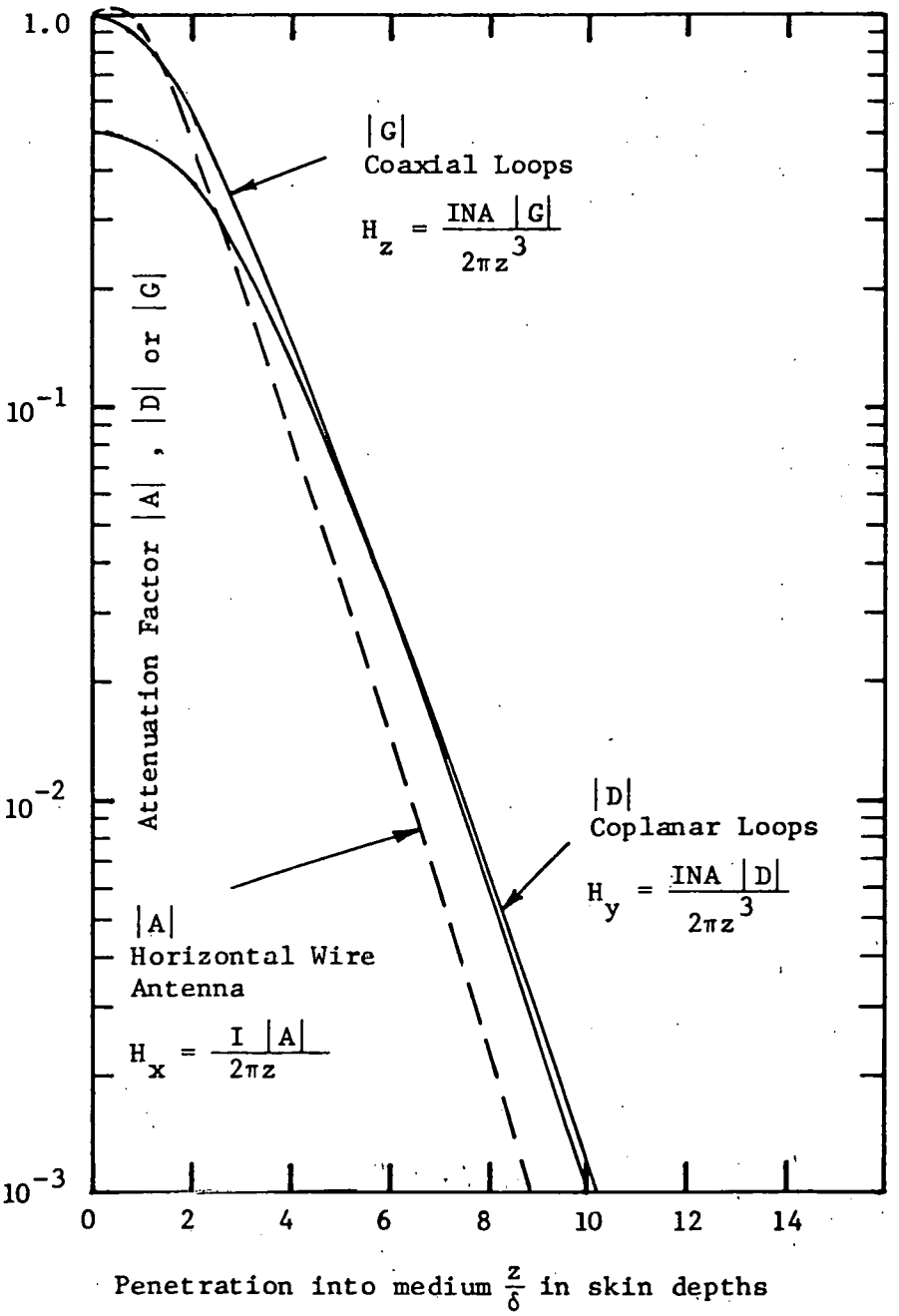


Figure 2 Electromagnetic Attenuation in Conducting Media

In nonmagnetic media, the above expression reduces to

$$\delta = \frac{503.3}{(\sigma f)^{\frac{1}{2}}} \quad \text{meters.}$$

Since skin depth is inversely related to the square root of conductivity times frequency, it follows that skin depth and communications distance can be maximized by minimizing frequency. In general, frequencies less than 10 kHz are required in uplink and downlink communications systems designed to penetrate to depths as great as those of the deepest U.S. coal mines, about 2,000 feet.

Note that in the expressions for calculating magnetic fields from horizontal or vertical magnetic dipoles, (HMD) or (VMD), in Figure 7 the field strength falls off as the inverse cube of separation distance between the transmitter and receiver. However, this is not the case for a transmitting antenna having dimensions large comparable to the propagation distance, where the field strength falls off as inverse distance. Expressions for fields produced by a long horizontal wire antenna (HWA) have been formulated by Wait (6) and these were used to determine the attenuation factor (F) shown in Figure 2 for the HWA. Normally, such an antenna is utilized on the surface as a downlink transmitting antenna. It is grounded at each end and driven directly with low frequency communications signals, setting up an effective return current in the earth which in turn produces secondary magnetic fields which can be picked up by lightweight portable manpack receivers in the mine. An alternate deployment would be to use a large area single turn loop where each leg of the loop can be treated effectively as a long horizontal wire antenna. This latter method has the advantage of requiring no electrode contact with the earth and presents a low and predictable antenna impedance to the transmitter terminals.

III. FIELD TEST PROGRAM

Low frequency magnetic field strength measurements for both uplink and downlink have been made at more than 70 coal mines on this program. The data obtained therefrom provides statistics on system performance for the coal mining industry.

A. Calibration Set-Up

Prior to taking the equipment in the field, the receivers were carefully calibrated in an environmental test chamber at the BuMines facility in Bruceton, Pennsylvania. It soon became evident that the receiver gains varied with temperature, especially from 30°F to 70°F at the lower dial readings. Temperature and dial reading charts were prepared by BuMines to aid in the data reduction phase of the field test program. Because of the difficulty in measuring temperature of the electronics and the excessive bookkeeping required, a more reliable method of field strength data had to be identified.

It was then proposed that we use a signal injection calibration technique which allows a direct determination of field strength based on the value of the injection voltage required to cause a receiver meter deflection to match that of the measured field strength.

The induced voltage in a multiturn air core loop antenna is given by

$$V_{induced} = 2\pi fNA\mu_o H$$

where

- f = the frequency in Hertz.
- N = the number of turns in the loop.
- A = the loop area in m^2.
- μ_o = the permeability of free space ($4\pi \times 10^{-7}$ h/m).
- H = the field strength in amps/m.

For the receiving loop used in these tests,

$$V_{induced} = 4.49 \times 10^{-4} fH.$$

By using an RC calibration set-up as shown in Figure 3, it was possible to externally induce a voltage (V_{cal}) across the one ohm resistor in the loop circuit such that (V_{cal}) could be directly related to the induced voltage ($V_{induced}$) from external magnetic fields. By choosing an appropriate value for the capacitor (i.e., C = .0715 μf), a one volt rms signal from the HP208A signal generator would induce a voltage in the loop circuit equivalent to that produced by a 1 ma/m magnetic field. This allows the field personnel to determine magnetic field strength in ma/m merely by recording the signal generator voltage required to produce the same deflection on the receiver meter as that produced by the measured field strength. This calibration scheme was used on the measurement program from the very outset.

B. Test Results

Evaluation of the test results obtained at the 70 mines tested thus far shows the manpack transmitting and receiving equipment to be extremely reliable even in spite of the relatively harsh coal mine environment. In the vast majority of the mines tested, measurements could be obtained at each of the four frequencies (630, 1050, 1950 and 3030 Hz). At some of the deeper mines, however, the two highest frequencies were difficult to detect because of poor signal/noise ratios.

EM noise measurements were obtained in conjunction with field strength measurements at most of the mines visited. The cumulative signal to noise ratio distribution for uplink measurements in the 70 mines tested thus far is shown in Figure 4 for the lowest frequency of 630 Hz and the highest frequency of 3030 Hz. In general, the deeper mines showed the best signal to noise ratio at 630 Hz while the shallower mines showed the best signal to noise ratio at 3030 Hz. Measured signal to noise ratios of less than 0 dB were obtained during times of intermittent interference, where the noise was measured during periods of interference and the signal was measured when the interference was not present.

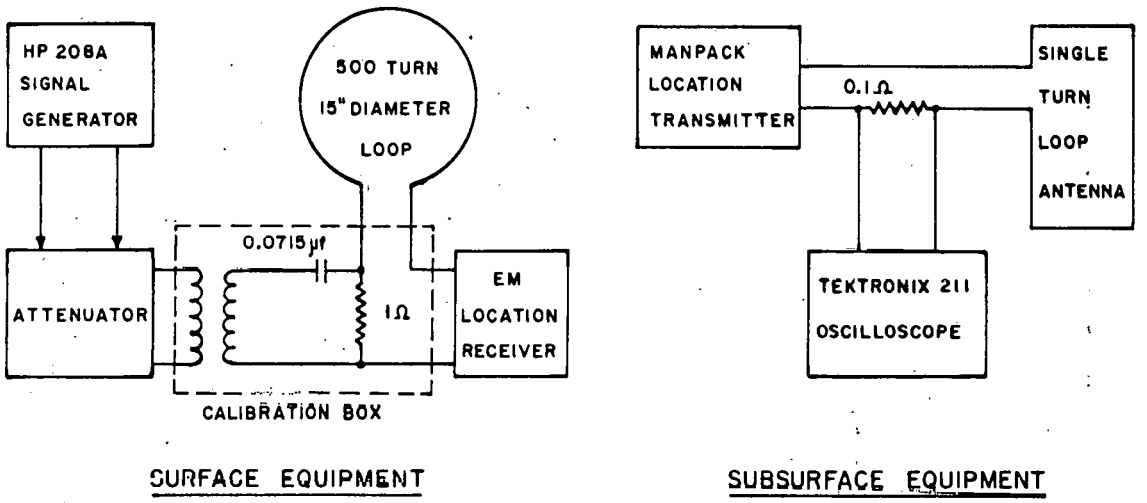


Figure 3 UPLINK CALIBRATION SET-UP

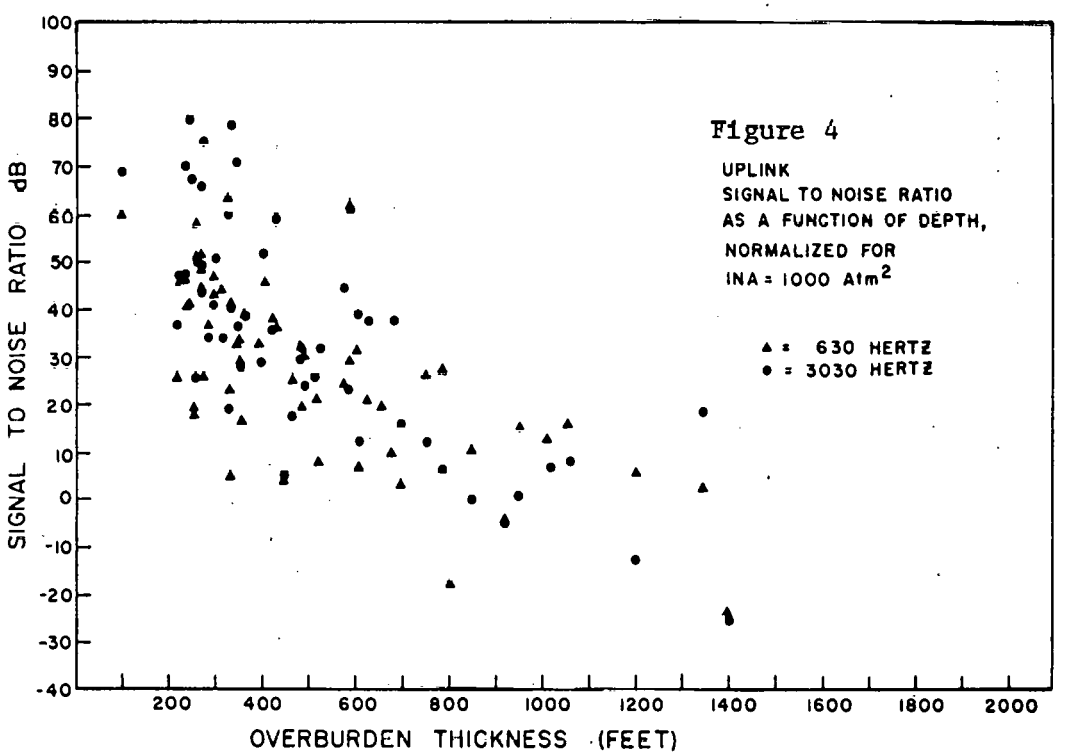


Figure 5 shows the calculated uplink magnetic field strength as a function of frequency for an underground transmitting moment of 1000 amp turns m^2 and an overburden conductivity of $\sigma = 10^{-2}$ mhos/m. Superimposed on this curve is the atmospheric noise spectral density for a 1 Hz bandwidth based on actual noise measurements made in Colorado during the summer season. This graph shows that the optimum signal to noise ratios as represented by the vertical arrows tend to lower and lower frequencies as the mine depth increases. This is consistent with the observed data in Figure 4 for the uplink measurements in mines.

Figure 6 shows the downlink signal to noise measurements obtained in the same mines. The same general trends are observed as in the uplink case, however, the noise in this case is generally that produced by underground power systems and electrical machines. In working mines, the underground electromagnetic noise environment in the frequency range from 630 Hz to 3030 Hz is, in general, more severe than the surface noise environment. Consequently, a larger transmitting moment was used on the surface for the downlink measurements to overcome the ambient noise in the mine. The data in Figure 6 is normalized to an effective transmitting moment of 1000 amp turns m^2 for comparison with the uplink data.

Figures 7 and 8 show the composite transmission loss measured at the 70 mines for uplink and downlink propagation respectively. Transmission loss (L) is that loss attributed to the conducting medium and is given by

$$L = -20 \log |G| \text{ dB}$$

where $|G|$ is defined by the curve in Figure 2. The transmission loss term is based on the assumption that the transmitting antenna is a dipole. (i.e., the loop diameter is small compared to the transmission distance through the overburden). At the shallow mines, the dipole assumption was not always satisfied. Consequently many of the abnormally high losses observed at the shallow depths (less than 300 ft) are influenced by non-dipole transmitting configurations.

In some of the mines, the uplink and downlink loss was nearly identical while at other mines there was a large discrepancy between them. Observed discrepancies between uplink and downlink propagation losses were due to the different way that underground metallic objects couple into the transmitting antenna for uplink and receiving antenna for downlink. In mines where underground measurement locations were free of metallic objects, the uplink and downlink losses were nearly identical.

Downlink voice transmission was attempted at many of the mines visited on this program. The Bureau of Mines supplied a high power Public Address type audio amplifier which was used to drive audio voice currents directly into a large square loop lying on the surface of the earth. The baseband voice transmissions were readily received in mines up to 700 ft. deep; however, the underground noise environment was usually the controlling factor as to whether the transmissions were

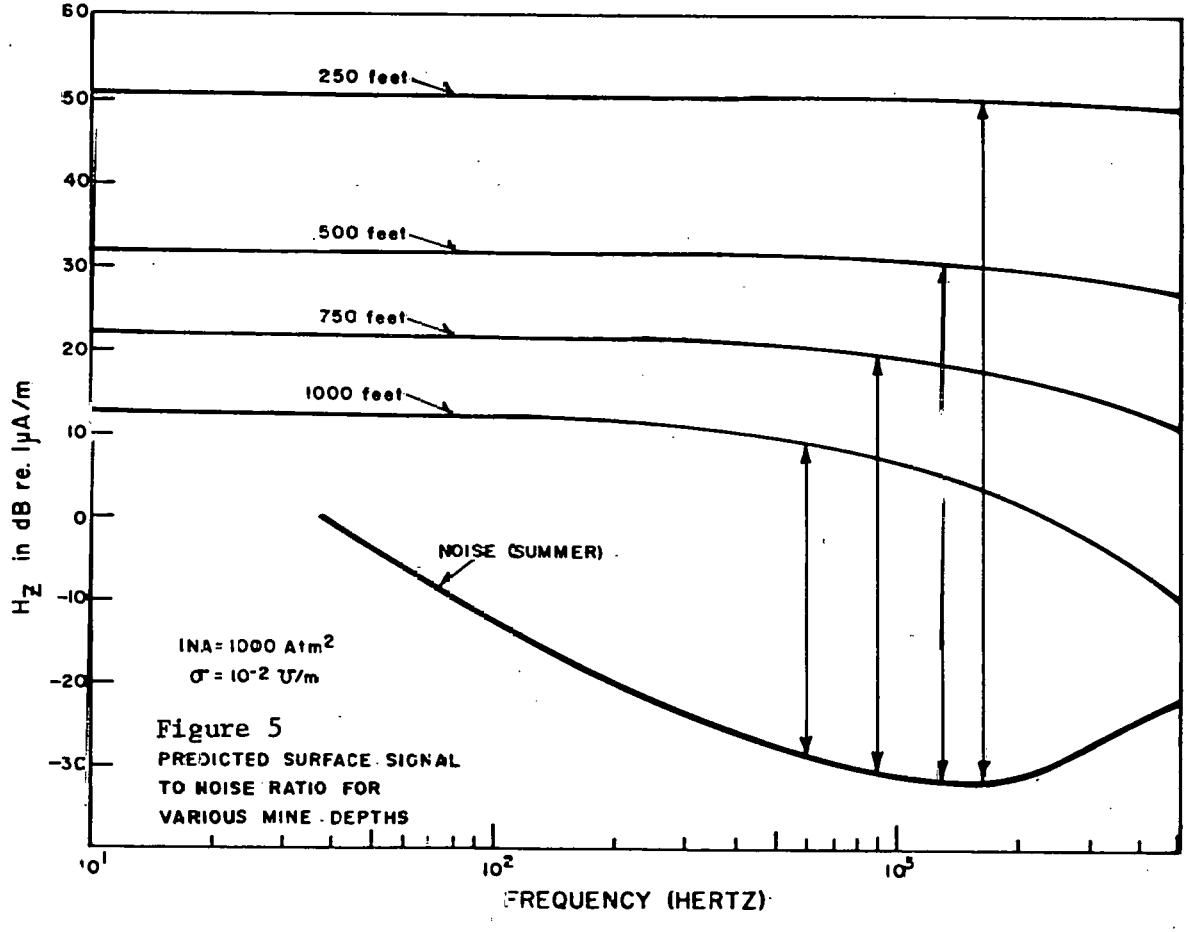
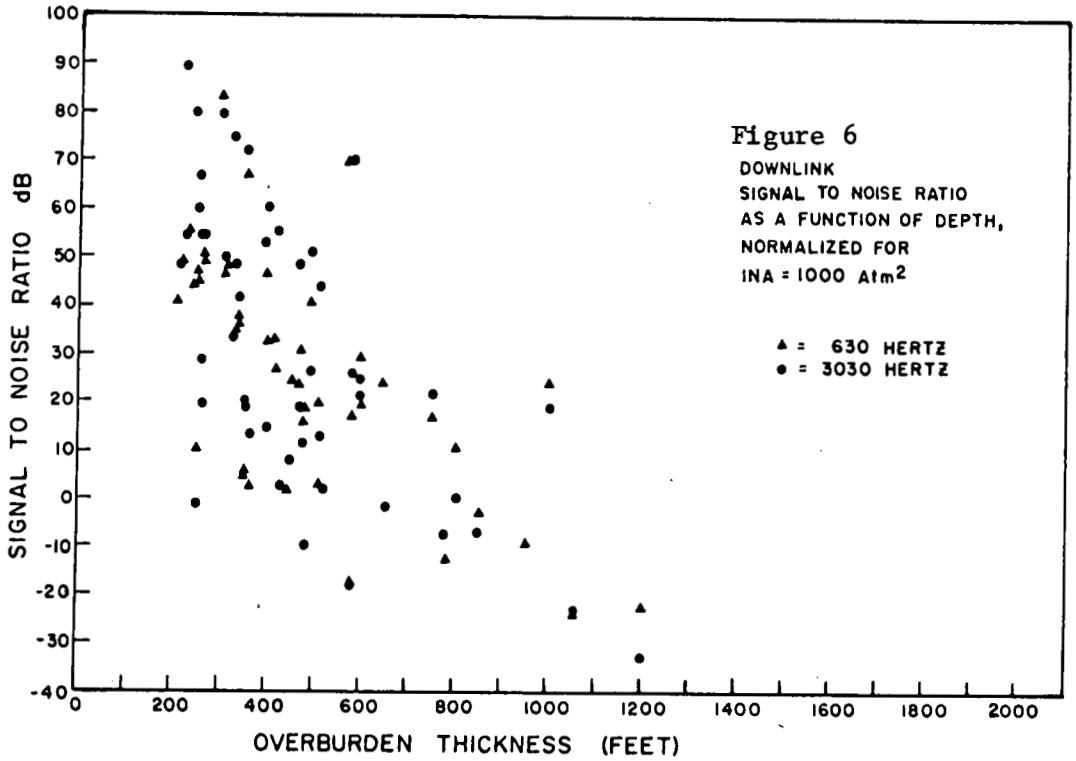


Figure 5
PREDICTED SURFACE SIGNAL TO NOISE RATIO FOR VARIOUS MINE DEPTHS



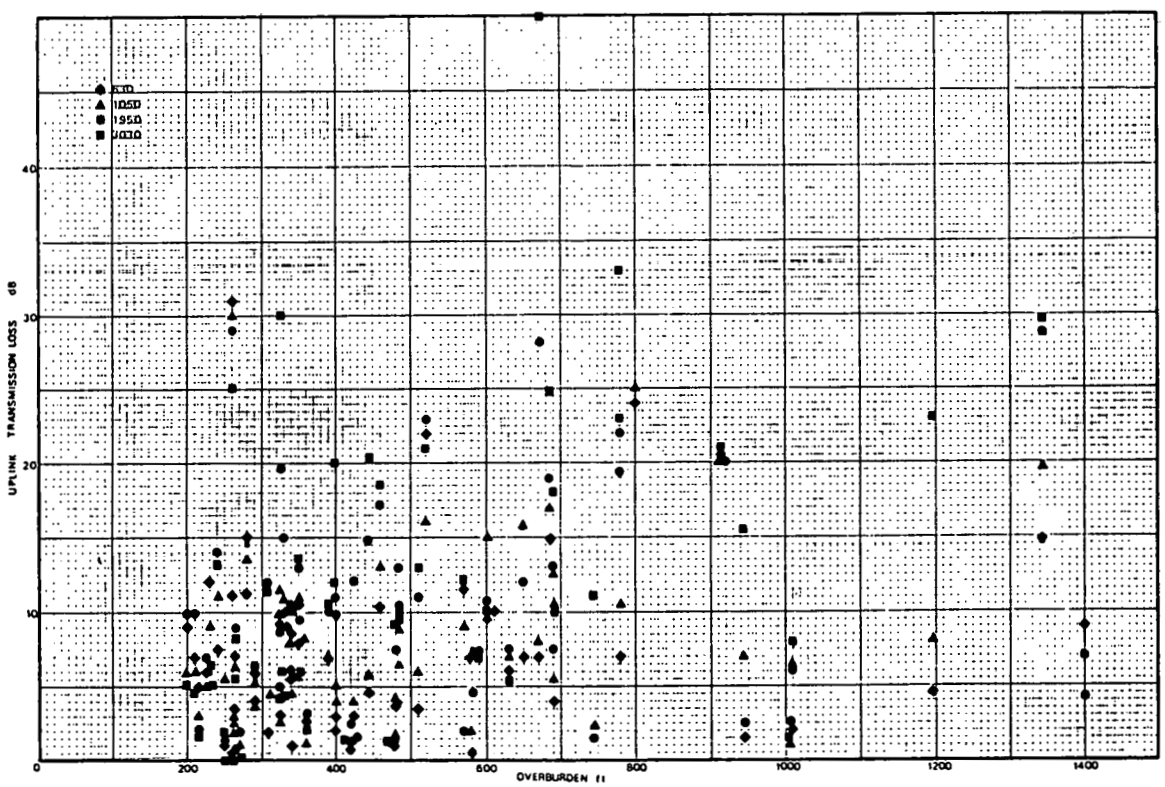


Figure 7 Uplink Transmission Loss Vs. Overburden Thickness

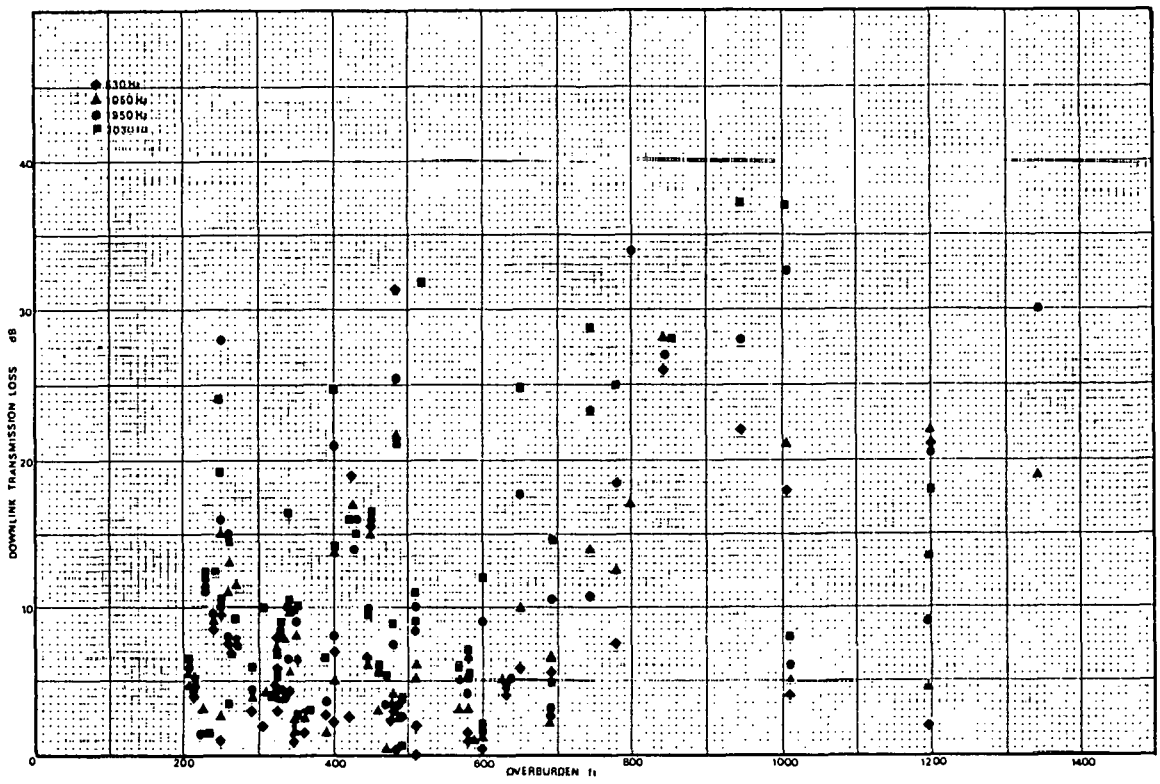


Figure 8 Downlink Transmission Loss Vs. Overburden Thickness

intelligible or not. In quiet areas away from power cables and operating machinery, the voice transmissions were highly intelligible and excellent voice recordings were obtained in the mine. In noisy areas, this was not the case. Furthermore, the antenna used to receive the voice transmissions was the same as that used to transmit the coded uplink signals and therefore could not be conveniently oriented for either minimum noise or maximum signal to noise ratio. For these conditions, the voice recordings were highly contaminated by mine noise.

IV. CONCLUSIONS

- The results obtained thus far on this program demonstrate that the BuMines Electromagnetic Location System has performed effectively in all of the 70 mines tested. Furthermore, the reliability of the hardware has proven to be excellent, in spite of the harsh coal mine environment.

- Uplink downlink transmission losses observed in the 70 mines tested thus far do not always agree. Discrepancies result from the different effect that underground metallic objects have on the transmitting antennas (uplink) and the receiving antennas (downlink).

- Electromagnetic noise and overburden conductivity impose the bottom line on the systems performance. This is true for both uplink and downlink, code and voice.

- Baseband voice transmissions are limited by underground EM noise especially when using an underground receiving antenna that cannot be conveniently oriented for minimum noise pick-up.

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IMPROVED RAIL HAULAGE COMMUNICATIONS

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ABSTRACT

Communications with moving tracked vehicles in a rail haulage mine pose a difficult problem. These communications take place from dispatcher to vehicles or from vehicle to vehicle via the trolley line, which is a very poor communications line. As a result, deadspots and high-signal areas can occur anywhere along the line, or signal can decrease simply as a function of distance.

An analysis is made of the trolley line as a communications line, and methods of improving its characteristics are discussed. Data from inmine experiments are presented, showing actual improvements in communications.

Recent Bureau developments in new hardware are also discussed. These include an adaptive speaker control for existing carrier phones and a new generation of improved carrier phones.

INTRODUCTION

Although trolley carrier phone systems leave much to be desired for haulageway communications, the fact remains that they represent the only practical means of dispatcher-vehicular communications. Most problems associated with these systems are transmission line related; i.e., a trolley line was never intended to be a good communications line, and it certainly is not. However, techniques do exist for improving overall communications. These techniques can be easily implemented, and the results are often excellent.

In addition to the transmission line problem, basic carrier current hardware is archaic in design. Simple improvements can be made to this hardware that make vastly improved communications possible. The most significant improvement is that of incorporating magnetic field pickup instead of voltage pickup on the trolley wire.

COMMUNICATIONS ON THE TROLLEY LINE

Trolley carrier phones presently serve as the most important communication system in rail-haulage mines. These phones are frequency-modulated (FM) transceivers based on tuned radio frequency (TRF) circuitry and usually operate at 88 or 100 kHz, although many other commercially

available frequencies are possible. The proper operation of the carrier phone system is essential for both operational and safety reasons. Up to now, there have been no real alternatives to this system for rail vehicular-dispatcher communications. The problems, and sometimes frustrations, of these systems were tolerated simply because there was no alternative.

The usefulness of a trolley carrier phone system stems from its ability to use the trolley line as the transmission path. Since all rail vehicles are physically connected to the trolley line for power, this transmission path is convenient.

However, anyone familiar with mine communications is aware of the fact that trolley carrier phone communications are highly variable in quality. There are times when good communications over great distances are possible. So at best it can be said that trolley carrier phone communications are tolerable, but vast improvements are needed.

The problem, of course, is more of a systems problem than a hardware problem, although improved hardware would certainly help. It is interesting to note that except for changing from vacuum tube circuitry to solid-state circuitry, there have been no basic improvements in the hardware since it was introduced.

The real systems problem is associated with the trolley line itself. Its prime function is to distribute dc power to vehicles and fixed loads. It was never intended to function as a transmission line.

Figure 1 illustrates the theoretical attenuation rate of an unencumbered trolley line functioning as a transmission line, assuming typical values of conductivity of the surrounding medium. The attenuation rate of a carrier phone system operating at 100 kHz is seen to be only 1 db/km. On this basis trolley carrier phones having a dynamic range of 70 db (25 V to 8 mV) should be able to communicate over such a line for 70 km (43 miles). In practice only a small percentage of this range is achieved.

A trolley line, since it is really first and foremost a power bus, is shunted by many mine loads. These loads are vehicle motors, vehicle lights, water pumps, track block lights, personnel heaters, rectifiers, etc. A large rectifier, for example, can have a radio frequency (RF) impedance as low as 2 ohms. When one considers that there can be hundreds of such loads on the trolley line, with some of them being mobile, it becomes apparent why the 70-km range is not achieved.

To learn more about the characteristics of unencumbered trolley lines, measurements were made on such a line in an operating coal mine. These characteristics are as follows:

Characteristic impedance, Z_o	200 ohms
Attenuation rate	1.6 db/km
Inductance/length	1 μH/m
Capacitance/length	30 pf/m
Velocity of propagation	.64 of free space

TRADITIONAL SOLUTIONS

Because of the problems in achieving good communications, several time-honored solutions have been tried to improve things. Perhaps the most widely used solution is the addition of so-called Z-boxes. The Z-box is simply a passive coupling device that permits RF signals to be transferred from one transmission line to another while blocking dc. Tuning is possible to achieve the best signal transfer. The technique is almost always that of transferring signals from a trolley line to a phone line in an attempt to compensate for low signals on the trolley line. In certain instances the approach works well; in other similar instances it can actually degrade communications. Sometimes it only transfers a poor communications area to somewhere else. The addition of Z-boxes to a trolley carrier phone system should be done with caution, because it rarely achieves the desired results. An explanation of this erratic behavior will be offered further in this paper.

When the most important communications link is from dispatcher to vehicles, rather than vehicle to vehicle, good results can often be achieved by connecting the dispatcher's transceiver to a phone line that runs in the same entry as the trolley line. This scheme is often superior to that where the dispatcher is connected to the trolley line also. But, as before, if there are Z-boxes connecting the two, inconsistent results can still occur.

Yet another solution is that of providing the dispatcher with a remote transceiver. In this scheme the transceiver is situated in a favorable location and only the audio and keying lines are run to the dispatcher. This method is especially effective when the dispatcher is located on the surface, or in a remote area of the mine.

One method of reducing the influence of RF shunting loads is to devise a method of isolating those loads. Although the power rectifiers represent the most severe loads on the line, they are often easily improved. Figures 2 and 3 show two possibilities.

If the rectifier is set back far enough from the haulage entry, tuning the leads is often possible. In cases where the leads are too short to exhibit sufficient inductance for tuning, large open air coils can be built and tuned. These and other schemes are discussed in detail in several Bureau of Mines reports.[1] *

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* Numbers in parenthesis refer to References at the end of the paper.

IMPROVED SOLUTION, THE DEDICATED WIRE

The dedicated-wire technique for improving rail haulage communications shows great promise for both rail haulage and whole-mine communications. This technique has been around for some time, although why it worked was not clearly understood.

In the last section, it was shown that coupling to phone lines with Z-boxes, or attaching the dispatcher's transceiver to the phone line, sometimes improves carrier phone communications. This is because a phone line is actually a type of low-grade dedicated wire. It is not as good as a true dedicated wire, because there are phones attached sometimes along with other RF loads. In addition, the phone line has numerous splices, uncontrolled branches, taps, untwisted lengths, etc., that increase attenuation. So the best way to get around this problem is to make sure the dedicated wire is truly dedicated, and that means an independent wire, perhaps 14 gauge, run down the entryway with the trolleyline on the wide side, but not connected to the trolleyline in any manner.

Such a wire, since it is unloaded, has a very low attenuation rate of about 1 db/km (the same as an unloaded trolleyline, since the geometry is the same). Therefore, if a signal is impressed on the dedicated wire, the signal remains high. Since the trolleyline and dedicated wire are located in the same entry, there is a mutual electromagnetic coupling between them. The effects of loads on the trolleyline are transferred to the dedicated wire, and the high signal on the dedicated wire is transferred to the trolleyline. Fortunately, if the separation between the two is large enough, the loading effects of the trolleyline are only weakly transferred to the dedicated wire, so that the attenuation rate stays low. But at the same time, the high signal levels on the dedicated wire are strongly coupled to the trolleyline. The net result is that communication is now possible in areas where it was not possible before.

Figure 4 illustrates the beneficial effects of the dedicated wire in an entryway with a trolleyline. Curve A shows what happens if transmission and reception take place on a dedicated wire only. This is possible between fixed stations but obviously not so if vehicles are involved. Curve B shows the voltage on the dedicated wire, from a transmission on the trolleyline. Curve C shows the reverse, the voltage on the trolleyline from a transmission on the dedicated wire. Curve D shows the signals received on a trolley wire if transmission takes place on the trolley wire.

The theoretical benefits of a dedicated wire to improve trolley carrier phone communications have been proven in practice. In U.S. Steel's Robena mine, a large section of haulageway toward the Willow Tree shaft was such that communication from vehicles to dispatcher was not possible. The solution, suggested by a carrier phone manufacturer, was to install a dedicated wire to the wide side, connected directly to the dispatcher. Z-boxes were also installed, although their benefit is unknown. As a result, good communications was established into an area where it was not possible before.

In Consolidation Coal' Montour #4 mine, the problem was even more serious. A remote dispatcher's transceiver was installed in the offending area, connected to the dispatcher by audio and keying lines. This remote transceiver helped some but not enough. Communications were still poor.

Even connecting the transceiver to the phone line and installing Z-boxes did little to improve the situation. Consol and the Bureau agreed to undertake a carefully controlled experiment to determine whether or not a dedicated wire would be of value.

In November and December 1977, a Bureau team and Consol personnel made measurements in the mine. These measurements were made from a jeep equipped with a Sierra* model 303A tuned voltmeter. The trolley line was encumbered with many loads. Initial measurements without a dedicated line showed that it made little difference if the Z-boxes were connected. However, the installation of a dedicated wire with the dispatcher's remote transceiver connected to it resulted in vast improvements (figure 5) especially in areas of initially low signals. The dedicated wire was properly terminated in 200-ohm resistors, and proper splitting networks were installed at branch points. As a result, communications along the entire entry (several miles) where the dedicated line was installed changed from one of near uselessness to one of good quality.

It should be emphasized that dedicated wire communications is not restricted to the situation where improved trolley wire communications is desired. Indeed, the greatest potential may be in mines that do not even have a trolley line.

One example is Consolidation Coal's Oak Park mine in Eastern Ohio (figure 6). In this mine standard carrier phones are equipped with loop antennas such that they transmit and receive on this antenna instead of by coupling to a trolley wire, which does not exist. This is a belt haulage mine; and the vehicles are used for personnel and supplies. This system permits whole-mine vehicle-to-dispatcher and limited vehicle-to-vehicle communications.

Other mines, such as Repbulic Steel's Kitt mine, are installing similar systems.

MEDIUM-FREQUENCY SYSTEMS

Bureau of Mines research has shown that medium-frequency (MF) (300 kHz to 3 MHz) systems have a very high potential for solving the ultimate problem: Whole-mine communications. Whole-mine communications as described here means the ability for two or more vehicles or personnel to communicate with each other or with a base station, no matter where they are in the mine.

Medium frequencies work well in the mine environment because they are low enough to not suffer severe attenuation while going through conducting mediums but high enough to couple parasitically onto metallic structures in the mine such as power cables or water pipes. Once coupled, linear attenuation on these structures is also low. Several highly controlled measurement programs[3] in the mine environment have been conducted to determine MF characteristics in detail.

*Use of brand names is for identification purposes only and does not imply endorsement by the Bureau of Mines.

Data so far have shown that in wire-free areas, the determining factor on range is the mine geology, the most favorable of which is low-conductivity coal and high-conductivity overburden and underburden. Under these conditions a parallel-plate waveguide exists, trapping the MF into a cylindrical spreading mode of propagation with low loss. Ranges of up to 350 m were measured under these conditions. In areas containing wires, ranges of many kilometers are not uncommon.

The Bureau is presently designing base stations, repeaters, and portable and vehicular radio to take advantage of the favorable MF propagation modes. The relationship between MF systems and dedicated wire techniques is obvious. A whole mine MF system with base stations and repeaters might be configured as shown in figure 7.

THE LOW-IMPEDANCE LINE

A bridging load R on a transmission line reduces the line voltage to the fraction $\frac{R}{R+Z_o/2}$. Hence, the loss of signal is not only a function of the value of the bridging load R, but also of the characteristic impedance Z_o of the line. It is also apparent that a lower value of Z_o would be beneficial.

Since Z_o for a trolleyline is around 200 to 300 ohms, a convenient way to lower Z_o must be found. For a transmission line, $Z_o = \sqrt{L/C}$. Therefore, one obvious approach would be to add artificial distributed capacitance to the line.

Figures 8 and 9 show an experiment that was recently done to test this low-impedance technique. A 4,460-ft (1.36 km) length of 300-ohm transmission line was deployed on the surface, a signal source was applied, and signal level was measured along the line, first unloaded, and then loaded with three modest loads. The three loads caused a signal loss of 55.5 db, but when 2,000-pf capacitors were attached to the line every 235 ft, only 12.5 db of loss was encountered.

At this writing inmine tests are being planned to determine whether or not this technique can be applied to a mine trolley line. In Consolidation Coal's McElroy mine, a section of trolleyline will be isolated and the characteristic impedance measured. The line will be loaded at various points and signal measurements will be made. The line will then be equipped with shunting capacitors, and the measurements will be repeated. If successful, the low-impedance line technique will be an alternative to the dedicated wire in situations where the installation of the dedicated wire is not desired or practical.

HAULAGEWAY HARDWARE, IMPROVEMENTS AND DEVELOPMENT

As mentioned before, trolley carrier phones have not undergone any basic improvements since they were first developed except for the change from vacuum tube to solid state circuitry. Also there seems to be an industry reluctance to move in the direction of more state-of-the-art circuitry. Whether or not reluctance is justified, there are several simple improvements that can be made to standard carrier phones that are of great value.

Under Bureau contract[4] an improved trolley carrier phone is being developed. Although it is still a simple TRF device, it contains several important different features.

The first feature is that it can receive both in the conventional manner, by voltage pickup from the trolleyline, and also via loop antennas. Inmine measurements have shown that standard carrier phones cannot communicate in the vicinity of heavy RF loads such as rectifiers because the RF voltage on the line is too low. However, RF current and the magnetic field (H-field) are generally high in the vicinity of such loads. Therefore, the situation is complementary. When RF voltage on the trolleyline is low, the current is high, and vice versa. The Bureau carrier phones permit manual selection between voltage pickup and magnetic pickup. A voting circuit could be installed to do this automatically. Inmine tests have consistently shown that magnetic pickup always exceeds voltage pickup, no matter where one is in the mine. It is also a cleaner signal. Because of these findings, there appears to be no reason why a carrier phone even needs RF voltage pickup from the line. Carrier phones with magnetic pickup would likely be extremely immune from dead spots.

Other features that these carrier phones have are phase lock loop FM detection, preemphasis/deemphasis, and a combination conventional-tone squelch circuit.

Another hardware improvement that was evaluated for trolley carrier phone use was an adaptive speaker volume control.[5] Most of the time the adjustment of the volume control on the speaker is a real problem on rail haulage vehicles. When the vehicle is in motion, the noise is so high that the operator sets the volume control to maximum. However, when the vehicle stops, the acoustic environment is very quiet. An incoming message at this vital time can virtually knock one over. Conversely, if the volume control is set low because the vehicle is stopped, failure to turn it up while in motion can be dangerous, for the operator may miss a vital message from the dispatcher or another vehicle.

The adaptive volume control is a small circuit that can be built directly into the speaker housing. A dynamic microphone samples the ambient noise, and a preset level detector activates a relay that applies full power to the speaker if the threshold is exceeded. Otherwise no action takes place and operation is via the volume control. A level detector inhibits the relay drivers from changing the relay setting during the reception of incoming messages, thereby eliminating latchup problems.

CONCLUSION

Several techniques have been discussed concerning ways to improve rail haulage communications. One method concerns using a low-attenuation dedicated wire, preferably not connected to the trolleyline by coupling devices. Strong signals on the dedicated wire couple strongly to the trolleyline, while unfavorable loads on the trolleyline couple only weakly back to the dedicated wire. Another method is to reduce the influence of loads on the trolleyline by lowering the line characteristic impedance. Other solutions have also been included, such as tuning, Z-boxes, use of phone lines, and remote location of the dispatcher's transceiver.

The benefits of medium frequencies were also discussed, and systems based on dedicated wires and medium frequencies were presented.

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THEORETICAL ATTENUATION RATE FOR TYPICAL TROLLEY WIRE/RAIL

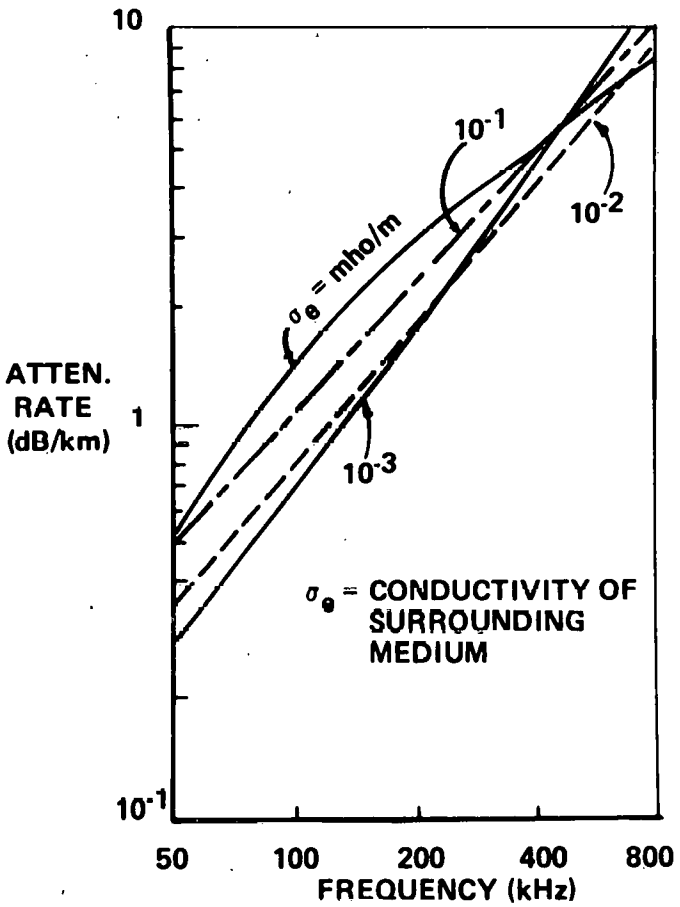


FIGURE 1.--Theoretical Attenuation rate for an unencumbered trolleyline

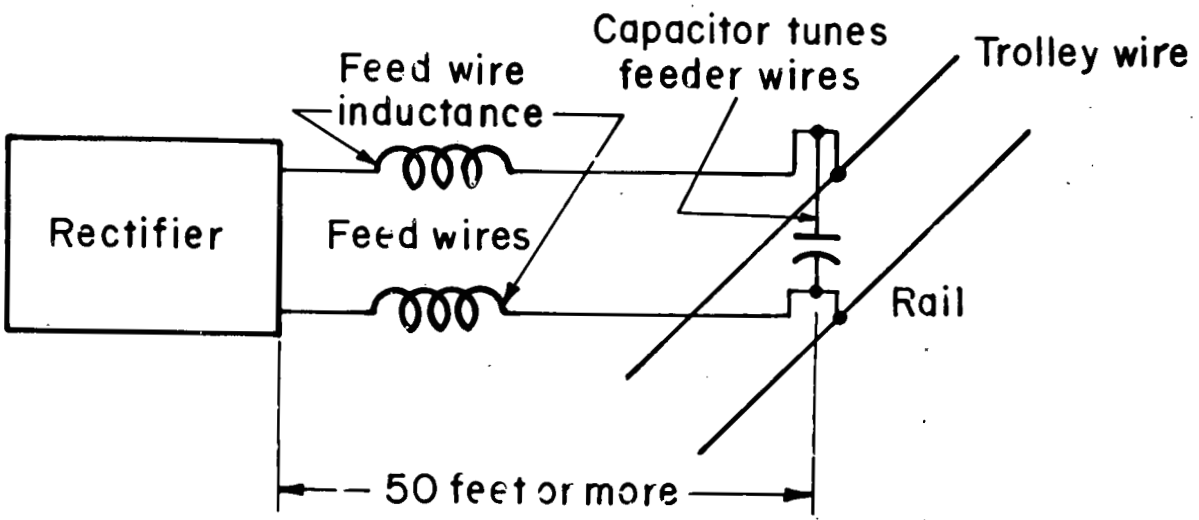


FIGURE 2.--Isolating a rectifier by tuning the feed wire inductance.

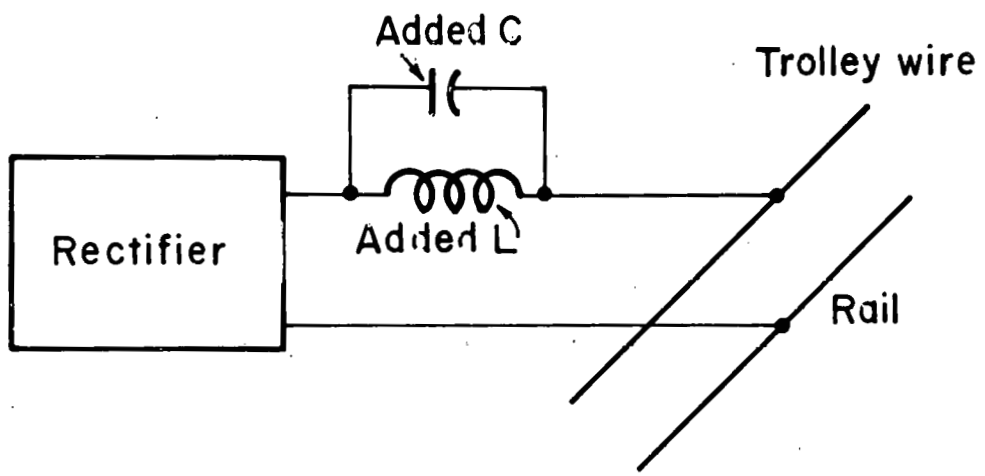


FIGURE 3.--Isolating a rectifier by adding L and C.

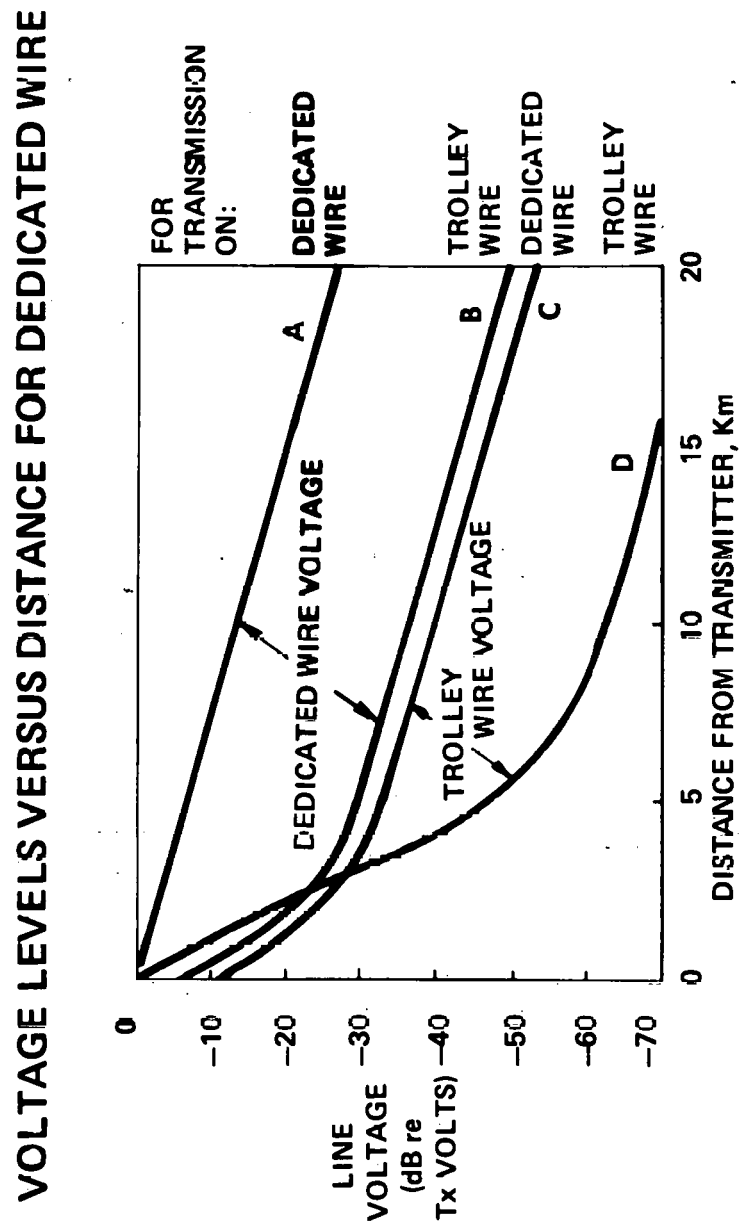
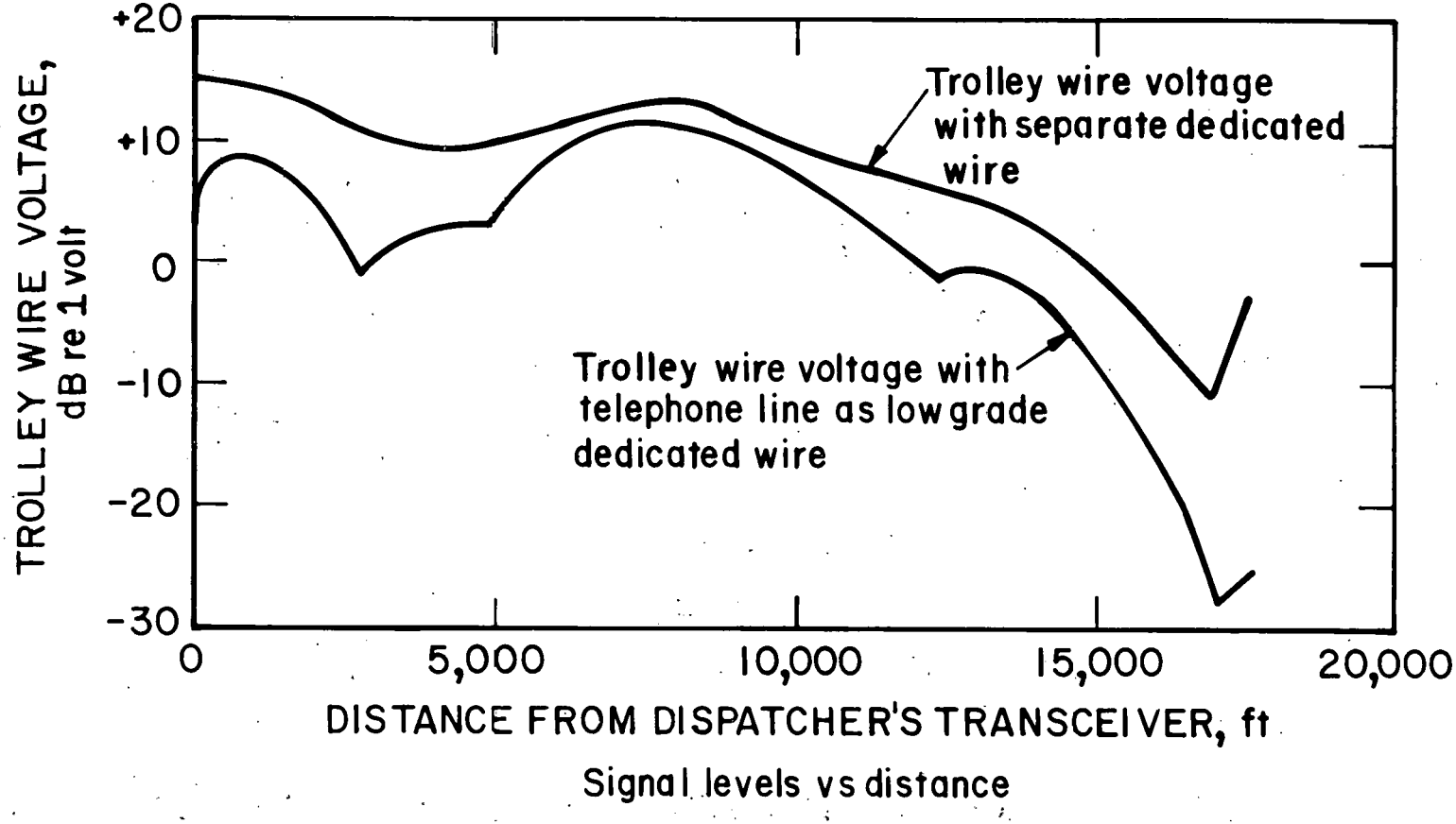


FIGURE 4.--Signal and distance for a dedicated wire.



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FIGURE 5.--Signal level versus distance with and without dedicated wire.

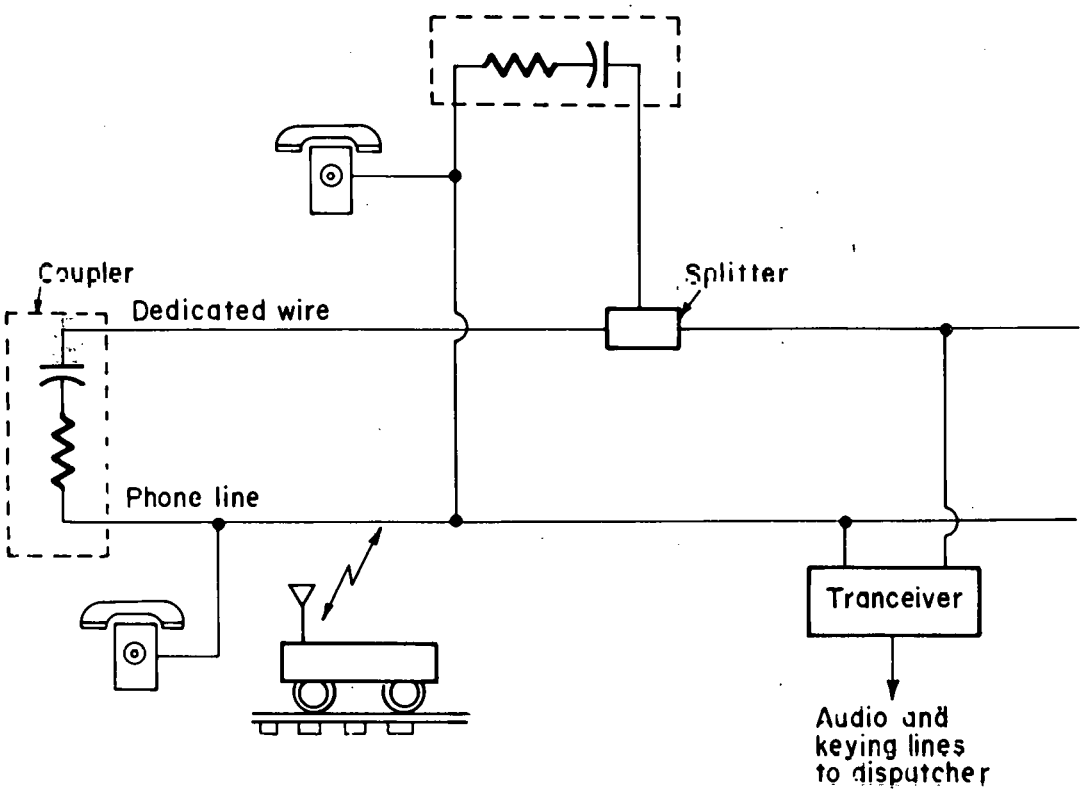


FIGURE 6.--Dedicated wire system for communicating with battery-powered rail vehicles.

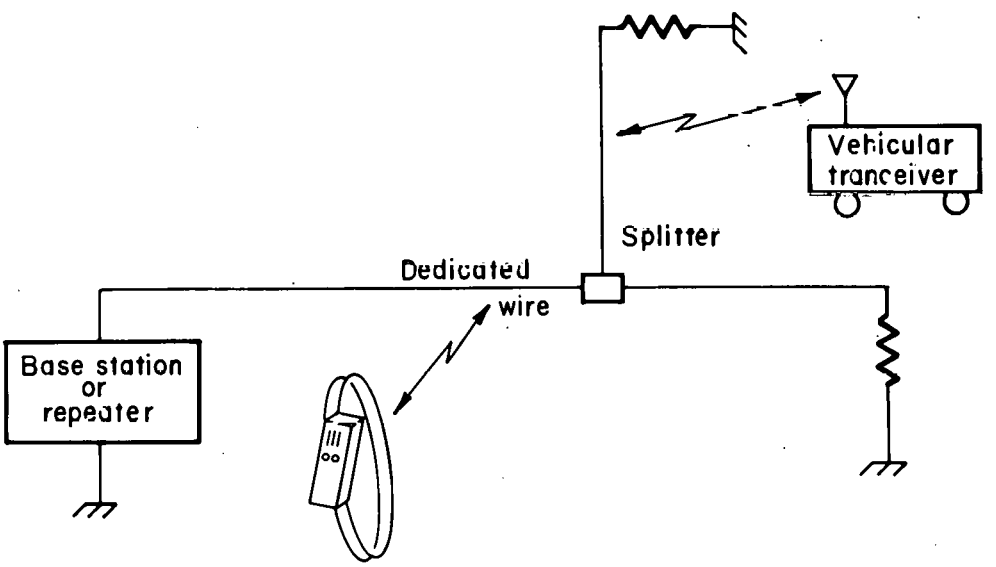


FIGURE 7.--Whole mine MF system using dedicated wires.

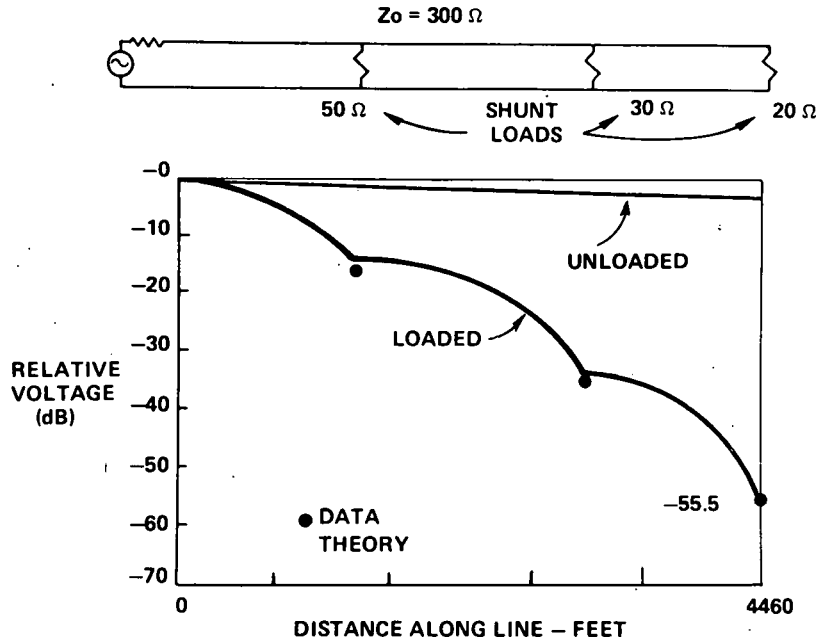


FIGURE 8.--Effects of loads on a simulated trolleyline.

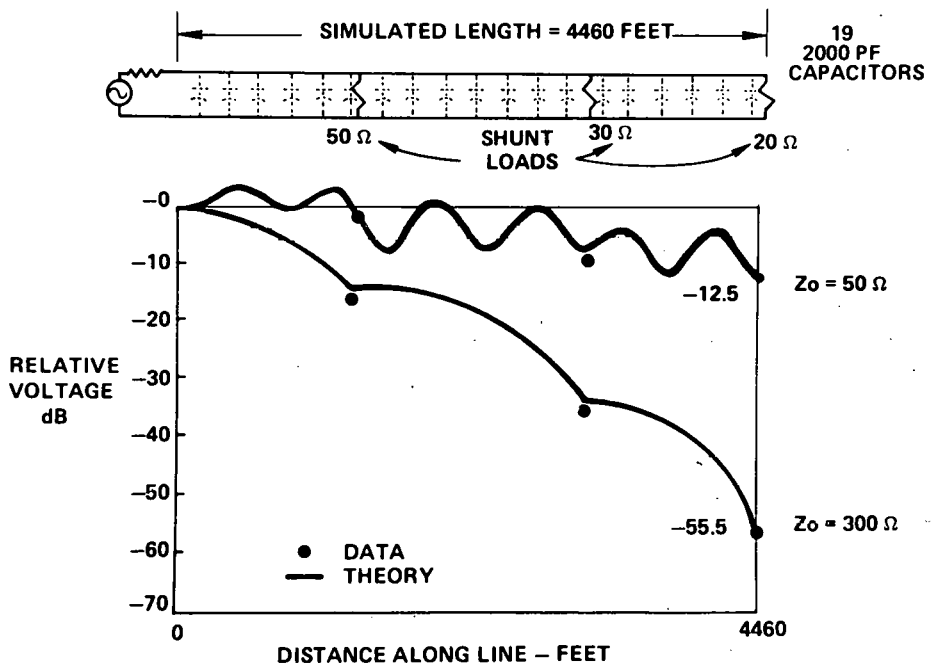


FIGURE 9.--Beneficial effects of reducing line characteristic impedance.

PORTABLE REMOTE CONTROL OF TROLLEY CIRCUIT BREAKERS

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Consolidation Coal Company
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ABSTRACT

The movement of off-track mining equipment in a mine entry under energized trolley or feeder lines is a hazardous procedure especially when there is less than 12" clearance between the mining equipment and the trolley. A portable Remote-Control System has been developed under the Bureau of Mines research contract Number H0366029 which is capable of controlling the overcurrent trip level setting of a D.C. Trolley circuit breaker. The system also has the capability of providing for the immediate shutdown of the trolley power by a member of the moving crew if the piece of mining equipment to be moved contacts the trolley wire or feeder.

BACKGROUND OF THE PROBLEM

Whenever off-track mining equipment is moved in an entry with energized trolley or feeder lines, there is danger that this equipment could come in contact with the trolley or feeder line, possibly causing serious damage by fire. To minimize this hazard, on October 31, 1973, MESA promulgated Section 75.1003-2, 30 CFR. This new regulation became effective on January 1, 1974.

Section 75.1003-2 lists the precautions that must be taken every time off-track mining equipment is moved in areas where energized trolley wires are present. It also lists additional precautions that must be taken when there is less than 12 inches of vertical clearance between the off-track equipment and the energized trolley wire. Some of these additional precautions are:

1. No person shall be permitted to be inby the equipment being moved.
2. The settings of all circuit breakers providing short circuit protection for the trolley wire shall be reduced to not more than 50% of the short circuit current that would flow if the equipment contacted the trolley wire.
3. A miner shall be stationed at the first circuit breaker outby the equipment being moved at all times.

He must be in direct communication with the miners actually moving the equipment. In fact, he shall have two independent means of establishing two-way voice communications.

The moving of off-track equipment in the presence of energized trolley lines creates a hazard to personnel, equipment, and even the mine itself. This hazard has a direct bearing on productivity and the well-being of all personnel in the area.

The successful development of a system for the remote control of circuit breakers should increase the probability of automatically tripping the circuit breaker in case of an accident. It should also substantially reduce the time required to manually trip the circuit breaker if necessary.

The objective of this contract was to design and demonstrate a reliable system to remotely control trolley line circuit breakers from a location close to the off-track mining equipment that is being moved in the vicinity of energized trolley or feeder lines and incorporate the additional voice communication devices required by Section 75.1003-2 30 CFR into the system.

Information was obtained from several operating mines on the details involved in the movement of off-track equipment in the mine. Supervisory, production and maintenance personnel from each of the selected mines were interviewed in order to answer questions such as:

1. What vehicles are used during an equipment move?
2. What safety equipment is used?
3. How many persons are required, along with their particular job functions?
4. What is the procedure once the move is started?

The answers to these questions made it possible to determine the necessary control functions required for the Remote Control System, along with the individual that should be in command of the system.

Preliminary field tests were necessary to determine the appropriate method of transmitting and receiving control information from the piece of mining equipment being moved to the trolley circuit breaker location. The tests determined that the carrier frequency range (60kHz-190kHz) would be the most appropriate frequency for the transmission of control information.

A preliminary prototype Remote Control System was designed and fabricated in accordance with the information obtained from the field test data.

The prototype equipment was successfully tested at the Shoemaker Mine of Consolidation Coal Company during four equipment moves. The overcurrent trip setting of the controlled circuit breaker was considerably lower than the normal settings of the circuit breakers during a

regular operating shift. Typically the trip settings were reduced to 300 amps during most of the equipment moves instead of trip levels near 3,000 amps during normal mine operations (300 V dc Trolley).

Because the preliminary field test program was successful, it was decided that the final prototype systems should be designed and fabricated as production prototypes.

PRODUCTION PROTOTYPE SYSTEM

The Remote Control System consists of three major subparts:

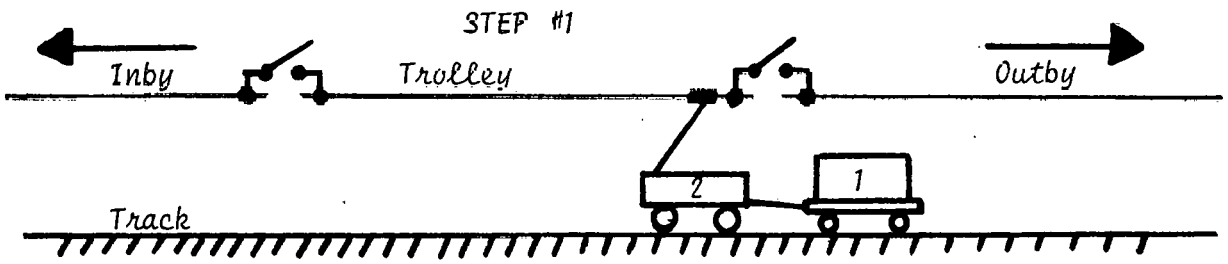
1. Transmitter Control Unit and UHF "emergency trip" radio.
2. Receiver Unit, Overcurrent Relay.
3. Circuit Breaker Unit.

The system will be used according to figure 1 for pulling a piece of off-track mining equipment into the mine. The reverse procedure applies to pulling a piece of equipment out of the mine.

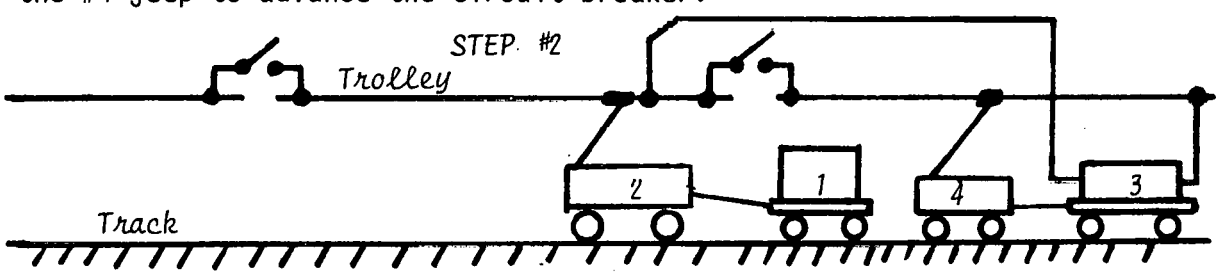
EQUIPMENT LIST

1. Off-track mining equipment mounted on a flat car.
2. Locomotive with the Transmitter Control Unit.
3. Circuit breaker and the Receiver unit mounted on a flat car.
4. Small utility jeep.

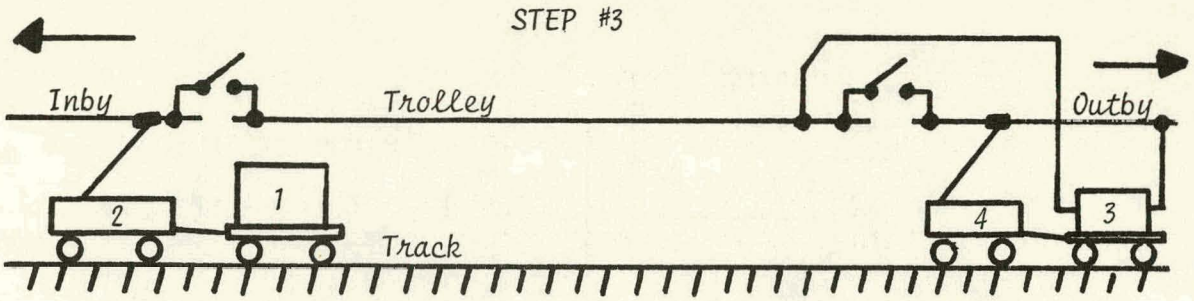
FIGURE 1
PULLING A PIECE OF OFF-TRACK MINING EQUIPMENT INTO THE MINE



The locomotive operator advances the locomotive until the locomotive trolley pole is on the inby side of the dead-block. The off-track mining equipment being moved is still on the outby side of the dead-block. At this point, the locomotive operator gives permission to the operator on the #4 jeep to advance the circuit breaker.



The operator of the #4 jeep, a certified electrician, disconnects the circuit breaker and advances toward the locomotive. He then disconnects the circuit breaker around the disconnect switch at the locomotive.



The locomotive operator selects one of the eight possible settings for the solid state overcurrent relay, located at the Receiver unit, and then closes the reset switch. The circuit breaker closes and the setting of the solid state overcurrent relay is determined. The locomotive can now pull the off-track mining equipment toward the next inby disconnect switch. During this move, the energized trolley wire above the equipment is protected by a circuit breaker, which has, reduced overcurrent setting and can be opened by the locomotive operator. If the Receiver Unit fails to receive the signal from the Transmitter Control Unit, the circuit breaker will open and it will not reclose until a reset signal is received.

Figures 2 and 3 show the Transmitter Control Unit and the Receiver Unit solid state overcurrent relay, respectively. The transmitter Control Unit and Receiver Unit were developed under the U.S.B.M. Research Contract No. H0366029.

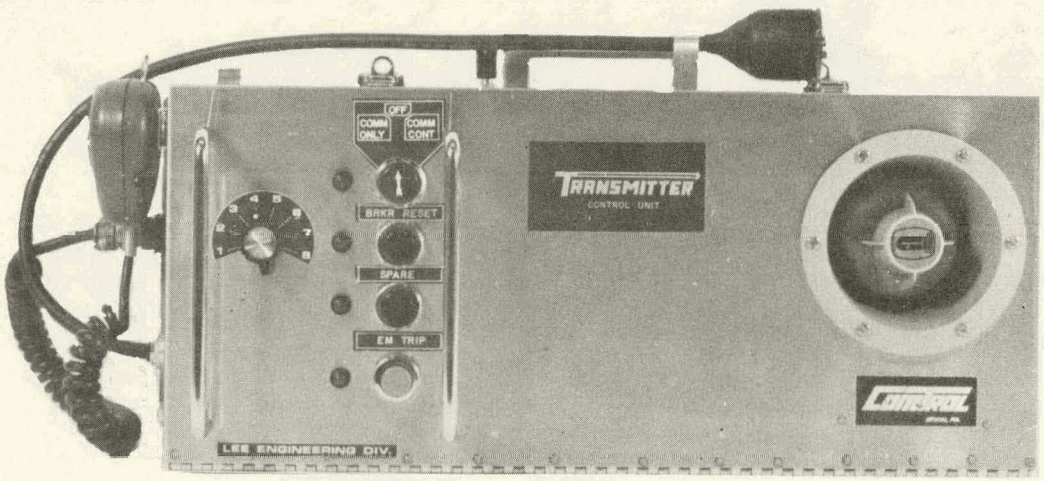
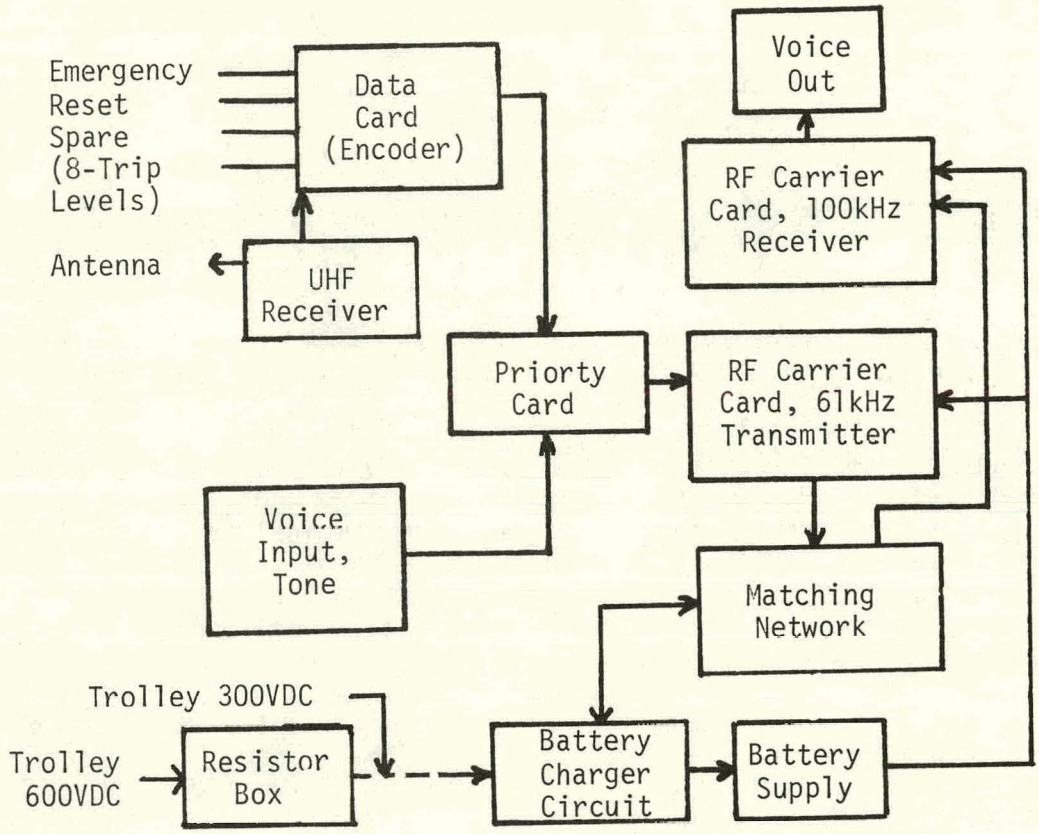
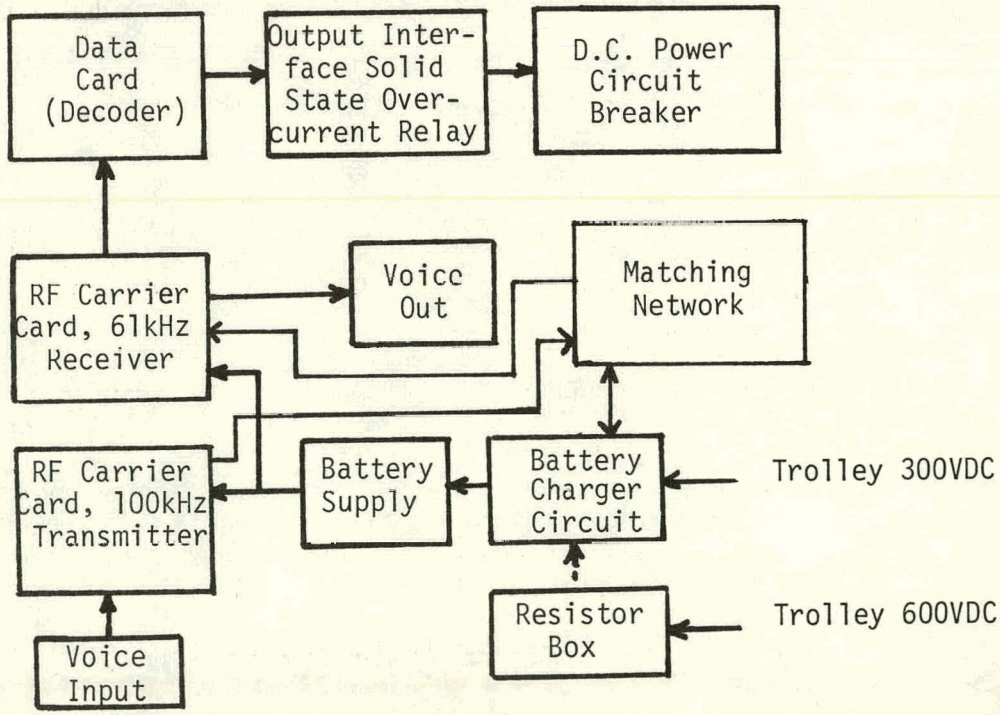


FIGURE 2

Picture of the Production Prototype Transmitter Control Unit



Block Diagram of the Production
Prototype Transmitter Control Unit
Figure 4



Block Diagram of the Production
Prototype Receiver Unit
Figure 5



Figure 3

Picture of Production Prototype
Receiver Unit with the Solid
State Overcurrent Relay

The Remote Control System has the following capabilities:

1. The system will be able to trip the power circuit breaker automatically in the event the mining machinery has come in contact with the trolley line. Other ways to trip power are listed below:

   a. If contact between the machinery and the trolley line is made and the current flow exceeds the trip setting, the breaker will open.
   b. The operator can depress the red pushbutton located on the Transmitter Control Unit. This will transmit a signal to the Receiver Unit which will open the power circuit breaker supplying power to the trolley line. The pushbutton is labeled "Emergency Trip".
   c. The Transmitter Control Unit also has the capability of an emergency trip by means of a portable, remotely carried UHF radio. This UHF radio can be used by an individual walking alongside the equipment move in the event that low coal conditions exist and clearances are minimal between the mining machinery and the trolley line. When the UHF emergency trip signal is detected by the Transmitter Control Unit as discussed above, a trip message is transmitted to the Receiver Unit.

      An individual carrying a UHF transmitter will normally be within 50 to 100 feet of the equipment that is being transported. The transmitting distance for the UHF radio under most mining

conditions should be in the range of at least a few hundred feet, which is more than adequate for this application.

d. Loss of carrier frequency signal from the Transmitter Control Unit after a predetermined adjustable time delay will result in the tripping of the circuit breaker. This feature will monitor the integrity of the system.

2. The Transmitter Control Unit will have the capability of resetting the Circuit Breaker Unit in the event that the power was tripped unintentionally or because the trip setting of the circuit breaker was initially set too low.

3. The Transmitter Control Unit will be able to adjust the overcurrent trip setting of the circuit breaker to one of eight fixed predetermined current levels. The eight overcurrent trip levels can be selected as shown on the front panel of the Transmitter Control Unit. A spare function switch has also been located on the front panel of the unit. This switch can be used to operate any desired control function necessary during the move as required. The spare function may operate a light, buzzer, bell or some other device to indicate a particular function.

The transmission of control data is based on a three-tone multicode concept. The tone code groups are in the audio frequency range (300-3000HZ). The appropriate tone groups are used to select the eight different overcurrent trip settings of the circuit breaker and also the "emergency trip", "breaker reset" and "spare" function controls.

The Transmitter Control Unit also has the capability of two-way radio communication with the Receiver Unit. The operating frequencies of the system can be selected appropriately from 60kHz to 190kHz.

The Receiver Unit will receive the control data and convert this information to the overcurrent trip level settings of the circuit breaker and also the various control functions; "Emergency Trip", "Breaker Reset" and the "Spare". The control information from the Receiver Unit is interfaced to the Circuit Breaker Unit through optical isolators which provide up to 2500 volts of electrical isolation between the trolley circuit breaker and the 12VDC receiver circuitry. The receiver will remember the last correctly received message and will not change status until another legitimate message is received or a loss of carrier frequency signal is indicated. Light emitting diodes (LED's) are used to continually display the status of the remote control system. Two-way voice communication with the Transmitter Control Unit is also included.

The control data for the Transmitter Control Unit is based upon a MOS/LSIFX207 chip[1] which is a logic programmable multicode transmitter which transmits any one of eight different code groups, all based on permutated sequences of the three programmed frequencies given by three separate resistor values. The desired code selection is

[1] Consumer Microcircuits of America.

carried out by supplying logic levels to the data inputs in 3-line binary coded form. A repetitive transmission scheme for the code groups is used in order that a continuous monitoring of the system can be accomplished. Each code group transmission is preceded by a delay interval determined by an internal monostable circuit. This is an adjustable delay which has been set to less than 1 second. This time delay provided a quiescent interval between successive code group transmission during cyclic mode operation, and the delay period is automatically re-triggered on completion of each transmitted code group. Different code data may be entered or changed any time, but only the data levels present at the instant the delay period expires is transferred to the internal storage area and the corresponding code transmitted. Once transmission commences it cannot be cancelled or the code group changed. To guard against the false transmission of data, the code group consists of a series of three tones. The frequencies used and their order of transmission determines the code value. The code group encoding technique used in this system operates according to the following rules:

1. All three tones in a code group are of equal nominal duration which must be sufficient to allow recognition by the Receiver Unit.
2. Consecutive tones in each code group must be of different frequency; example, frequencies A, B, C or B, A, C are legal codes; however, A, A, B or B, B, A would not be accepted as a legal code.
3. The Receiver Unit will recognize a correctly addressed code "only" if consecutive tones in the group are received within a specified gate period.
4. A minimum time interval (quiescent period), must elapse between transmission of consecutive code groups in order to avoid false codes based on a "run-on" tone sequence present in two consecutive code groups.

The FX-307 is the heart of the Receiver Unit which receives the tone code groups and decodes them to useful information. Three resistors are used to determine the frequencies that the Receiver Unit will decode. These three resistors are closely matched to the three frequency determining resistors located on the FX-207 encoder chip. This insures that the tone groups will be within the passbands of the input filters in the Receiver Unit. At least 10 cycles of each tone must be received in each code group to insure correct recognition. The tone recognition system employed by the FX-207 is based upon a period sampling technique. Input signals are amplified and shaped to provide clock signals to a bistable flip-flop, the output of which is a square wave having a period equal to the interval between successive input wave form zero-crossings. The flip-flop output triggers a dual monostable, which generates reference periods that represent the tone channel edges (bandwidth). These periods are determined by resistor-capacitor combinations and a D.C. voltage reference level. The reference periods are compared against the signal input period in a comparator, the output of which controls a special counter/storage system. This counting system discriminates against random spurious information and delivers an output only when a true inband tone signal is received. Gating circuits are incorporated which prevent false operation when harmonics of the tone frequency are applied. This tone recognition system yields extremely sharp

tone channel definition, coupled with a fast response time and high immunity to false turn-on due to outband noise, regardless of the noise frequency and amplitude. The period sampling system requires, however, a regular input waveform for correct operation. Input noise can if mixed with the required inband tone, prevent the circuit from recognizing that the tone is present. This effect is produced because the noise jitters the interval between successive signal zero-crossings and the comparator/counter rejects the information as not being consistent with an input signal having a frequency actively within the tone channel limits.

If the noise is random in nature, such as noise spikes or short term interference, it will have little effect other than possibly to increase the overall tone recognition time by a few cycles.

If the noise is continuous, such as when parallel inband and outband tones are on the signal line simultaneously, the device will not recognize the inband tone. If the noise is semi-continuous, such as being mixed with speech signals, increasing the inband tone duration will allow the tone to be recognized during a momentary "noise" pause. This system operates on a basis that voice and data are never sent at the same time. This will reduce or eliminate parallel inband tones from delaying the proper receipt of a functional message. The "Emergency Trip" data has been given priority in the transmitter Control Unit over voice messages and other information. This insures that in an emergency situation, the Circuit Breaker Unit will be tripped as soon as possible.

CIRCUIT BREAKER UNIT

The third major part of the remote control system is the Circuit Breaker Unit which consists of the following subparts:

1. D.C. circuit breaker.
2. Electrical shunt.
3. Solid state overcurrent relay.
4. Insulated single conductor power cable with heavy duty clamps attached to the ends of the cable.
5. Heavy duty metal enclosure used to house items 1 through 3.

The solid state overcurrent relay is connected to the Receiver Unit by means of optical isolators located within the Receiver Unit which provide up to 2,500 volts of electrical isolation between the two pieces of equipment. A set of normally open contacts located within the overcurrent relay must be closed to pick up the "M" contactor coil which will apply d.c. power to the trolley wire. The normally open contacts will only be closed under the following conditions.

1. The Transmitter Unit must be transmitting a carrier frequency signal of the proper amplitude.
2. The Receiver Unit must be turned on and must receive the proper "Reset" code and trip level data from the Transmitter Unit.
3. The solid state overcurrent relay must be operating properly.

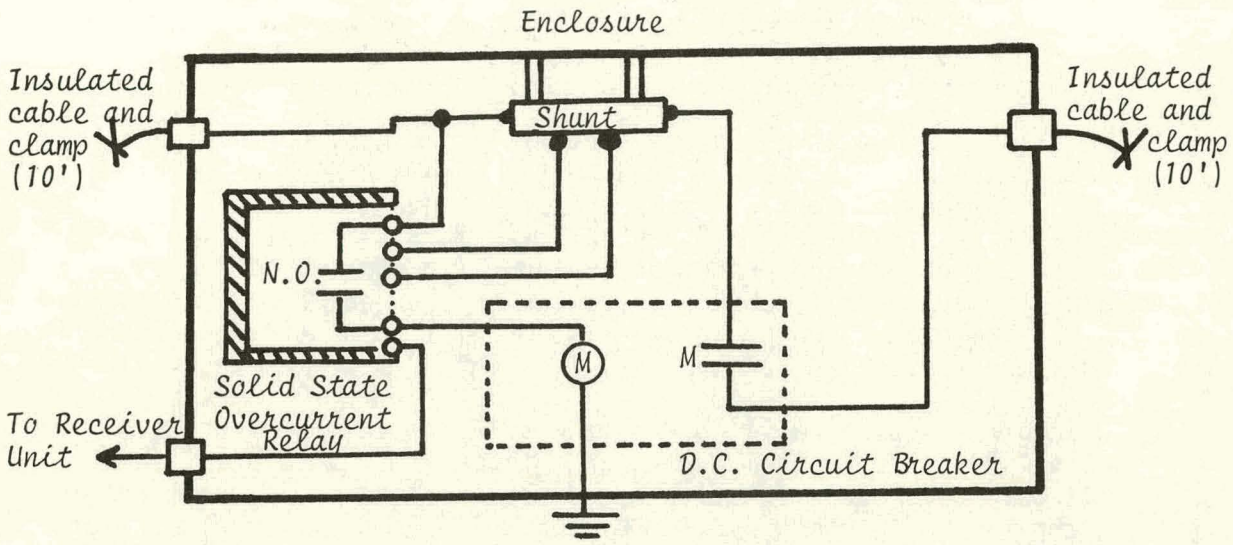


FIGURE 4

Wiring Diagram of the Circuit Breaker Unit

The physical size of the circuit breaker unit enclosure will depend upon the type and size of the D.C. circuit breaker to be used. The guage of the insulated power cable depends upon the amperage requirements expected during an equipment move. Figure 5 illustrates the Receiver Unit and the Circuit Breaker Unit mounted on a vehicle during the equipment move.

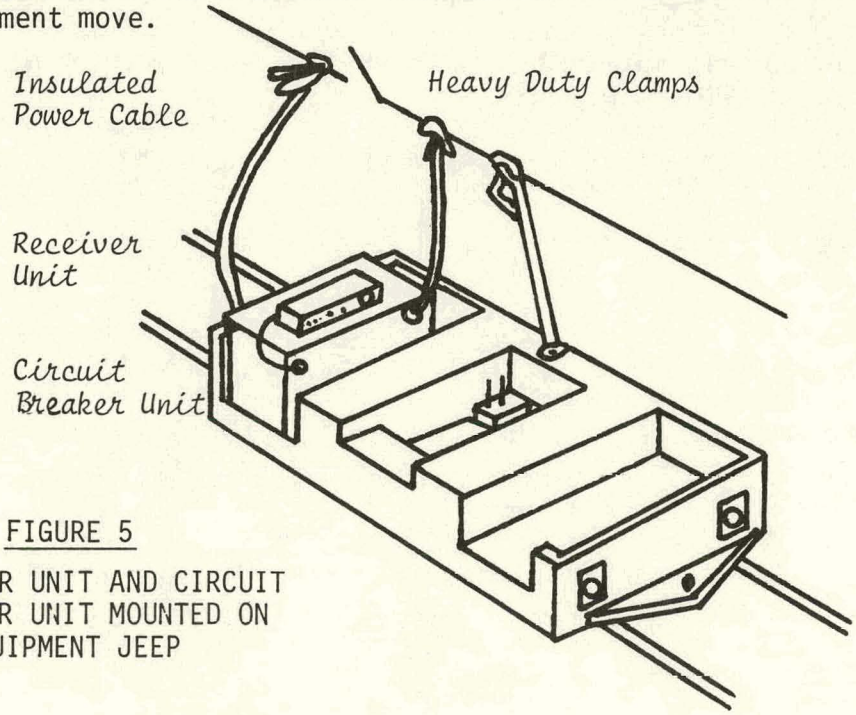


FIGURE 5

RECEIVER UNIT AND CIRCUIT BREAKER UNIT MOUNTED ON EQUIPMENT JEEP

CONCLUSION

A preliminary prototype Remote Control System has been designed, fabricated and successfully field tested in the Shoemaker Coal Mine of Consolidation Coal Company on four separate equipment moves. The information and experience gained from the field test program made it possible to design and fabricate a production prototype system. Two such systems have been fabricated and successfully bench tested. It is expected that by the fall of 1978 the two production prototype systems will be placed into operation at two different operating coal mines.

The control systems will control the D. C. circuit breaker used for the trolley protection system during an equipment move. The testing program will be a completely functional test controlling the circuit breaker and the two-way voice communication system which is an integral part of the overall system.

A detailed explanation of the control system will be available as soon as the printing of the final report has been released by the United States Bureau of Mines. The control system which has been designed and fabricated as a production prototype system, will be available from Control Communications Inc., upon the final field testing of the prototype hardware.

ACKNOWLEDGEMENT

This project was accomplished under United States Bureau of Mines Research Contract No. H0366029.

Personnel at the Shoemaker Mine for making the field test program successful.

AN ACOUSTIC COUPLER FOR MINE TELEPHONE SYSTEMS

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Adrian C. Billetdeaux

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ABSTRACT

The requirement by many state laws for direct communications between underground personnel and local health care facilities has created the need for a means of coupling the underground to the public telephone system. MSA has chosen acoustic coupling to achieve this. The need for flexibility and MESA's requirement for intrinsically safe telephone lines and systems underground was instrumental in evolving the Acoustic Coupler's design.

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Voice communication is one of the most effective means of message transfer presently available to be used underground. This communication is normally done with hard-wire, party line type of telephone systems in most mines (Pager System; MSA Trademark). When assistance is needed, for example, a simple "help" call is heard throughout the mine over the telephone system. This characteristic has made the system a critical link in the mine operation. The operation of the units in this type of system is simple and straightforward: Push to Talk, Release to Listen (a battery saving and noise reducing operation); Push or Pull to Page (an operation that allows a person to have his message broadcasted over the loudspeakers of the other units).

In most mining operations, there are two telephone systems: (1) the public telephone system aboveground for normal business traffic, and (2) the company owned telephone system underground for working traffic. The mine operator has sought a way to tie the two together. Many systems have been made to update the underground units to make it look and act like the aboveground public system, but this is not what is needed in many cases.

Each existing system works and works well in its own area; a simple tie link is all that is needed. This link should permit temporary hookups for exchange of message flow between the two. Permanent connection of the two together would put unneeded traffic flow on both systems with possible overloading to one or both systems.


Pager III
part no.
485177
permissible mine communication unit
mesa
Approval No. 98 — 85
this approval and this plate apply only so long as the electrical units and wiring are maintained in accordance with MINING ENFORCEMENT AND SAFETY ADMINISTRATION permissibility requirements.
MFD BY
MSA
MINE SAFETY APPLIANCES COMPANY
PITTSBURGH PENNSYLVANIA U.S.A. 15208

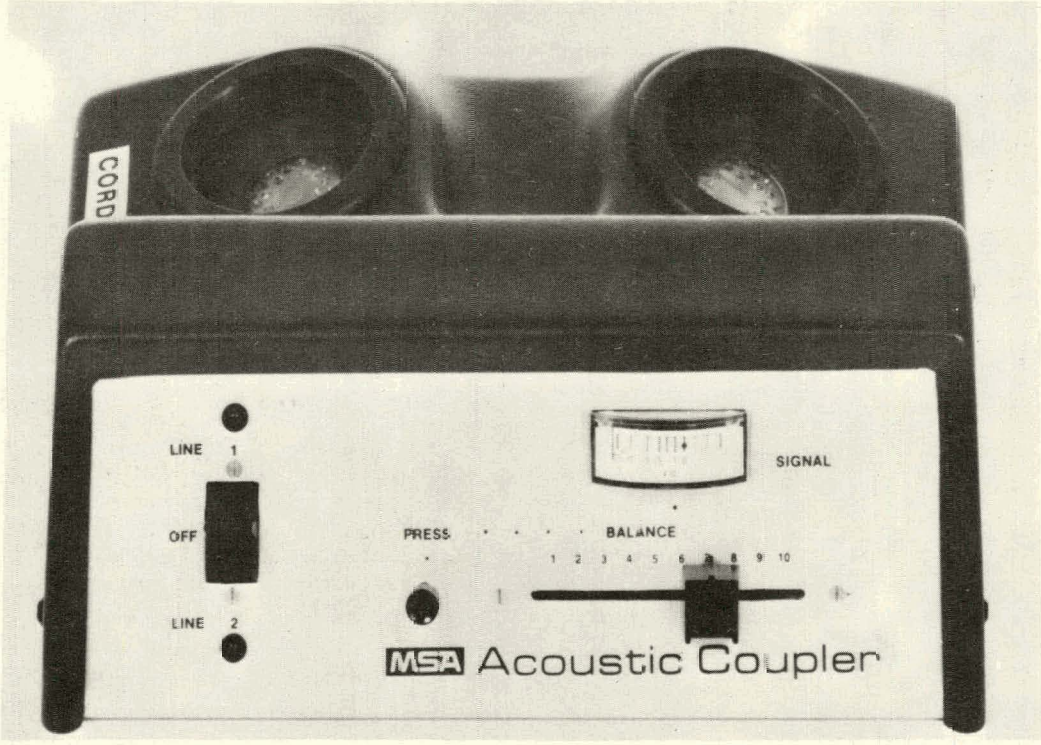
The real need for such a link became evident when underground personnel desired communication with health care facilities for emergency medical advice and instructions. And, in fact, West Virginia Senate Bill No. 285, effective January 1, 1978, requires that all underground telephone systems be capable of communicating from within the mine directly to health care facilities.

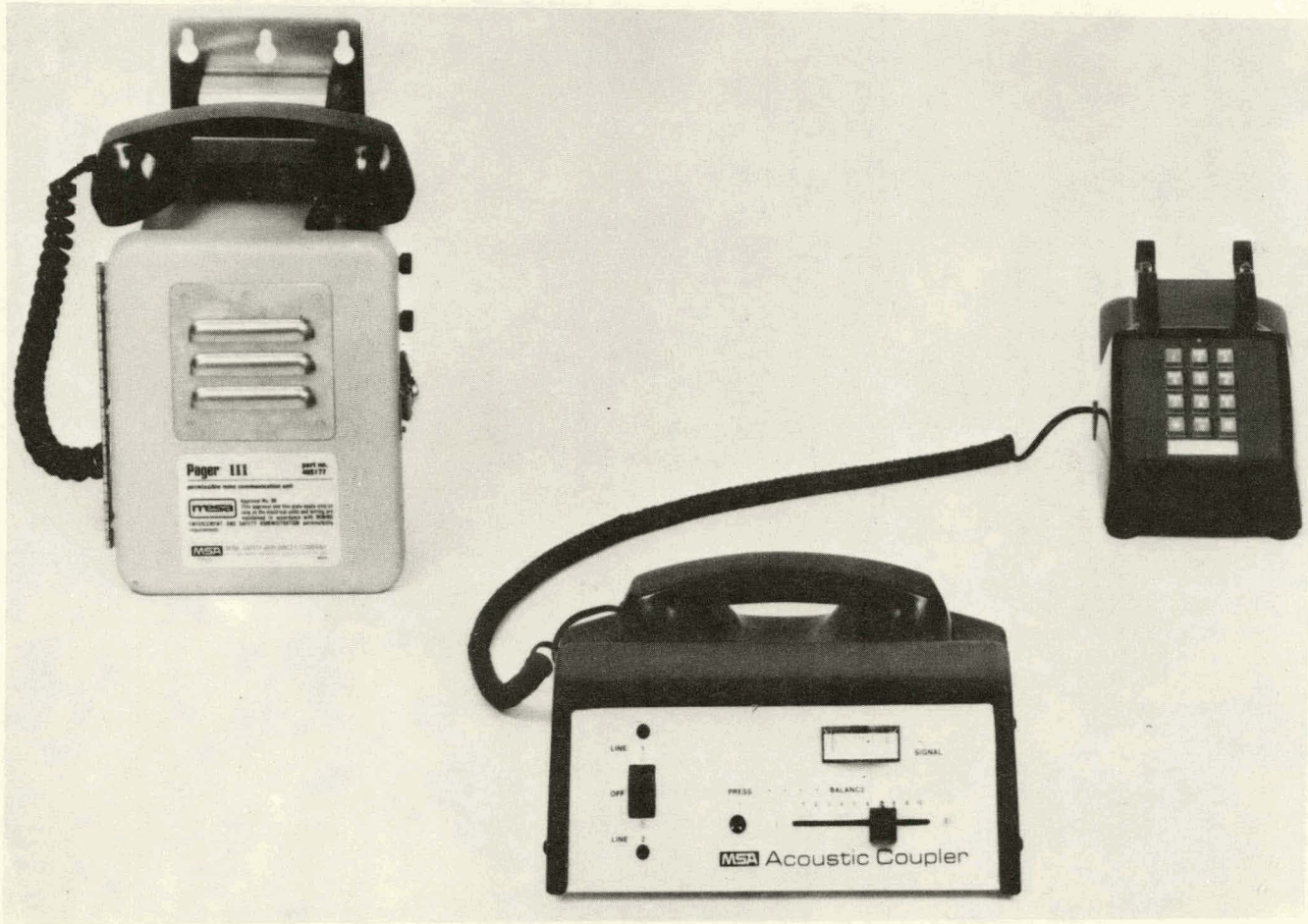
In order to perform its functions, the design of the link must consider the basic parameters of both systems. Reduced to its simplest, the public telephone system maintains a single line to every customer. A manual or semiautomatic switchboard is necessary, for switching signals around these lines, and controlling the signal levels, along with the proper loading of these lines. The underground telephone system is in many cases a single line with many units on it, operating at uncontrolled signal levels with unknown loading. Line integrity is usually not known as lines are not maintained or controlled to a standard. Each company installs their own telephone system and maintains it to their own standards. Thus, underground telephone installations can and do vary tremendously between systems. Therefore, to be workable, the link must of necessity have a large working range on the underground end of the link.

To explain its application and use, let us take a theoretical mine that has one underground telephone system using 30 Pager III (MSA Permissible mine telephone unit) telephones on a long line (10 miles of Number 12 gauge solid copper wire) and a C&P Telephone Co. of W. Va. public system (PBX of 80 telephone hookups; 4 outside lines). The underground system has the following parameters. Pager III: unit loading, AC 2500 ohms, DC 33.3K ohms; output voltage, 11 VAC RMS open ckt.; output impedance, 110 ohms. Line: Resistance per mile, 8.4 ohms; Capacitance per mile, 14.3 pf.

The PBX system has the following standards per unit; receiver output sound level 60 dB to 80 dB; transmitter, input sound level 70 dB to 100 dB (values are those for normal listening levels).

The coupler unit is hooked up directly to the underground telephone line and placed close to one of the public telephone sets. The acoustic link is made by placing the public telephone handset in the unit's cradle, which now ties the two systems together for communication. For proper operation, the signal level on the underground mine telephone line should be between 100 and 600 millivolts RMS for good, clear communication between systems. Note underground line impedance should not be below 50 ohms A.C. for proper operation of the link. The power level delivered to the public telephone handset transmitter is about 94 dB. An AGC circuit to hold this level over a 20 dB range of the underground line signal is designed into the link. The link is designed to deliver 300 millivolts RMS of signal to a 100 ohm underground telephone line with a power level of 74 dB out of the public telephone handset receiver. It should be pointed out that this level is dependent on the line impedance and the sound level of the public telephone handset receiver. The coupling medium being air allows the unit to be designed for permissibility and operate on permissible lines. The unit is powered by a NEDA

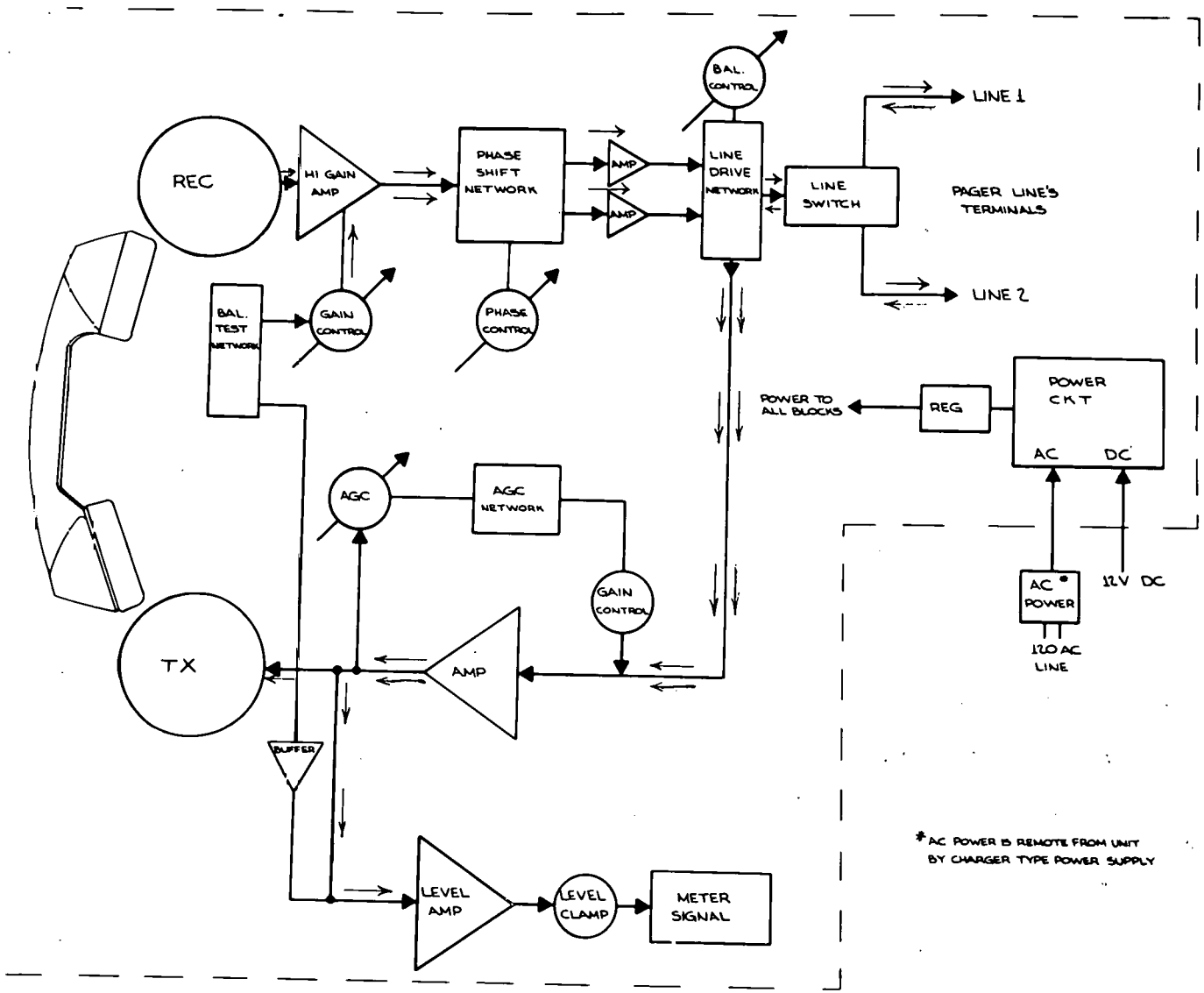

CORD
LINE 1
OFF
LINE 2
SIGNAL
PRESS
BALANCE
1 2 3 4 5 6 7 8 9 10
MSA Acoustic Coupler


Pager III
LINE 1
OFF
LINE 2
SIGNAL
PRESS
BALANCE
MSA Acoustic Coupler

926 Dry Cell Battery (permissible) or optional AC Power Pack (non-permissible). The optional AC power pack is not permissible at this time, but will be in the future. The balance network allows the two-way communication to move smoothly between the two systems even with minor impedance changes in the underground telephone line. The unit is small and can be moved easily from one desk to another. There are no special public telephone taps to work with. The link is made only when needed, thus limiting the traffic on both systems.

Finally, the use of such a link is not limited to emergency use. The link will allow the man underground to communicate with any point that can be reached through the public telephone system. Electricians can talk to superintendents at home, mechanics can talk to parts suppliers directly without leaving the point of inspection, and superintendents can be underground and can handle calls without waiting. Because the unit is man operated, the amount of unauthorized use can be held to a minimum.

ACOUSTIC COUPLER
BLOCK DIAGRAM



BAL. – BALANCE
AGC – AUTOMATIC GAIN CONTROL

UNIT HAS BEEN APPROVED FOR W.VA. SENATE BILL 285 BY MR. W MILLER, DIRECTOR OF MINING, W.VA.

THE ARTHUR D. LITTLE ZONE PAGER PROGRAM

Martin L. Cohen
Arthur D. Little, Inc.
Acorn Park, Cambridge, MA 02140

ABSTRACT

A system for communication, monitoring and control is described which makes use of the twisted pair pager phone line found in almost all underground coal mines. The system has a capacity of 1000 remote units controlled from one central station and can transmit monitor measurements with less than one percent quantization error. The central control makes use of a microprocessor and is programmed for the desired operations. Because of this the system is inherently flexible and even the above size and data accuracy limitations are not limits. With additional memory capacity and the necessary program, almost unlimited size and sophistication are possible.

INTRODUCTION

The original purpose of the zone pager development program was to expand the capabilities of the visual pager developed under U. S. Bureau of Mines contract No. H0252093. The visual pager system is limited to a display with fifteen different discs, and while many display units may be connected to the mine pager phone line, all of these will respond in unison on the dispatcher's signals. This fifteen disc capacity was judged inadequate for many mine applications and therefore this program was begun under the sponsorship of U. S. Bureau of Mines, Contract No. J0177020 to develop a system which would have much more capacity and flexibility.

There are two key concepts in the visual pager. First is the idea of using a visual signal, a display disc, controlled from a central station for paging. An audible page is transitory and is gone with its echo, but a visual page remains to indicate that some one is wanted. Second is the idea of using the pager phone line to control the visual pages. The resulting system is economical because no additional wiring is needed.

While studying the requirements for an expanded visual paging system, the importance of the second concept was recognized. The pager phone line reaches most of the mine workings and can be and should be used to communicate, page, monitor environment and control equipment underground.

Another factor which would increase the usefulness of a communication system using the pager phone line is inherent versatility or flexibility. The system should be readily adapted to perform any or all func-

tions which at present may be thought useful in a mine and should also be able to handle most functions which may be thought of at some later time.

The following system has been designed:

- A central control unit to be located at the pit head or mine office. This unit will transmit signals to remote units on the phone line and receive the answering signals from these remote units. The central control unit will use a microprocessor and be programmed to perform the designed functions. System flexibility will be achieved through programming the central unit.

- Visual pager remote units located in the mine. These will be made up from single disc display modules. Multiple display units will consist of the required number of single disc modules mounted in a common housing. Each module will have a unique address so that it can be individually selected by the central control. All communications will be initiated by the central control. Upon receiving its address and a coded operation instruction, the visual pager unit will echo back the address and the operation code. These units will respond to three operation codes: post, which will set the display disc to post a page; clear, which will reset the display disc to clear a page; and test, which will not change the state of the display.

- Monitor units located in the mine. These will have their own individual addresses like the visual pager units and will respond to calls initiated by the central control unit. The echo response from a monitor unit will consist of the unit address with measurement data, replacing the operation code.

- Remote control units located in the mine. These will provide for remote control of pumps, fans, regulators or other equipment in the mine. Each will have its own individual address and will echo this address and operation code when called by the central control unit. For increased reliability these may use a two step control cycle. First, an initial call will be transmitted consisting of the address and operation code. This will be echoed back by the remote unit to indicate that it is on-line and ready to operate. A second call will then be transmitted which will cause the remote control unit to perform the desired operation. The second echo will indicate that this operation has actually taken place.

- Electromagnetic, call alert, page units located in the mine. These units will respond to a number of different

addresses as transmitted by the central control unit. After echoing the address and operation code, the electromagnetic page unit will retransmit a binary coded signal on a 3000 Hz carrier through a loop antenna. This signal will be received by a call alert pocket page receiver.

In a large system, more than one central control unit may be used, perhaps one for paging and one for monitoring and control. These may use different carrier frequencies or else may be interlocked to time share the pager phone line on a noninterfering basis.

SIGNAL TRANSMISSION

The signals for the zone pager will be transmitted by phase inversion modulation of a carrier frequency. This method of modulation has the advantages that it is easy to modulate and demodulate the signal and that it is highly efficient. Most of the transmitted power is contained in the sidebands which convey the information. Only a small amount of power remains at the carrier frequency. Phase inversion, in theory, has a 3 db signal-to-noise advantage over frequency shift keying. Both the carrier signal and its complement, that is, the inverted signal, will be fed through gates to the transmitter power amplifiers. The phase inversion modulation is obtained by gating one phase to the amplifier for a "one" and the other phase for a "zero". Demodulation will be done with a phase locked loop. Whenever the incoming carrier changes phase, the phase locked loop will momentarily lose lock and produce an output pulse. Note that the phase locked loop produces a "one" (a pulse) whenever the phase changes and a "zero" (no pulse) if there is no change of phase. In a sense this output is the derivative of the modulation on the carrier. In order that the received and transmitted signals may be the same in this system, all signals will be recoded just before modulation.

Published data shows that the spectral density of noise on the pager phone line has a broad minimum at about 16 kilohertz. Attenuation increases with frequency by about 0.25 db/mile/octave at this frequency. A standard electronic watch crystal (32,768 Hz) will be used to produce the 16,384 carrier frequency and, through a countdown chain, the 128 bit per second data rate and all other timing signals.

The signal from the central control station to a down mine receiver consists of two bytes of eight bits each. Ten of these sixteen bits contain the address of the unit being called. The system therefore has a capacity of 1000 remote units. The other six bits determine the operation to be performed by the unit addressed. These six bits can also be used to transmit monitor data back to the central station with less than one percent quantization error.

DOWN MINE RECEIVERS

A block diagram for the down mine receivers is shown in figure 1. To conserve power, only the receiver preamplifier, demodulator and a wake-up circuit are connected to the battery during standby. The first part of a signal transmission is a wake-up signal, a sequence of alternating "zeros" and "ones". This is detected by the wake-up circuit which then energizes the rest of the receiver.

The first byte of the signal enters the receiver portion of the Universal Asynchronous Receiver/Transmitter (UART) from the demodulator in serial form and leave as parallel data to the address detection and operation decoding circuits. The first byte is also fed to the transmitter section of the UART through an eight bit latch. When the second byte of the signal is fed from the UART to the address detection circuit, this circuit compares the received address with the stored addresses of display discs or other equipment controlled by the down mine receiver. If there is a match, the operation called for takes place and an echo, consisting of the first byte, already in the transmitter section of the UART, and the second byte, held in the eight bit latch, is transmitted back along the pager phone line. When the called for operation includes transmission of environmental monitor data, this data is supplied to the transmitter section of the UART and becomes part of the echo.

The down mine receivers will be housed in a standard NEMA hinged cover enclosure. The display discs will be mounted in a Lexan tube fastened to the enclosure cover by cast end caps. This package design, shown in figure 2, is economical for small lot fabrication.

CENTRAL CONTROL UNIT

The control panel for the central control unit is shown in figure 3. The panel contains a keyboard for numeric entries, three four digit displays and their associated switches and controls.

The "Present Page" display shows the address and operation (Post, Test or Clear) of the page being entered. If a mistake is made when entering a page, the "Erase Entry" pushbutton clears the display and the entry so that the proper entry may be made.

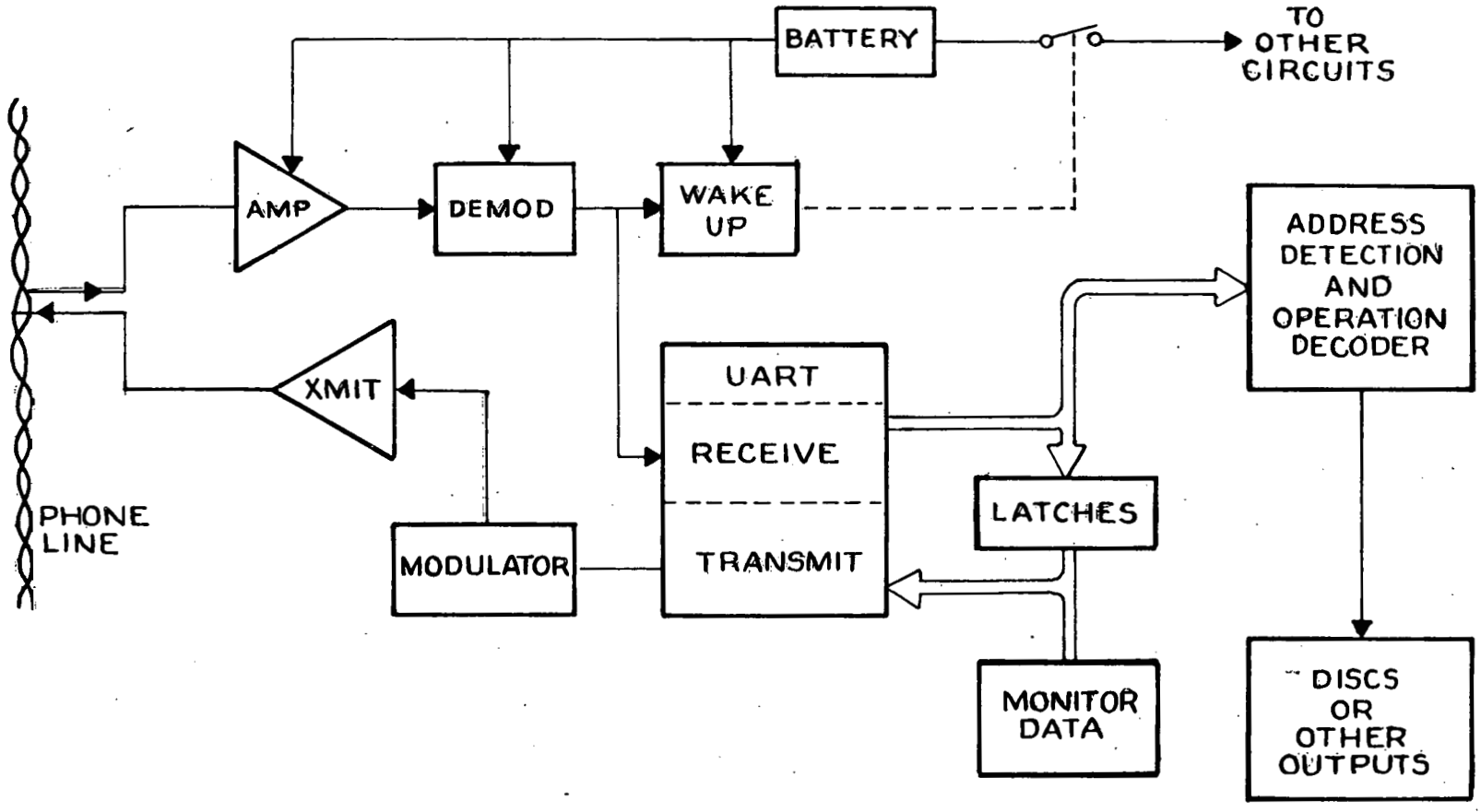
The "Last Page" display shows the address and operation of the last page entered. This helps the dispatcher keep his place when entering a long list of pages.

The "Lists" display shows the contents of the list of pages selected by the switch in the upper right hand corner of the panel. These lists are: "Current Pages" - the pages most recently posted, "Old Pages" - the pages unanswered or not cleared after some set length of time, and "Echo Errors" - the addresses which did not respond with the proper echo when called. The "Lists" display may be made to scan the list at a rate of

about two seconds per address or it may be made to step through the list one address at a time. The list entry on display may be deleted by pushing the "Delete" pushbutton or re-entered as a present call by pushing the "Re-enter" pushbutton.

The "Ready" pilot light indicates that the system has processed the present page and is ready for the next entry. The "Received" pilot light indicates that a valid echo was returned, while the "Error" pilot light indicated that no valid echo was returned.

The heart of the central control unit is a microprocessor, a read only memory for program storage, and a random access memory for list storage and scratch pad use. The other components are input-output devices: the keyboard, numeric displays, the transmitter/receiver, and the associated interface circuits: binary to decimal decoders, and optical isolators. It is advantageous to use commercially available equipment as much as possible for these purposes and the Control Logic, Inc., line of microcomputer modules (M Series) and micrologic modules (C Series), designed for industrial digital control systems will be used. These will be housed in a standard NEMA sloping panel cabinet.



DOWNMINE RECEIVER

Figure 1

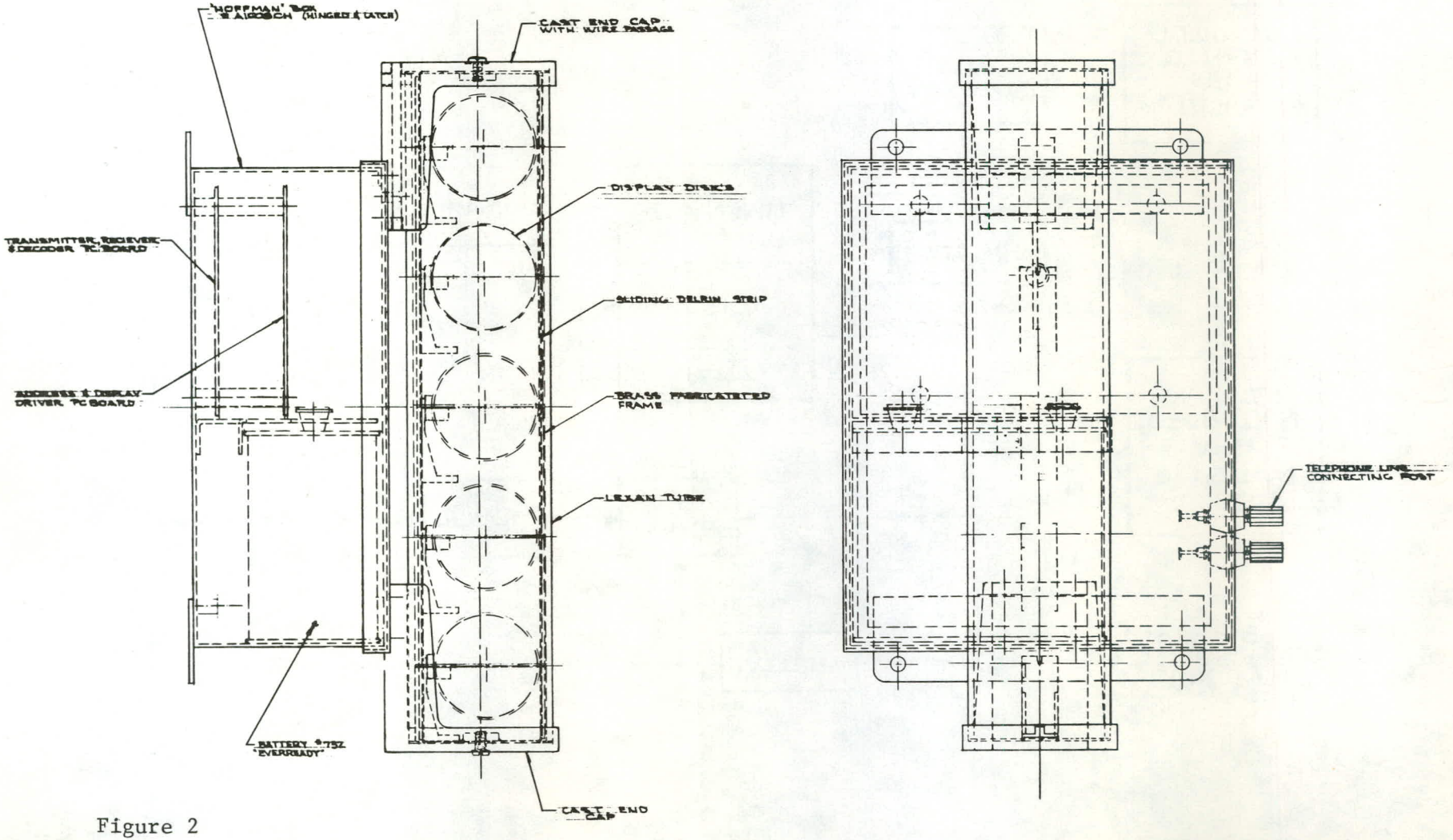
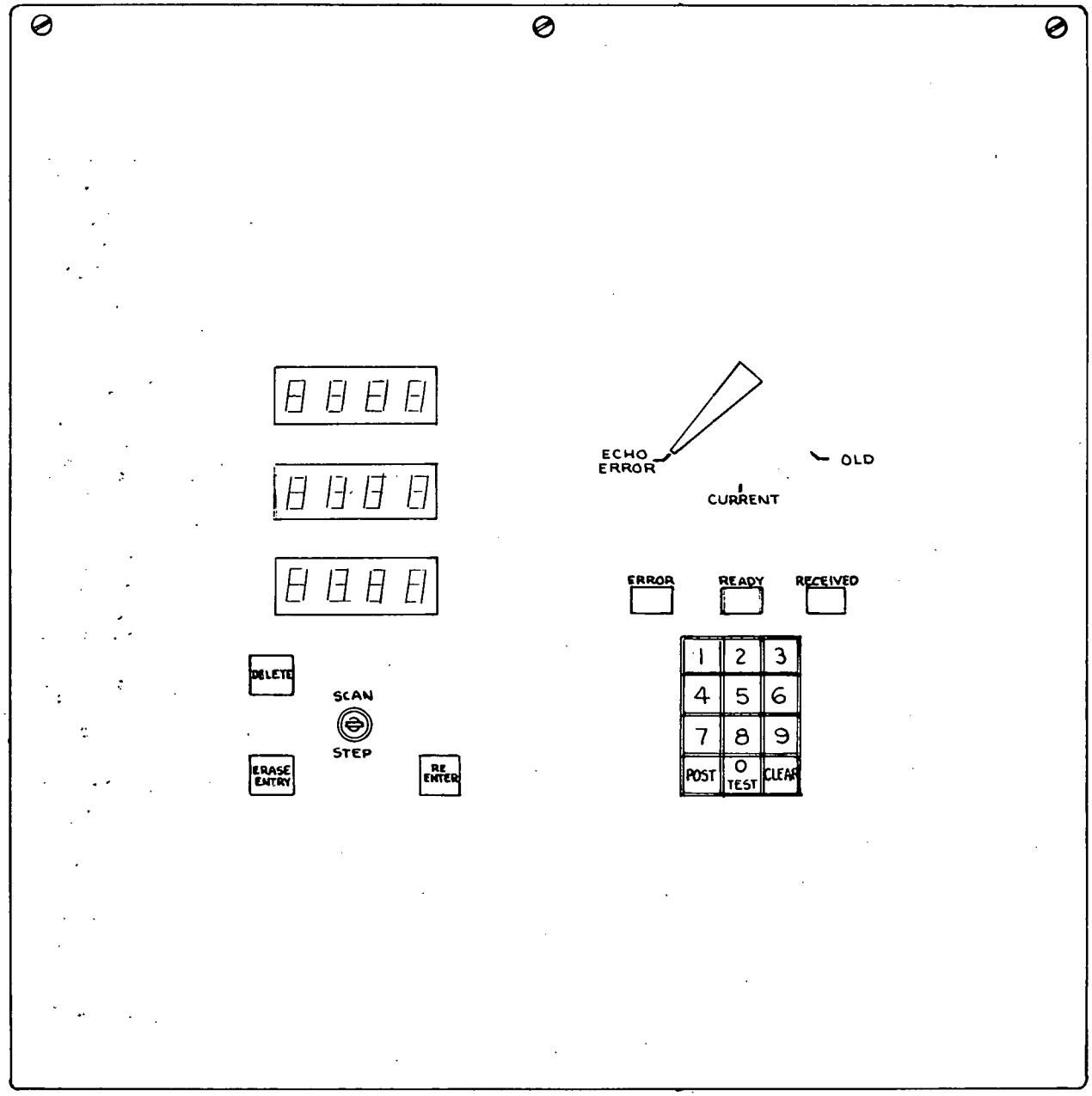

'HOFFMAN' BOX
W AIRDECH (HINGED & LATCH)
CAST END CAP
WITH WIRE PASSAGE
DISPLAY DISCS
TRANSMITTER, RECIEVER
& DECODER PC BOARD
SLIDING DELRIN STRIP
ADDRESS & DISPLAY
DRIVER PC BOARD
BRASS FABRICATED
FRAME
LEXAN TUBE
BATTERY #732
'EVERREADY'
CAST END
CAP
TELEPHONE LINE
CONNECTING POST

Figure 2



Central Control Panel

Figure 3

OVERVIEW OF THE DEPARTMENT OF ENERGY'S PROGRAM ON THE DEVELOPMENT OF AUTOMATED MACHINERY FOR UNDERGROUND MINING

T. J. Fisher and E. R. Palowitch

INTRODUCTION

Last year nearly 700 million tons of anthracite, bituminous, and lignite coal was produced by surface and underground mining to help meet this Nation's energy needs. And in an effort to stretch domestic reserves of oil and gas, the National Energy Plan calls for the doubling of coal demand over the next decade to fuel the ever-increasing needs of the electric power industry and to provide coals for such future markets as coal gasification, coal liquifaction, and MHD.

The Department of Energy's mining research program is directed toward the improvement of current mining systems and the development of new mining systems which will ensure adequate supplies of coal at acceptable costs.[1]* More specifically, tomorrow's coal mining industry must meet the following minimum requirements:

1. It must guarantee the health and safety of the worker;
2. It must minimize adverse environmental impacts;
3. It must maximize recovery of coal resources; and
4. It must remain competitive with other energy sources.

Systems for mining coal underground are categorized as either full extraction or partial extraction.

I. Full Extraction Systems

Full extraction systems include longwall and shortwall mining methods. Longwall methods are used predominately by all major coal producing countries except the United States and Australia. With retreating longwalls, a rectangular block of coal is developed for extraction by driving a set of 3 or 4 entries 450 to 750 feet apart between main entries 2,500 to 5,000 feet apart. These developed panels then are extracted by a shearer or plow which cuts the coal and loads the broken coal into a chain conveyor which runs the full length of the face. The roof is supported by a series of hydraulic, self-advancing roof supports which provide full support over the cutter-loader and conveyor. After each cut across the face, the conveyor is pushed forward by double acting rams on the supports. Once the conveyor is advanced, the individual roof supports are pulled up to the conveyor; the unsupported roof caves harmlessly into the mined-out void.

Longwall mining accounts only for about 5 percent of the Nation's underground coal production. And although one face did produce over a million tons of coal in 1977 (and more than 12,000 tons in a single day),[2] most longwalls average only about 1,000 tons per two-shift day.[3]

*Numbers refer to items in the list of references at the end of the paper

One of the ways to make longwall mining more attractive to U.S. operators is to improve the efficiency of the cutting and loading subsystem. For example, if the cutting horizons of the cutting drums on a double ranging shearer could be automated, it would: eliminate one operator; eliminate the need for the other operator to accompany the machine across the face; increase the amount of coal recovered; decrease the amount of rock inadvertently taken; and reduce wear-and-tear on the machine. Thus, while complete automation of the total system may never be cost-effective, some level of automation can extend, amplify, and perhaps even sharpen human perception through electronic sensors which can provide additional and more quantitative operating information, relieve the monotony of repetitive operations, and permit monitoring from safer and more comfortable remote locations.

A. National Aeronautics and Space Administration (NASA).

NASA, at their Marshall Space Flight Center in Huntsville, Ala., is under contract with the Department of Energy to automate the longwall mining extraction process.[4] This program consists of defining, testing, and recommending a practical guidance and control subsystem for a bidirectional cutting, double-drum, ranging shearer-loader. At the present time, operation of a double-ranging drum shearer requires two operators: one to range the leading drum at the coal-roof interface, and the other to range the trailing drum at the coal-floor interface. The machine operators must function as both error detectors and controllers; i.e., they must observe the elevations of the cutting drums and operate controls to raise or lower the drums as required to cut at the desired elevations. Because coal is opaque, the operators must rely on the noise vibration, color of cutting, marker bands in the coalbed, and relationship of the present cut to the past cut. The efficiency of this feedback system thus depends largely on the human ability to observe errors and to take corrective action.[5]

Improved control of the cutting-loading functions will increase the amount of coal produced per lift by maximizing coalbed extraction. The amount of non-coal material dilution will decrease by minimizing the amount of roof and floor material that is cut and loaded unintentionally with the run-of-mine coal. The machine operators will be upgraded from "hands on" operators to machine supervisors. While automating the cutting-loading function may not be considered cost-effective in thick, flat-lying, easy-to-mine coalbeds, it becomes increasingly attractive in thinner and more difficult to mine coalbeds. Ultimately, to achieve high production safely in thin seams, automated cutter-loaders may be mandatory.

Automatic control for the longwall extraction process will consist of several basic functions. One of these functions is the vertical control system. It will adjust the vertical positions of the shearer drums to insure cutting of the coal from the seam at a nominal specified distance from the coal/noncoal boundary in accordance with specified error criteria. The use of human operators will be minimized, consistent with good practice, in the actual control of the machine, however, human supervision of the face activity will be used. Human operators

will figure most prominently at the turnarounds at the end of each pass and in any unusual face conditions, e.g., proceeding through faults, etc. Similar modes of operation apply to the yaw and roll controls which are required to maintain the coal face normal to both the entries and the seam boundries. Techniques are being investigated that will automatically determine the alignment of the face and correct the misalignment, while automatically advancing the face conveyor and roof supports with each succeeding pass.

A series of operational tests were performed on a longwall shearer located at the DOE's Bruceton mock longwall facility by NASA. The purpose of these tests was to determine the transfer function and operational characteristics of the shearer's hydraulic control system. Hydraulic pressure transducers, temperature probes, accelerometers, rate gyros, and displacement transducers were used to instrument the machine. Two 11-channel tape recorders were used to record the raw analog data. The data obtained were for an unloaded condition since the drum was not cutting coal. Input commands to the system, both sinusoidal and pulse, were used to drive the boom cylinder control valves. The instrumented outputs were boom position, rate of change, and acceleration. These characteristics were used to generate a simulation model of the longwall shearer which subsequently was used to develop a closed-loop vertical control system. The simulation model developed is an end-to-end model which incorporates the component characteristics but does not include a model of the components. Because the data obtained were from an unloaded machine, engineering judgment was used to put in additional anticipated hydraulic flow characteristics which the equipment will have when it is in the process of cutting coal.

An assessment was made of the performance of the longwall shearer with the hydraulic system model incorporated into the digital simulation. Mine roof characteristics obtained from a survey of a number of longwall operations by Arthur D. Little Co., Inc., and simulated machine speeds of 20 fpm and 50 fpm were used. The cutting strategy was to leave four inches of coal on the roof and to take all of the coal to the floor. The shearer control system was based on 2 sensitized picks, 180 degress apart, on each drum, a last cut follower with a ± 2-inch limit, and a nucleonic coal interface detector. The computer simulation generated shearer cutting profiles for the aforementioned case with the result that neither the inclusion of the hydraulic system model nor the increased chassis speed significantly degraded the machine's performance. This is probably an accurate comparative assessment of the effect of the hydraulic model and chassis speed; however, but with only one mine roof profile. Other profiles, if significantly different, would yield significantly different results. Future work will require modeling of different mine roof profiles, further refinement of the simulation model, by assessing the effects of coal cutting loads, and the modeling of other types of coal interface detectors.

In order to perform vertical control subsystem integration tests, NASA has constructed a mechanical simulation of the shearer-loader arm

dynamics. This facility has the capability for testing and checking out various vertical control subsystems prior to installation on the shearer-loader at DOE's mock longwall aboveground test facility at Bruceton, Pa.

The simulator includes a mass analogous to the mass of the shearer drum and cowl and a supporting arm structure for this mass. The support arm pivots on bearings which are identical with the bearings used on the shearer boom. The hydraulic actuator, control valve, needle valve, and check valve are also commercial shearer components. And the hydraulic interconnect lines are of the same size and length as the lines found on the shearer.

The test program will use a NASA-developed Dynamic Simulator which is capable of carrying a wide variety of experimental CID's over a test bed of coal or simulated coal. The CID output signal is fed to an HP9825 controller which evaluates the signal and generates a boom position command which is fed to the servovalve of the Shearer Arm Simulator. The position command response from the boom position command is returned to the HP9825 and evaluated and a new position command is generated.

The design, fabrication, and mounting of the simulator is complete and the assembly of the hydraulic system is in progress. When completed, this facility will permit the dynamic simulation of various CID's and vertical control subsystem arrangements operating over ranges of coalbed and geologic parameters.

B. Bendix Corporation Study.

The Bendix Corporation (Energy, Environment and Technology Office) under contract to NASA is performing a system definition study for a prototype automated longwall guidance and control system.[6] As a basis for this work Bendix is utilizing the results of NASA's earlier work on coal interface detectors, control system concepts, and mathematical simulations. The objective of this work is to conduct an engineering analysis to define the guidance and control subsystems needed to automate a longwall mining system. The three major subsystems are: the Vertical Control Subsystem (VCS) which positions the elevation of the shearer drums, the Roll Control Subsystem (RCS) which controls the perpendicularity of the cutting drums with the coalbed, and the Face Advance Subsystem (FAS) which controls the alignment of the shearer with respect to the coal face and fixes the depth of cut. The subsystems are treated separately and may be implemented separately; however, the concepts developed must function together as a total system to be monitored and controlled, and to be compatible with other operational requirements. An analysis is also being made to assess and determine the most desirable system concepts in terms of technical feasibility, practicality of operation, relative potential in increasing production rate and efficiency, personnel health and safety, and for reducing mining costs. Based upon the requirements and concepts aforementioned complete preliminary designs and development specifications for the VCS, RCS, and FAS are to be made along with a definition of any changes to the existing longwall shearer or other equipment necessary to interface or accommodate the Guidance and Control System.

In order to develop the optimum control system, Bendix is using advanced mathematical modeling and computer simulations to evaluate various designs. One of the more important parameters used to evaluate the control system is an analysis of how it responds to various roof and floor interface profiles. Data on mine roof profiles on six longwall faces obtained from a Bureau of Mines contract to Arthur D. Little was digitized and is being used as input. Of particular interest is an algorithm Bendix has devised that will generate a coal/shale interface profile. These profiles are then used to simulate the results achieved after repeated passes of the shearer across the longwall face as mining proceeds.

II. Partial Extraction Systems

Partial extraction (room-and-pillar) systems include those underground mining methods which develop a coalbed by driving three to nine entries in series leaving pillars of coal standing to support the workings and to provide ingress and egress, ventilation, haulage, etc. Once the area is fully developed (and if geologic conditions permit), the pillars are removed on retreat.

Entry development and pillar recovery are accomplished ordinarily by either of two methods: (1) continuous miners which cut the coal directly from the coalbed and load it into shuttle cars, and (2) loading machines which load coal (extracted by undercuting, drilling, and blasting the coalbed) into shuttle cars. In both methods, the geometry of the workings is similar, the shuttle cars normally discharge onto a conveyor belt and the roof is supported by roof bolts.

In contemporary room-and-pillar mining, face-to-face tram time, intermittent haulage, and roof bolting delays are major factors that limit coal production. Continuous miners seldom mine "continuously" due to safety regulations which limit penetration into solid coal to about 20 feet when the machine must be trammed out and permanent roof supports installed. In addition to the place changing, decision making to determine correct heading, location of the roof and floor interfaces of the seam, the relationship to other machines, and other factors all reduce production significantly lower than the machine's potential. In order to more fully utilize these machines, DOE is conducting research in the following areas: the development of a reliable, cost-effective system to work in conjunction with the continuous miner system to provide timely support of the roof as the mining operation proceeds; the development of a simple, reliable, automated high-speed bolting system that can drill holes and place roof bolts without the need for an operator present in the immediate area; and the development of reliable, cost-effective control systems for continuous miners that will eliminate the delays due to human decision making.[1]

A. Automated Extraction System (AES) for a Drum Type Continuous Miner.

A DOE contract was let to the National Mine Service Company for the development of an automated extraction system.[7] The basic machine, a

rotary drum continuous miner similiar to the Marietta Miner with a hydraulically actuated roof support system, is positioned between two chock-type roof supports and a drill platform. Four manually operated roof drills are mounted in front of the operator's platform. Two operators are required--one for the left two drills and one for the right pair; a third operator is required for the operation of the miner. The cutting drum sump operation is accomplished by hydraulic cylinders, rather than by tramming; mining and roof bolting occur simultaneously. The AES machine incorporates ventilation fans located in the roof support structure which draw air from the face and exhaust it out the back into flexible, extensible tubing.

The AES features an automatic programmed cutting cycle and also an automated temporary roof-support system. The operator initially unlocks the roof support system and makes any necessary machine position corrections. He then raises and locks the outer roof supports and initiates the automatic cutting cycle, which: unlocks the inner roof support, sumps, locks the inner roof support, shears, unlocks the inner roof support, reverse trams and cuts the cusp, raises the drum to the roof, trams back to face, sumps, locks the inner roof support, shears, unlocks the inner roof support, backs up and cuts the cusp, raises the drum to the roof, trams back to face, locks the inner roof support, waits for operator's consent, unlocks and moves the outer support forward, and locks the outer support. The above cutting cycle will continue to repeat as long as the operator continues to reinitiate it. The operator has full override capability during the automatic cycle and can make adjustments in the sump and shear positions while the machine is operating. The two drill operators and the miner operator each have consent buttons, which must be depressed before the outer roof support structure will advance itself. This is necessary so as to allow time for the completion of the drilling operation and to prevent unexpected machine moves. A remote control pendant is provided for tramming between workplaces.

The automatic control cycle is programmed by using a rotary drum programmer. The program is stored by positioning tabs and, as the drum rotates, the tabs activate individual snap action circuit switches. This approach was taken because of its ruggedness and simplicity of design. Program modifications are possible by altering the position or sequence of the tabs. The cutter boom position control uses a potentiometer mounted at the boom pivot to sense the position of the boom relative to the gathering head. A Jordan amplifier compares the operator boom control preset potentiometer with the boom-sense potentiometer, and the amplifier output is used to control a solenoid valve operating the boom jack. This type of control is very rugged and simple in design and yet provides infinite positioning of the cutter boom. The depth of the sump is measured by a shaft encoder mounted on the tram drive gearbox, which counts the number of pulses as the tracks move. The hydraulic solenoid valves are intrinsically safe so that permissible enclosures are not required. It is possible to actuate the solenoid valves manually in case of an electrical failure.

After the mechanical aspects of the machine are debugged and proven, it is planned to replace the drum programmer with a computer based control system.

B. Automated Miner/Bolters

To improve safety and production, contracts have been awarded to Joy Manufacturing Company and Jeffrey Mining Machine Company for the development of an automated bolter in conjunction with a remotely controlled continuous miner.

1. Joy Miner-Bolter Machine[8]

The Joy system consists of a single-pass, 16-foot-wide, ripper type umbilical cord remote-controlled miner. It will be equipped with four automated roof drills, capable of drilling and installing resin-grouted bolts on 4-foot centers simultaneously with coal extraction. The drills are located over the track frames and are also remotely controlled via electrical cable. Operator's protective cabs are located on each side of the conveyor at the back of the main frame. Other components of the machine are similar to those used on the Joy 12CM ripper miner.

The heart of the bolter control system is an RCA 1802 Cos/Mos microprocessor, which operates with a stored program and processes all electrical functions. Each function is controlled by a series of steps in the microprocessor (instructions) which are stored as a series of bytes in a memory. As each instruction is executed, another is called from memory and executed in order. The resultant control is rather simple electrically and quite flexible since a great variety of combinations exist with simple changes in instructions and no changes in wiring. The instructions are revised by removing the integrated circuit memory element and either replacing or reprogramming it. The memory element is called a PROM (Programmable Read Only Memory) and can be erased by exposing it to a concentrated ultraviolet light source for approximately 10 minutes. Under normal operation the PROM is a permanent memory and even exposure to sunlight will not alter it. A PROM programmer available from many manufacturers, is needed to rewrite the altered instruction back into the device. A mineworthy permissible programmer is not yet available so that this step must be done either in fresh air or on the surface. The reprogrammed PROM is then plugged back into the machine, which is then ready for service.

The bolter control system has approximately 120 inputs, which are of two basic types: (1) 120-vac interlock signals from rotation motors, dust collector motors, and traction motor circuits; and (2) switch closures such as sense switches, control switches, and limit travel switches. (Solid state Hall-effect devices are being used for the limit switches.)

The 120-vac signals are isolated from the control circuit and at the same time converted to low-power logic signals by signal-conditioning circuits using optical coupler devices. All inputs are connected to multiplexers (solid state devices which take many parallel inputs and

convert them to a single serial output) near their point of origin to minimize machine wiring complexities. The multiplexer signals are decoded and stored in temporary latches (solid state devices which retain momentary bits of information until they are called for) in the controller case. As required, the microprocessor selects the proper input signals from the latches (usually four at a time, one for each drill) and acts on the status of those signals as instructed by the program. It then selects which output devices are to be energized or deenergized, storing that information in output latches, and then continues doing other functions. All output signals from the microprocessor are optically coupled to a triac or transistor which in turn operates a line starter or solenoid valve, respectively. There are approximately 50 output control circuits from the microprocessor.

Control switches and logic signals are generally arranged in a fail-safe configuration. Additionally, the large memory capability and speed of the microprocessor allow a great deal of cross-checking and interlocking of signals. The emergency stop pushbutton located on the operator's station is connected directly to the microprocessor and output device reset lines, providing immediate response if an emergency stop is required. Signals to and from the operator's remote station are of low-energy, intrinsically safe current level and are totally isolated from the main control wiring via optical couplers and a special split winding power transformer. The continuous miner's manual controls in the operator's cab are available for overriding the solenoid valves in case of a failure. The roof bolter's manual controls will be on the operator's remote control panel and will rely upon the electrical remote control system.

2. Jeffrey Bolter-Transfer Machine (BTM)[9]

This system consists of a bolter-transfer machine (BTM) employed behind a Jeffrey 122 radio-remote-controlled ripper-type continuous miner with a 15.5- to 16-foot wide cutterhead. The BTM is a Caterpillar tracked vehicle, similar to a shuttle car. It will have four completely automatic roof bolters, capable of drilling and installing resin-grouted bolts of lengths up to 1 foot shorter than seam height. Operator cabs will be on either side and to the rear of the machine over the tracks. The BTM operator will occupy the left cab, and the continuous miner operator will occupy the right cab. The BTM will be capable of installing 7 rows of bolts on 4-foot centers in 20 minutes, after which reloading of the magazines will be required. Total bolt installation cycle time, including 4 feet of tramming, is to be less than 3 minutes.

The control system for the four roof bolts will be implemented using an Intel 8080 microprocessor as the main logic element. In the automatic mode, the operator only needs to initiate the operation sequence; all functions will then be performed automatically. The bolting cycle will have a manual override to enable the operator to control any single bolter if any malfunction occurs.

Two modes of manual control are possible, depending upon the nature of the difficulty. Cab-mounted manual controls will send signals through the microprocessor. This mode of override control will be used to overcome small difficulties, which do not require the operator to be physically at the bolter module. Local manual control will be possible by means of control pendants at the drill heads, which will send signals to the solenoid valves directly, bypassing the microprocessor completely. Two local manual control pendants will be incorporated, one on each side of the machine. Only one bolter can be operated at a time by each pendant. Local controls will be provided with a safety interlock to invalidate manual and automatic control signals initiated in the cab. The local control mode will be used for more difficult problems which require that the operator be physically at the bolter module; also this type of control will be able to operate the bolters manually in case of a failure in the electronic control system.

To provide protection against transients and to keep the signals intrinsically safe, optical couplers are used on all inputs and outputs on the microprocessor. Magnetic sensing devices will be used for control limit switches and for machine position inputs. They will indicate the drill position and determine where the supply carousel is indexed on the bolter. Sensed inputs and control output signals will be carried on individual wires to the microprocessor, where each will be coupled through driver circuits and optical couplers. The processor will update itself every 20 milliseconds by scanning all the inputs and giving the appropriate outputs according to the stored program. The operating sequence will be stored on PROMS for ease of system modification and for cost effectiveness.

Design features of these machines should improve the safety and health standards for the operators and crew, allowing them to function more efficiently. Through the utilization of proven high-reliability components and system burn-in, downtime due to electronic failures should be minimized.

C. Automatic Control System for a Boring Type Continuous Miner.

Boring type continuous miners possess an attribute of extreme merit in that the arched entry formed by the cutter rotors, when mining certain seams, produces a roof that is often self-supporting. These machines are used in those geographical areas of southern Illinois and northern West Virginia where proper seam conditions exist to allow operation of face machines under unbolted roof. The borers are permitted to advance a complete breakthrough (up to approximately 100 feet) without permanent roof support. This gives a significant production advantage over drum-type continuous miners which can advance only 20 feet before a place change is required to permit the installation of permanent roof supports. Where roof conditions require roof bolting over the miner, requiring it be trammed out away from the face while roof bolts are being installed, the miner's productive potential is severely impaired. The machine's slow tramming speed and physical size (a full face machine with little side clearance) further aggrevate this problem.

The incorporation of remote control on the borer would allow the operator to remain under supported roof while the borer advanced long distances under self-supporting roof. In addition, automatic control of certain of the borer functions would increase the efficiency of the remote control operation and increase the productivity above that presently being achieved. A study to determine the technical and economic feasibility of remote/automatic control of borers and to develop a conceptual design of a remote/automatic control system was awarded to the Bendix Corporation (Energy, Environment, and Technology Office) and was completed in October 1977.[10] The study consisted of three phases: Phase I included the compiling of information on borer-type miners from manufacturers' literature, observing the borers in operation underground, and interviewing the miner operators; Phase II provided for an analysis of these data and a technical and economic feasibility of remote/automatic control system; and Phase III consisted of a conceptual design of the control system.

The results of this study showed that it is technically and economically feasible to equip a borer-type continuous miner with remote and automatic control and that this improvement should result in increased production. However, if the borer is forced into place changing for roof support its productivity would likely drop to less than 50 percent of its present production and hence it would become uneconomical to continue their use. It was found, for example, that when advancing only 20 feet at a time and then roof bolting, the productivity would be 157 tons/shift compared with 340 tons/shift obtained under present operating conditions.

The operations which would benefit most by automatic control were found to be coal seam tracking, entry alignment, and tram guidance.

D. Sensory Feedback for Remote Control of a Continuous Miner.

With the introduction of remote control for continuous miner operation, there is concern that the lack of feedback information may be degrading operator control of the machine. Most remote control units available today provide for only one-way communication between the operator and the continuous miner. The design philosophy has been that since the machine operator is seldom out of sight of the machine and certainly always within hearing distance of it, no additional information regarding machine performance was necessary. Therefore, the original controls have simply been remoted via an umbilical or radio link so that the control levers can be implemented from a distance of 25 to 50 feet behind the continuous miner. However, it has been noted that some operators feel uncomfortable being physically isolated from the machine. Operators have been observed walking along beside or behind the machine resting the remote control box on the machine frame to regain a degree of intimacy with the continuous miner. Some operators state they no longer have a "feel" for the machine. Other continuous miner operators disagree and maintain that they can do as well remotely as they can on board the machine. There is no doubt, however, that some sensory feedback formerly available to the operator is lost with the present method of remote control.

The Bendix Corporation (Energy, Environment, and Technology Office) under contract to the Department of Energy is determining the feasibility of applying sensory feedback techniques to remote control continuous mining machines.[11] By feeding back certain machine functions to the remote control station, it is anticipated that the remote machine operator will be capable of achieving more effective performance.

As part of this program Bendix is investigating and defining the necessary machine parameters to be measured, and determining the most efficient and cost effective method of providing this information to the operator. A prototype system is being designed, built, installed on an existing continuous miner, and soon will be tested underground to determine its effectiveness.

As a first step in the program, six sections of three mines were visited. The same section was visited on multiple shifts so that the greatest number of machine operators could be observed and interviewed. A total of 14 remote control operators were interviewed. Sections visited had seam heights ranging from 4 to 7 feet with entries ranging from 16 to 20 feet wide. Roof varied from poor to good and the bottom from wet to dry and hard. Both development and retreat operations were studied. All sections used either the Jeffrey 120L or 120M continuous miners with umbilical remote control. The machine operators had from 1-1/2 to 28 years of general mining experience, 1 to 17 years of continuous miner operating experience, and from 3 months to 4 years of remote control experience.

Without exception, all of the remote controlled machine operators observed trammed from place to place with the remote control box resting on the machine frame. Their reason appeared to be due to the size and weight of the remote control pendant and its umbilical cable.

When it came to actually extracting the coal however, individual preferences were noted. Of the 14 machine operators observed, 4 performed all mining functions with the remote control pendant placed on the machine frame, 5 performed all mining functions with the remote control pendant off the machine in a truly remote control fashion, and the remainder performed with the remote control pendant both on and off the machine at various times during the mining cycle.

Those machine operators who performed well off the machine in a remote control mode have obviously successfully substituted sight and sound senses for the more immediate machine vibration feedback. On the other hand, those machine operators who still prefer to operate with the remote control pendant resting on the machine frame apparently have not been able to make this adjustment.

The sensory feedback remote control system is described as follows: Data from the sensors (accelerometers, power and current monitors, and microphone) mounted on the machine are processed and summed to generate the sensory feedback signals. These actuator control signals are then multiplexed by the multiplexer/demultiplexer electronics on the miner and transmitted via the umbilical to the operator's remote console. The multiplexer/demultiplexer electronics in the console demultiplexes the

data to operate the sensory feedback actuators. Commands are multiplexed in the remote console, transmitted via the umbilical and demultiplexed in the continuous miner. These signals are used to actuate the hydraulic valves, the shear cylinder valve, and the electrical contactors. The auger shear cylinder control valve will be replaced with a variable flow control valve.

A lightweight, small diameter umbilical cable will be used to connect the remote control panel to the machine. The cable will be contained in a spring loaded takeup reel mounted on the continuous miner to eliminate cable handling problems. Eight conductors will suffice to provide the following functions: signal transmission, dedicated emergency-off, power and electrical ground.

E. Remote Supervisory Control System.

The DOE is encouraging the evolution and development of automated and remotely-controlled continuous-mining machines, with the ultimate goal of developing an automated remotely-controlled mining section. It is realized that during the next 10 years the likelihood of being able to develop and install in mines mining equipment operating on totally new concepts is very small. Therefore, an important feature of many of DOE's current programs is to improve existing equipment and mining techniques. As part of this philosophy, Arthur D. Little has been contracted by the DOE to investigate industrial supervisory control systems in other industries and determine the applicability of these technologies and hardware to automating mining systems.[12] It is important that potential problems with future system integration be identified now so as to allow orderly development of new control system hardware and to integrate foreseeable system requirements in the development of automated face equipment.

The overall purpose of the study was to investigate the feasibility of remote supervisory control of a continuous mining system, and identify the types of mining machines that are prime candidates for automation in terms of an integrated mining system. In addition, the study identified the modifications, new features, and functional control needs of such mining machinery that would be necessary to meet the long-term goals of automated system operation. It is recognized that incremental improvements to mining machinery performance--achieved through advances in functional design, and enhanced by mine-worthy control technology added by stages over a period of many years--will be necessary to reach that goal. In this evolutionary (rather than revolutionary) process of development, each new feature must be incrementally cost effective and be seen as desirable by mine management. Otherwise the eventual goal of a fully automated system under remote supervisory control will not be realized.

The scope of this investigation is limited to room-and-pillar mining using the following types of mining machines: rotary-drum, chain-ripper, or boring-head type. Roof support methods include conventional roof bolts or resin-anchored bolts. Face haulage methods include shuttle cars or continuous haulage systems.

The initial steps of this study were devoted to a survey of mining operations and mining machinery to identify the combinations, or systems that offer the most promise in terms of their adaptability to automation and remote computer control. Concurrent with this investigation, parallel surveys were conducted of existing automation and control techniques used in other industries to explore the possibility of technology transfer. In addition, an investigation of mine's physical and electrical environment was conducted to define the operational constraints upon control equipment used on face machinery. The results of these surveys were then evaluated to determine the mining systems best suited for automation and remote supervisory control. This selected configuration was then further developed in a conceptual design analysis to illustrate the nature of the problems faced in achieving this objective.

As a result of the screening process, systems having the characteristics listed below were selected for the conceptual design configuration:

Machinery:

- full-face continuous mining machine (either rotary-drum or boring-head configuration),
- continuous bolting with temporary roof support (incorporated as part of the mining machine), and
- continuous face haulage system employing bridge conveyors and mobile bridge carriers with a panel belt tailpiece slide; and

Mine Plan:

- 3-, 4-, or 5-entry systems,
- acute cross-cut angles,
- 60- to 105-foot lift length mined with an entry plus cross-cut sequence, and
- single face ventilation system (blowers not required).

Perhaps the most convincing qualitative arguments in support of the decision favoring continuous haulage over shuttle cars were:

- The elimination of the cable-handling problem for which no automated solution was evident;
- The attractiveness of continuous haulage linkages as a potential source of mechanical guidance data; and
- Continuous haulage more closely resembles a "single-mechanism ideal" than shuttle cars.

Not all coal seams in the United States will be amenable to this extraction system. However, in those instances where it can be employed, system performance is expected to exceed the performance of current systems in terms of both cost per ton and total tons produced.

The difficulties in automating underground room-and-pillar coal mining are considerable. Most of the technological problems result from lack of headroom and the hostile mine environment. Because of this hostile environment, there are potential benefits to be realized through the use of automated underground mining methods, both in increased production and improved worker safety and comfort. In the analysis that preceded the conceptual design phase, it is estimated a 77 percent increase in coal production per section, with a 53 percent reduction in section labor cost per ton, could be achieved through automating the system described. While these economic benefits alone are quite significant there are additional indirect benefits that cannot readily be quantified. There surely would be an improvement in health and safety conditions from eliminating the need for face workers in the vicinity of the automated mining system while it was mining or tramming.

Automated machinery is protected by the control system from exceeding design working limits, so that longer machinery life between rebuild cycles should be expected. In addition, the performance of an automated section would be less sensitive to individual and group skills. An inexperienced operator still could be relatively efficient; and section performance would be less sensitive to absenteeism. Although the ecomonic benefits above have been projected from present coal mine work rules and habits, it is not unreasonable to expect that the use of automated mining techniques could lead to revised work rules, reflecting the less hazardous and arduous duties of the section workers that would result in a higher utilization time per shift and hence higher production.

CONCLUSION

The Department of Energy is engaged in a number of development projects, only a few of which have been addressed in this paper, to improve the control of underground mining equipment. Some of these have to do with improving remote control and have near term industry feedback; others involving machine automation will not be available to the mining industry for some time. Mine-worthy control technology will likely become available and be adopted by industry in stages over a period of many years as they become incrementally cost effective and as they are deemed desirable by mine management.

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COAL INTERFACE SENSORS FOR AUTOMATED MINING MACHINES

by

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ABSTRACT

There are several underground and surface coal extraction machines which could be highly automated if adequate sensors were available. The primary sensors required to keep automated mining machines working within the coal seam are coal interface detectors (CID). A brief review of the Department of Energy's (DOE) efforts to develop both depth measuring and surface recognition coal sensors is presented. The discussion highlights work on nucleonic, EM radar, and machine vibration type detectors.

INTRODUCTION

The goal of DOE's Advancing Mining Technology coal program is to develop techniques and hardware that will increase production and recovery of coal. Subprograms under this program include:

- Automated Longwall Mining Systems
- Automated Remote Controlled Continuous Miner
- Surface Mining Equipment Development

Each of these subprograms call for the development of highly mechanized machines; in each case a control system with appropriate sensors will be developed to control the vertical position of the cutting elements. For example in the case of double-ranging drum longwall shearer the control system will automatically vary the elevation of the cutting drums as the shearer travels across the face; on room-and-pillar continuous miners the control system will operate in conjunction with the machine's programmed cutting cycle to insure repeatable roof/floor levels. In surface auger mining the control system will permit a machine to recover coal from seams whose thickness almost equal cutterhead diameters (18 to 36 inches) and to depths up to 500-foot into the highwall. Today's auger machines with their small-diameter cutterheads seldom exceed penetration depths over 150 feet because of their inability to keep the cutterhead within the seam and on a proper heading.

A machine with controlled cutting in the vertical plane is capable of mining all or any desired part of a coal seam. Advantages of such operation includes higher coalbed extraction, reduced dilution by minimizing roof/floor material taken, potential increase of coal cutting rates, reduced maintenance and potential for removing the operator from hazardous areas.

Potential economic benefits resulting from just one aspect of controlled cutting is illustrated by the following example. During a visit to a longwall mine, it was observed that from 8 to 12 inches of coal was often left unmined for protection of the roof while only approximately half that amount was required; however, the operator was unwilling to risk a closer cut. It is believed that with a vertical control system, on the average from 4 to 6 inches of additional coal from each pass could be recovered. This translates into an increased coal recovery of approximately 50,000 tons for a typical longwall panel 500 ft wide by 5,000 ft long. Depending upon the coal's market value, the additional economic return for such a panel could be as much as 1.5 million dollars.

Many sensor concepts have been investigated to date. In some instances existing devices that could be adapted to the coal thickness measuring problem of a particular mining method were evaluated. For example a depth measuring optical reflectance sensor that mechanically scans vertically across the coal/roof interface was investigated for use on an automated augering system. The concept requires that a kerf, typically 3/4-inch wide, four to five inches deep, be cut with a circular saw through roof coal and slightly into the roof itself. Although a continuous roof slot may not be objectionable in auger holes, the same cannot be said for room-and-pillar mining. Sometimes new or different concepts were studied without regard to specific machines - natural gamma radiation, electron magnetic resonance. Other times only analytical studies were performed such as the ones conducted by the University of Colorado on the feasibility of using resonant loop antennas or the mutual coupling of small co-planar loops for coal sensing applications. Still further, at times the effort involved breadboard construction, laboratory demonstrations and or in-mine measurements on roof/floor coal in-situ. Table 1 lists relevant ongoing or completed projects by sensor concept; because of the large number of concepts investigated to date, each is not discussed individually. Rather, a few of the more promising CIDs currently being developed are highlighted along with two recently studied novel concepts.

DISCUSSION

The key element in any controlled coal cutting system is the coal interface detector (CID). Its function is to locate the cutting element's position relative to the coal seam boundaries. Ideally it would operate at point of cutting, not require contact with the roof or floor, generate reliable real time measurements continuously, have at least a 1 to 12 inch range with $\pm$ 10% accuracy, 0.5-inch resolution, not require frequent or elaborate recalibration, and operate on both roof and floor so as to allow bidirectional cutting.

In order to locate the boundary between a coal seam and its adjacent strata, some measurable contrast must exist. Thus for discrimination we must be able to detect the difference in some material property. These properties may be grouped conveniently into three categories: basic, e.g., density; mechanical, e.g. compressive strength or elasticity; and non-mechanical, e.g. electrical, magnetic, thermal, or optical.

The roof and floor enclosing most U.S. coalbeds consist of shales, sandstones, limestones and clays, which may vary widely in thickness and in physical, chemical, and petrographic properties. These changes often are subtle and the transition from one rock type to another may be more a gradation than a discernable change thus making discrimination by a CID difficult. The coalbeds often contain such inclusions as streaks, bands, and partings of non-coal material and such concretions as sulfur balls and coal balls.

Because of the various geologic formations present, plus the different mixes possible, a wide range of boundaries in terms of physical properties can be expected - from discrete to diffuse. Thus boundary detection could very well depend upon the physical property being measured. For example, mechanical properties of coal can approach those of shale becoming harder and less friable closer to the interface; a sensitized drill sensor would not distinguish between them. Yet, the same boundary would be detected by an optical reflectance sensor because of definite change in color.

Many different concepts for CID application were explored and the development of several has progressed to experimental hardware. These CID's can be conveniently catagorized as: 1) depth measuring and, 2) surface recognition. Depth measuring devices measure the amount of coal between the sensor and seam boundary; examples are backscatter gauges natural radiation detectors and EM radar. Surface detectors classify material remaining after extraction or material prior to extraction as coal or non-coal. Examples of this type include the machine vibration sensor, the sensitized pick, and impact penetrometers.

DEPTH MEASURING SENSORS

Radar

Radar CID's are based on the principal that different coal/shale electrical characteristics will cause a reflection of EM waves at their interface. Thus by measuring the transit time required for waves to travel to and from coal boundaries and knowing the propagation velocity of air and coal materials comprising the total travel path, one can determine distance to an interface. Resolution and penetration range (minimum and maximum coal thickness that can be measured) are dependent on operating frequency. Resolution of radar systems is basically an inverse function of bandwidth; the wider the bandwidth the smaller the distance between two resolved targets. High bandwidth can be achieved in several ways, each resulting in a different class of radar. Two radars being applied to the CID problem are pulse and frequency modulated continuous wave.

Experiments using both pulse and CW radars have been performed by NASA, Ensco, Southwest Research Institute, and Foster-Miller Associates. NASA concentrated mainly on FM-CW; they built/tested several different systems in the lab and Bruceton Mine. One unit operated from 2.2 to 4.3 GHZ, used identical horn antennas for transmitting and receiving, and

had a measuring range of 2 to 5.5 inches. Another model, designed for 2.75 to 11.8 inches of coal, operated from 1.7 to 4.4 GHZ. It used a single 13 x 9.5-inch operature horn antenna and 100 HZ triangular wave modulation along with submodulation to add extra spectrum lines to the video output. National Bureau of Standards (NBS) made roof coal measurements in three different mines with a 1-2 GHZ FM-CW system. They used two broadband rectangular aperature horns with flush mounted lucite lenses to reduce sidelobes. Southwest Research Institute designed an experimental unit that operated from 2-6 GHZ and used a matched pair of dual-polarized quad-ridged horn antennas; total packaging volume was 150 cu in. Tests performed included laboratory measurements and field trials.

Ensco, R. A. Hanson, and Foster-Miller Associates worked with pulse radar; typically 1.6 and 1.0 nanosecound radiated pulses with 625 and 910 MHZ center frequencies respectively. Pulse data was obtained from underground mines and auger holes; extensive signal processing and data analysis was performed.

In general results indicated that 1) radars are feasible for coal interface detection, 2) they have many attributes of the ideal sensor, and 3) continued development is warranted. However, they are not without their problems. Short target range, severe signal attenuation, multiple target returns and diffuse boundaries all make return signal interpretation difficult and at times unreliable. Various ongoing projects are directed toward solving these problems.

NASA is striving to improve range, reliability, and target resolution in FM-CW radar performance. Plans call for preparing and providing a 2.2 to 4.3 GHZ radar for inclusion in the Mock Longwall Vertical Control System test program and the building of an advanced model that operates from 0.7 to 8 GHZ with a theoretical measuring range of 1 to 12 inches in coals having 3 dB/inch attenuation. Emphasis is being put on resolving the masking effect of the initial return (air/coal) upon the second return (coal/shale boundary). Several different processing techniques for enhancing detection and discrimination of weak interface signals will be tried; two techniques offering immediate improvements are the front surface local oscillator concept (FSLO) and the use of weighting functions. With FSLO, the initial or front surface signal is used as the reference signal instead of a portion of the transmitted signal as is the case with the more conventional system. Hence, the difference signal is directly proportional to coal thickness and, as the front surface is no longer detected, the masking effect is reduced considerably. By applying appropriate weighting functions to the receiver difference frequency signals, sidelobe reflections can be reduced so that the signal-to-noise ratio of interface targets can be improved significantly.

NBS is under contract to solve the problem of variations in surface dielectric constant encountered in their earlier work (contract No. J0155124). They are to design, fabricate and field test an FM-CW radar that will automatically correct for changes in surface dielectric constant. At this point in time they are performing in-mine evaluation tests.

At the present state of development most of the analysis of both pulse and CW radars is performed by a skilled operator. In most cases the operator will select the interface correctly. However, for radar to be a useful tool for interface tracking in automated extraction machines, algorithms must be developed which can generate an accurate servo error signal.

Ensco Inc. has made significant advances toward this goal. Working with in-mine pulse signal returns (350 MHZ center frequency, 2-1/2 octave pulse bandwidth) they were able to process the data in such a way that a preselected coal/shale interface was automatically tracked along a 50-foot long entry. The processing algorithm set used included: deconvolution to partially remove internal hardware reverberations; correlation to get the best estimate of interface location; integration or averaging of returns over 4-foot lengths to smooth instantaneous mine roof undulations; and an adaptive process to update the reference return. We are now extending this work to demonstrate the generality and accuracy of the processing set developed by Ensco. Under this contract pulse radar (900 MHZ center frequency, 2-1/2 octave pulse bandwidth) measurements will be made in several mines on roof coal; the algorithm set will then be applied to the raw data. Radar measurements will be compared with borescope observations to measure radar accuracy. If successful, the project will result in advanced prototype hardware.

Nucleonics

Several different concepts using nuclear or atomic emissions capable of penetrating coal layers were investigated by NASA. Four approaches chosen for study were:

1. Emission of prompt gamma due to the interaction of neutrons with the nucleii of coal or non-coal materials.

2. Backscattered thermalized neutrons.

3. Backscatter of gamma or X-radiation.

4. Measuring naturally occuring radiation emitted from rock strata.

The techniques employing prompt gamma and slow neutrons were judged unsuitable for mine use because both were relatively insensitive for interface detection and could cause a residual radiation problem. The backscattered gamma ray and natural gamma radiation concepts were shown to have definite coal thickness measuring capabilities.

Gamma Backscatter

The operation of a backscatter gauge for coal depth measuring depends on the scattering and attenuation coefficients of coal and shale materials. These coefficients in turn are dependent upon the materials' atomic number. When a narrow beam of gamma radiation from a source such as Cesium-137 is directed into coal it is subjected to Compton scattering resulting in some incident rays getting scattered back. The amount of radiation backscattered is inversely proportional to the scattering medium's density.

Several laboratory prototypes were tested having coal penetration ranges from 4 to 10 inches, weighing from 40 to 140 pounds, and source to detector separations ranging from 7 to 20 inches. Gamma sources tried included Cesium-137 and Americium-241. Two major sources of measurement error were encountered; (1) air gaps between the device and coal surface, and (2) non-coal impurities within the seam. In addition there was concern over the potential safety hazard, even though minimal, posed by the radioactive source.

Attempts to solve both problems have met with mixed success. The effects of different source strength and source/receiver angular orientation were studied. NASA found, in general, that accuracy and usable range could be increased by canting the source and receiver toward one another. No increase in penetration was noted when source strength was increased - only an increase in number of counts received. The net result was that for coal thicknesses of interest, typically 0 to 10 inches, the source is of sufficient size to require licensing by the Nuclear Regulatory Commission.

As nothing can be done about dirt bands occuring naturally in coal and inasmuch as air gaps contribute the largest error source, NASA concentrated on the latter problem. They designed and fabricated a two-point floating-head sensor capable of operating from 0 to 8-inch thickness range with $\pm$ 1/2-inch accuracy. This design allows the source and the detector to move independently of each other thus minimizing air gaps. It is currently undergoing tests at NASA's Marshall Space Flight Center along with a National Coal Board developed type 709 nucleonic coal sensor.

Natural Gamma Radiation

Prospects for the development of a practicable natural gamma radiation sensor looks good. It is particularly attractive because no internal radiation source is required and it does not require roof contact. The device uses the level of gamma emissions from naturally occuring radioactive potassium in shale as a means of measuring coal thickness. Since coal acts as an absorber, a gamma detector (essentially a sodium iodide crystal, photomultiplier, and associated electronics) placed near the mine roof will produce a signal which is a function of coal thickness.

Using laboratory instruments NASA made natural background radiation measurements on roof shale strata covered with various thicknesses of coal in the Bureau of Mines Safety Research Center Mine in Pittsburgh and in a mine in Alabama. Data obtained at the two locations were quite similar indicating that: 1) an air gap up to 6 inches does not affect the level of radiation from rock if the detector field of view is collimated; 2) low count rates resulted in coal depth measurement errors on the order of $\pm$ 1-inch; and 3) coal thicknesses up to 6 inches are possible.

The question of intensity and distribution of natural radioactive materials in shales enclosing coalbeds is being pursued. A U.S. Geological Survey study conducted for the AEC to determine the amount of uranium in black marine shale showed that consistent background radiation measurements

could be expected in eastern but not in western coalbeds. To verify these predictions measurements are now being made on coal and shale samples from a number of eastern coal mines. To date 18 samples from 14 mines extending over 5 states have been analyzed; preliminary results indicate generally consistent radiation levels.

To improve sensor accuracy NASA plans to increase count rate by using a 5-inch diameter detector crystal. Their goal is to design, fabricate, and test an experimental natural gamma sensor that can operate from 0 to 6 inches with $\pm$ 0.5-inch accuracy. Detector shielding will be obtained by surrounding the detector with a sodium iodide crystal annulus to provide anticoincidence signals in the scaler circuit.

In addition to NASA's work, an experimental unit designed by the British NCB has been obtained for evaluation under an exchange agreement.

SURFACE RECOGNITION SENSOR

Machine Vibration

The use of machine vibrations as a basis for interface detection has progressed very well. General Electric Company, under contract No. H0155120, developed a prototype system which discriminates between cutting coal and noncoal by detecting differences in the time frequency characteristics of the machine's vibrations. Basically it consists of an accelerometer mounted on the machine's ranging arm to sense shearer vibrations followed by appropriate signature processing.

While a number of interface discriminants were investigated, two were found to be accurate and sensitive. One was a frequency domain technique based on the vibration amplitudes at resonant frequencies; It utilized a narrow band filter to detect the 1750 HZ resonant frequency of interest. This signal was then amplified, converted to an rms value, and averaged over a short time to eliminate instantaneous fluctuations. In order to minimize effects of such machine parameters as drum or haulage speeds, the output signal was normalized by dividing it by a similar signal taken at a different frequency. The voltage level was indicative of the material being cut. One major failing of this technique was that it had to be fine-tuned to each shearer model.

A time domain technique was also identified. This technique is based on the level of amplitude modulations in the vibration signal and is less dependent on the shearer models. The accelerometer signals were first normalized by an automatic gain control (based on amplitudes of the signal peaks averaged over 1 sec) and then bandpass filtered (2 KZ to 4 KHZ). A root-mean-square converter with integral DC-level subtractor yielded the fluctuating level of the filtered signal as a function of time, subsequent demodulation gave the envelope of the RMS signal. Rectification and peak detection then provided a signal proportional to vibration modulations, i.e., coal/rock discriminant. This signal as in the previous case was compared with preset voltage levels corresponding to rock material.

GE has provided DOE with four prototype units based on their time domain design. These units will soon be undergoing field evaluation tests, both in the U.S. and in the U.K. We plan to have a similar unit developed for continuous miners.

NEW CONCEPTS

Electron Magnetic Resonance

One potential alternative to existing depth measuring sensors is a radio frequency resonance absorption technique known as electron magnetic resonance (EMR) or electron spin resonance (ESR). It is being investigated by Southwest Research Institute under a subcontract to NASA.

This technique detects free or unpaired electrons which occur naturally within a material. When the material is situated in an appropriate magnetic field, the electron's dipole moment interacts with the magnetic field to produce a resonance effect that may be detected externally by an RF electromagnetic field. One effect of the EMR response is an increase in the material's absorption of electromagnetic waves at the resonant frequency.

SRI has reported some encouraging results. Laboratory measurements on 133 coal and shale samples from multiple selected locations on each side of and across the interface zone were performed. SRI's findings indicate that the coal/shale interface, as well as the air-coal interface is readily discernable on the basis of the large differences in EMR amplitude characteristics of coal compared with both air and shale.

In addition, an analysis was performed based on the findings of the sample studies, and a conceptual model of a low frequency EMR system considered suitable for coal thickness gauging was developed. Results from this analysis indicate that a sensor with 1/2-inch or less resolution over a thickness range from 1/2-inch to 6 inches is possible. Future work will be directed toward developing a breadboard model of the EMR coal thickness gauge suitable for demonstration and evaluation of measurement range and gauging accuracy.

Annular Slot Antenna

Whereas positive results have been achieved in most areas of CID sensing, one aspect remains elusive - depth measuring at point of cutting. NASA is attempting to resolve this problem by going to a predictive scheme relative to longwall mining whereby previous cut information is used with trailing CID readings.

One EM technique identified by the University of Colorado appears to offer another possibility in this area. The concept deals with a probing method using an annular slot antenna driven by a coaxial line.

In practice the antenna would consist of a rigid coaxial cylindrical transmission line with the open end flush-mounted at right angles onto a large metallic ground plane. They have shown analytically that it can be applied to coal thickness measurements. A feature of this antenna structure is that EM interference from surrounding objects is minimized because of the isolation inherently provided by the ground plane. Furthermore, if it is operated above a frequency of a few MHZ, a small antenna can be used which may allow its installation onto the surface of a rotating drum of a mining machine. Plans are to pursue this concept further.

CONCLUSION

The coal/non-coal interface detection problem is a difficult one. Results of all CID work to date indicates that no single technique investigated can be used as a general solution to the problem. Each technique studied was shown to have limitations in terms of range, resolution, accuracy, and deployment. As a result, the concept of using a suite of sensors which when combined approaches the performance of an ideal sensor has evolved.

Presently it is envisioned that the final CID will consist of one or more coal depth measuring devices complimented by one or more surface detecting devices. This mix will insure the highest likelihood of providing correct information for an automated machine. Specific combinations will depend upon the particular application and take into account local geological conditions, coalbed characteristics, the mining machine used, and extraction strategy. Thus, parallel development of individual devices for both categories (depth measuring, surface recognition) is actively being pursued.

TABLE (1)

Sensor Concept	Investigator*
Electrical Resistivity	1,8,6
Electrical Dielectric	1,8
Magnetic Susceptibility	8
Pulse Radar	6,7,1,5
FM-CW Radar	4,2,1
Microwave Resonance	6,2
Induction	6
Magnetic Spin Resonance	1
Natural Gamma Radiation	1
Gamma Ray Backscattering	8,1
Acoustics	1,8
Mechanical Drilling	8,6,1
Sensitized Pick	6,1
Penetrometer	6,1
Hydraulic Drill	1
Machine Vibration	3
Optically Sensitive Drill	6
Optical Hardband Tracker	6
Optically Sensitive Saw	4
Infrared Transparency	6
Optical Reflectometer	1
Thermally Sensitized Pick	1
Infrared Radiation	1

*Identification of investigator and contract number follows.

	Title	Conducted By	Status
1	The Development of Automated Longwall Shearer (H0155092)	NASA	Current
2	Microwave Measurement of Coal Layer Thickness (S0144086)	National Bureau of Standards	Completed
2	FM-CW Electromagnetic Technique of Measuring Coal Layer Thickness (J0155124)	National Bureau of Standards	Completed
2	Evaluation of an Electromagnetic Coal Thickness Sensor Equipped with In-Situ Dielectric Constant Calibration (ET-77-C-01-8881)	National Bureau of Standards	Current
3	A Vibration Sensor for Horizon Control in Automated Longwall Mining (H0155120)	General Electric Co.	Completed
4	Highwall Mining Equipment Vol I, Vol II (H0262055	Southwest Research Inst.	Current
5	Development of Highwall Mining Equipment Vol II (H0262033)	R.A. Hanson Company	Current
6	Evaluation of Coal Thickness Sensors Vol I, Vol II (H0357000)	Foster-Miller Associates	Completed
7	Mine Roof Stratigraphy Using Electromagnetic Radar Interim Report - Advance Signal Processing of Short Pulse Radar Data to Isolate Coal/Shale Interface (J0366076)	Ensco Inc.	Completed
8	Feasibility of Remote Control and Development of Remote Control Devices: Continuous Mining Machine Seam Interface Sensors (H0111194)	Bendix Research Laboratory	Completed

MICROPROCESSOR-BASED SUPERVISORY CONTROL SYSTEM FOR COAL MINE APPLICATIONS

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INTRODUCTION.

The supervisory control of systems with large remote stations, such as are typical of electric utility applications, may be considered a mature technology at the present time. The technological challenge at the moment would seem to be to extend the benefits of integrated supervisory control to those applications where the remote stations are very small --- stations having perhaps a half-dozen on/off controls, and a few points of analog telemetry. Coal mine requirements are somewhat typical of such applications, where a relatively few factors are monitored or controlled at any given location, yet many such locations are involved. Such factora might include the control of fans, and the monitoring of methane concentration, the presence of carbon monoxide, air flows, pressures, temperatures, and relative humidity.

In the past, remote operation of such systems, where it was attempted at all, has been accomplished by discreet hard wire, or by a pastiche of individual narrow-band tone channels or scanners for the controls, and various types of analog telemetry for the data retrieval. Many of these systems have expanded to the point where the capacity of the available communication circuits is severely taxed. Even less acceptable is the situation to be found at the central control station, where the walls are festooned with all manner of chart recorders, annunciator panels, and row upon row of control escutcheons in great confusion. In one case known to the author, the control room had been so expanded to accomodate the growing number of chart recorders that the operator scanned the distant walls with opera glasses! Such systems are prime candidates for replacement by some form of integrated system capable of providing a compact display of the relavant data on a cathode ray type of visual display unit (VDU), preferably by exception, as is done in larger supervisory control systems.

Recent advances in the technology of integrated circuits, and especially of analog integrated circuits, have done a great deal toward reducing the cost of the remote station equipment to the point where a minimum supervisory remote need cost no more than a couple of analog telemetry channels. The central station, however, has proven less amenable to efforts at minimization, due in large part to the necessity for a complex operating system program and the need for disk or drum storage to contain it.

SYSTEM DESIGN.

In creating the MicroDAQ master station, we determined to eliminate the operating system by providing several microprocessors, each dedicated to a specific function, operating asynchronously. With no operating system, the memory requirement would be only that which is needed for the actual storage of data, plus some storage for the operating program of each microprocessor. These operating programs would be in hard (read only) memory, and thus immune to power failure or transient disturbances. Because semiconductor memory is available in relatively small increments, only as much data storage would be supplied as was required by the particular system being implemented. At current prices of semiconductor memory, it was estimated that sufficient data storage to service 240 standard size remote stations could be provided for the cost of the smallest, fixed-head disk.

To simplify the design of the master station to the greatest possible extent, and to minimize the memory requirement, certain departures from standard practice were taken in the overall system design. For example, the boundary between "station address" and "point address" was made variable on a per-station basis. Thus, a block of master station memory can serve one large remote station or several small remote stations, whereas, with a fixed number of point addresses per station, memory would be wasted on sub-equipped remote stations. Each remote station reports its size, along with its address, as part of the first word of each message. In general, each remote reports, along with the data, all of the information required for the processing of that data. Each word, for example, contains a

data "type" code, as well as the point address, so that this information does not need to be stored at the master station (nor does it have to be programmed in). All telemetry data are transmitted in decimal format, so that conversion prior to display is not required. Scaling and zero offsetting, however, are done at the master station. Obviously, sending information of which the system has previous knowledge reduces transmission efficiency somewhat, but, inasmuch as the system is intended primarily for smaller remotes, the sacrifice of some transmission efficiency to gain hardware simplicity is considered justifiable.

As is customary, all transmitted words (of 31 bits) are protected by cyclic parity codes. The particular code used is a modified BCH, having a Hamming distance of four, and thus capable of detecting all triple errors, as well as synchronization slippages of up to three bits in either direction. No attempt is made at error correction; faulty words are simply rejected. Since the data blocks in this system are normally very short, careful attention was paid to word synchronization. To this end, modulation is by three level FSK, with the third level reserved for word synchronization.

A further simplification of the master station (at slight cost to the remote stations) is in the method of requesting blocks of data. The normal request by the master station is simply "next block". The remote station remembers which block was last transmitted, and sends the next block automatically. This method of operation is made possible by the inclusion of the point address within the data word. To expedite alarm access, and to minimize control response time, telemetry data blocks are limited to 16 words, each word containing one data quantity. A complete alarm poll is conducted between data blocks. The remote stations transmit status or telemetry information as requested, in descending order of address. Thus, the last transmission is always point zero, and, as such, is readily recognized by the master station as marking the end of the transmission. This is particularly helpful in the case of partially filled blocks. Acknowledgement of transmissions from the remote station by the master station takes the form "I have received N words of your last transmission". If the remote sent more than N words, it will repeat the entire transmission during the next

poll.

The capacity of the system is 248 standard stations, distributed on four circuits, with a maximum of 62 stations on any one circuit. A "standard" station has a capacity of 32 control points, which may be either binary or ternary, 32 binary or ternary status points, 16 analog variables, and 16 proportional (set point) controls. Because of the variable number of station address bits which may be employed, any location may be configured as a quarter, half, full, double, quadruple or octuple station, with proportionate capacity.

MASTER STATION OPERATION.

The master station is housed in an operator's desk. Atop the desk are mounted the 19-inch CRT visual display unit, and a slope front turret containing the operator controls. All of the required electronics are housed in the pedastals of the desk. The printer, which is optional, may be located elsewhere nearby. The operator controls are in two groups, one for system operation, and one for controlling the VDU. The system controls consist of a numeric keypad for entering device select numbers and set points, and several common function buttons for such functions as: device select, set point select, on, off, raise, lower, stop, start, cancel, clear and alarm acknowledge. In addition, there is one lighted pushbutton for each remote station in the system. As well as providing the station select function, the station buttons flash red or yellow when a major or minor alarm occurs at the station.

The display controls consist of pushbuttons for calling up the various types of display: system alarm summary, system off-normal summary, station alarm summary, station off-normal summary, station control listing, station telemetry listing, and graphics 1, 2, 3 and 4. A "page" button is al provided for calling up a continuation of pages of any listing which exceeds the capacity of a single display page (48 lines).

The VDU display is in color (7 foreground and 7 background may be specified). Points in alarm are flagged by flashing the value (or status) in a different color. After acknowledgement, the flashing ceases, but the off-normal color persists until the condition

clears. Tabular listings show the full user-assigned name of the device or data quantity, along with the status or value. Graphics pages of the "one line diagram" type may be displayed. A font of 68 user-definable graphic symbols is available for constructing such pages.

On both tabular and graphics pages, an arbitrary three digit number appears adjacent to the listing or symbol. This user-defined "display number" is what the operator enters to select devices for control. This number must be unique to the display, but need not be unique to the system.

The bottom line of all pages of the VDU display is reserved. Any control action by the operator will be printed there, permitting verification before execution. Any alarms occurring at locations in the system not currently on display will also cause an alarm message to appear on the bottom of the screen. Should the operator attempt operation of a device subject to interlocked control, a "blocked" message will appear at the bottom of the screen, if the interlock conditions are not satisfied.

From the above description of the system operation, I believe it may be fairly conceded that most of the functions normally associated with modern, computer-based supervisory master stations have been implemented by the microprocessor array.

MASTER STATION IMPLEMENTATION.

As shown in the block diagram of Figure 1, the backbone of the system is the main memory bus, consisting of 19 address lines, an 8-bit bi-directional data highway, and various control lines. Physically, this bus is the backplane wiring of a standard card bin which is capable of holding 20 cards, approximately 8 x 10 inches each. Into this bin are plugged the several microprocessor cards, to a maximum of ten, and memory cards to a maximum of eight. The main bus provides the means for each of the microprocessors to access the central data store. The central data store contains all the fixed information required to define each point in the external system, as well as the current status, data values and various flags. The content and organization of the storage will be described later.

Microprocessors.

All processors used in the MicroDAQ system are Zilog Model Z-80. This unit is quite similar to the more familiar 8080 machine, and it will, in fact, run software written for the 8080. The Z-80, however, has block transfer instructions not available on the 8080, which are particularly valuable in the shared-bus environment of the MicroDAQ system. The processor is capable of being clocked at 2.5 megahertz, and a 4 megahertz version has been announced. When operated at 2.5 megahertz, the CPU is capable of block transfer of data at the rate of 8.4 microseconds per word. In the present system, because of the double buffering required by the shared bus structure, it was decided to clock the processors at 1 megahertz. This speed has been found adequate for all applications thus far considered, and it permits the use of slower and less expensive memory circuits than would otherwise have been required.

Each of the processors in the system has access to a local memory which contains its operating program, inplemented in PROM, plus a small amount of scratch-pad memory as may be required for data manipulation. In all cases, except that of the display controller, this local memory is located on the same circuit card as the processor itself. Any other devices connected to the processor are regarded as I/O devices. (The Z-80 processors use separate address space for memory and for input/output.)

Contention among the processors for access to the main memory is resolved by polling. A poll counter, implemented in hardware, samples the processors at the rate on one every 4 microseconds. When any processor attempts to access the main bus, it automatically latches itselt into a "wait" state until polled by the counter. It then seizes the bus by outputting a memory address. This action stops the advance of the poller for approximately 25 microseconds. As long as the selected processor continues to output addresses at the rate on one every 21 microseconds, it can retain control of the bus. The "operating system" places no limitation on the size of the block which may be transferred at one time; however, the program for the individual processors have been so written as to limit the block size to four words, except in the case of the line processor,

which is permitted to transfer 8 words in some instances. The worst case delay for any processor attempting access works out to be just under one millisecond, and is normally very much less than that. This delay has not caused problems with any of the programs attempted to date. Since the operating system does not allow the simultaneous access to the bus of more than one processor, direct communication between processors is not possible by that route. When such communication is necessary, small buffer areas are reserved in the main memory wherein one processor may write messages for transfer to another. There is also a link, via the operator's console, between the console controller and the VDU processor. No other communication between processors has been found necessary, indicating that the separation of functions is reasonably clean.

Line Processor.

The role of each of the processors will be described, beginning with the line processor. This unit is directly connected to the communication line buffer, and all communication with the external system passes through this card. Polling requests and control commands (generated by the console controller) are placed on the outgoing circuit by this machine. All arriving words are checked for validity of the parity, and for correct station of origin. Responses to control commands are passed back to the console controller via a buffer area in the main memory. Arriving status words are compared against status previously stored, and flags are set in main memory where changes are encountered. Telemetry values are compared against the stored limits, and out-of-limit counters are incremented or decremented. (The system requires three successive out-of-limit readings to log an alarm.) The data is then multiplied by the stored scale factor, the zero off-set is applied, if indicated, and the data is written into memory. Exponential smoothing can also be applied at this time, if called for. The decimal arithmetic required by these operations is performed in hardware which is located on the processor card, and is directly controlled by the processor. The line processor also deals with any failures of the remote stations to respond to polling requests. If the station has been responding normally, the machine will make three attempts at polling the station, and will, if unsuccessful, flag a station

failure. Subsequent to failure, only one attempt will be made per poll.

Systems having the remote stations distributed among several circuits may employ more than one line processor. Each will poll its remotes separately, and system response time will be reduced.

Console Controller.

The console processor views the operator's console as an I/O device. Any button pushing by the operator provides vectored interrupts to the processor, causing it to take the appropriate action. This would normally consist of sending control commands to the remote stations in place of the polling requests. Commands generated by the console processor are forwarded to the line processor for transmission to the remote stations. Command actions also cause flags to be set in the main memory to inhibit alarms from occurring in response to commanded changes. The console processor also scans the station status in main memory, searching out stations having points in alarm. When it finds one, it causes the station lamp on the console to flash. It also communicates with the display processor to cause an alarm message to appear at the bottom of the display screen. When there are stations in alarm, the processor searches the point memory space of those stations, and maintains the flashing status of the station lamp until all such points have been acknowledged. The acknowledgement, as will be seen later, is done by the display processor.

Printer Controller.

The printer controller responds to alarm conditions written into its buffer area in main memory by the line processor, or the console controller. These messages are highly abbreviated, consisting only of the station and point address, and an action code. The print controller, therefore, looks up the hard memory and the lexicon entries appropriate to the point, as well as clock time, to generate a message in plain text.

The printer controller may also be programmed to produce periodic or demand logs of all system variables.

The printer controller is also assigned the task of locating data block failures. Because of the method used for data polling as

described earlier, failure of a remote station to report a particular block of telemetry data is not immediately apparent. Each time a block of data is written into memory by the line processor, it sets a "recent write" bit. Periodically, these flags are scanned (and then cleared) by the printer controller. If it finds the flag bit not set, this indicates that the block has not been written during the last time period, and the printer controller will then generate a line failure message, and will set appropriate flags opposite the data involved.

Display Processor.

The display processor generates all display pages appearing on the VDU. It requires about 2 seconds to write a page on the screen, the limitation being the 9600 bit per second, serial I/O channel used to transfer data to the internal refresh memory of the VDU. Once the page has been written, the processor confines itself to the updating of the variable information, and will not require the "boiler plate" unless the page is re-selected. Update time is 1 to 2 seconds, and is carried out at intervals of approximately 3 seconds.

Status changes or alarm conditions, as has been mentioned, are flagged to the attention of the operator by flashing any data which is found to have a "change" flag set in main memory. When the operator acknowledges, the display processor, on its next scan of the variable data currently on display, will reset any "change" flags which it encounters. Thus all points, and only those points, on the current page will have their alarm status cleared. Alarms elsewhere in the system will retain their unacknowledged status.

Arithmetic Processor.

In addition to the processors so far described, which are required in every system, provisions have been made for the incorporation of other, special purpose processors.

An arithmetic processor is required if the system is to include either derived data quantities or interlocked controls. In the case of derived variables, the defining area in main memory specifies the address of up to six variables which may be manipulated to obtain the derived value, together with a pointer to the algorithm

to be used in the computation. Each of the six variables specified may itself be a derived variable, so that a total of 36 variables may be compounded to produce the desired result.

Map Board Processor.

A display processor, operating in a manner analogous to the VDU controller, may be provided for operation of a map board or mimic panel display. This processor would store, in its local memory, an associative table of the variables to be displayed, and the display device to which the data should be routed.

Data I/O Processor.

Because of the limited amount of memory available to the system, storage of data for historical analysis or for archival purposes in main memory is obviously impractical. Provision has therefore been made for a processor which would periodically write all system variables to magnetic tape for subsequent off-line processing by a computer. Alternatively, this processor might be used to provide a direct, two-way path to the computer, which would permit the system to retrieve processed historical data from the computer for display on the VDU.

MAIN MEMORY ORGANIZATION.

The main memory is organized as shown in Figure 2. Addressing is so arranged that the memory space appears as lines having a length of 16 words each. In the main portion of the memory, the first 8 words of each line are hard memory, implemented with 8,000 bit (1000 word) ultraviolet eraseable PROMs. The second 8 words of each line consist of soft memory, implemented with 2,000 bit (256 word) RAMs. Typically, only 4 words in the second half of the line are actually occupied. This somewhat unusual arrangement facilitates addressing of the variable data.

The bulk of the memory is used for point definition. Each line of memory contains the information pertinent to one data or status point. The hard portion of each line contains such things as a user-defined number, data type code, high and low alarm limits (optionally these limits may be located in soft memory), scale factor and zero offset. The soft memory portion contains the current value or status and several flags for out-of-limits, disabled, failed, changed, etc.

The lowest memory addresses are reserved for a station directory which holds the station identifier, station size, and flags for alarms within the station, station out of service, and similar items. Each station requires half a line of memory in this area.

The high end of the memory address space is used for a lexicon of station names and device descriptors, which are required for generating alarm messages in plain text. A typical entry in this area might read "Power Fail, Remote Station 4". As printed, this message would be followed by the customer-assigned number for that machine, and the current value of the variable. Each entry in the lexicon requires two lines of memory. The lexicon is limited to 256 entries, and is entirely PROM.

Another 32 lines of memory (soft) are reserved as a buffer area for the alarm printer. This area is sufficient for 128 alarms.

Each memory card contains 1024 lines of memory (16,000 words), which is enough to service 32 standard size remote stations. In addition to the 16 memory chips, the card contains the bus buffers and other supporting circuitry. A maximum of 8 memory cards may be installed in the system, while a minimum of 2 are required (one for data, and one for the lexicon and buffer area).

PROGRAMMING THE MEMORY.

Programming of the memory is accomplished by means of a separate, portable device called a programming panel. The panel includes a paddle, attached to the panel by means of an umbilical cable, which may be plugged into the main memory bus or the display memory bus. Within the panel is a memory card emulator, implemented entirely in RAM, into which data may be entered by way of the panel's own keyboard. Any block of memory address space may be assigned to the emulator. To expand the system, the operator enters the necessary point descriptors into the emulator, and connects the panel to the system. When he is satisfied that the new entries are operating correctly in the system, he can cause the programming panel to copy the emulator contents into a normal memory card which is plugged into the panel for this purpose. The newly programmed card may now be installed permanently in the system.

To make changes in existing memory cards, the programming panel may be caused to copy, into the emulator, the entire contents of any memory card, while the latter is on-line. Any desired changes may be entered into the programming panel, where they will overwrite the existing emulator contents. The original card may now be removed, and the system operated from the emulator, to be certain that the changes have accomplished the desired objective. The emulator contents are then dumped into a spare memory card, which is tested in the system. If the operator has not done the job correctly, the original card may be re-inserted in the system, and a second attempt made to change the program, and no permanent damage will have been done. When entirely satisfied with the changes, one may erase the original card for re-use.

It should, perhaps, be emphasized that the programming process, as described above, consists entirely of entering point descriptors in accordance with rigidly defined rules, and does not require any creativity on the part of the operator. A trained programmer is not required.

Since the fixed-format data in memory cannot be altered from within the system, unauthorized tampering is easily prevented by keeping the programming panel under lock and key.

The operating programs of the various processors are contained, in hard memory, on the same card as the processor. These programs are loaded at the factory, and are not considered as user-alterable. It is planned that the benefits of any improvements in, or explanations of, the operating programs would be conveyed to the user by sending him new, pre-programmed memory chips to replace his current programs. This is feasible because all integrated circuits in the system are mounted in sockets for easy replacement.

CONCLUSION.

In summary, we feel that by the use of an array of microprocessors, each dedicated to a specific portion of the entire task, we are able to extend most of the benefits of a large computer-based master supervisory to the smaller supervisory control systems at a reasonable cost.

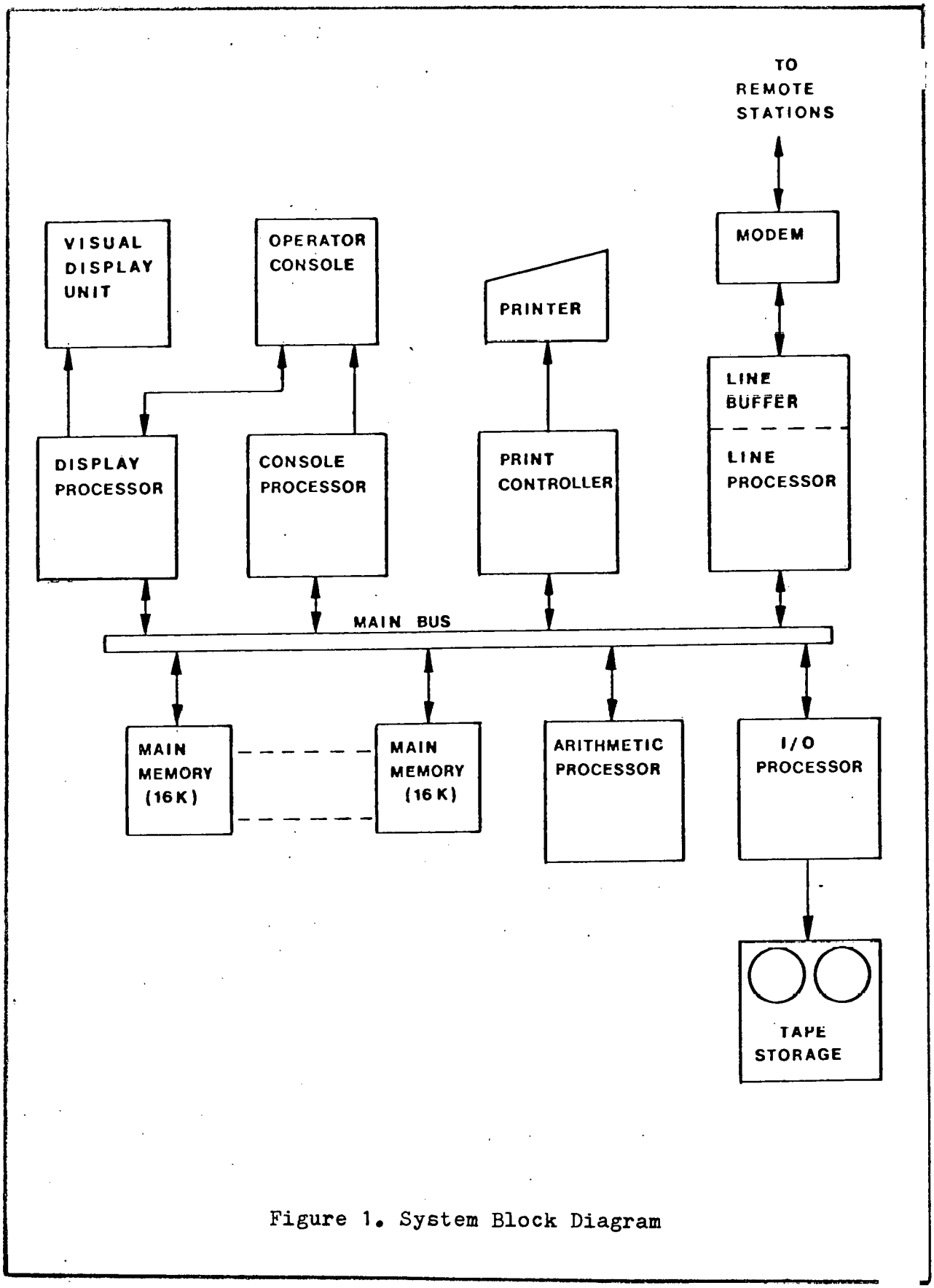


Figure 1. System Block Diagram

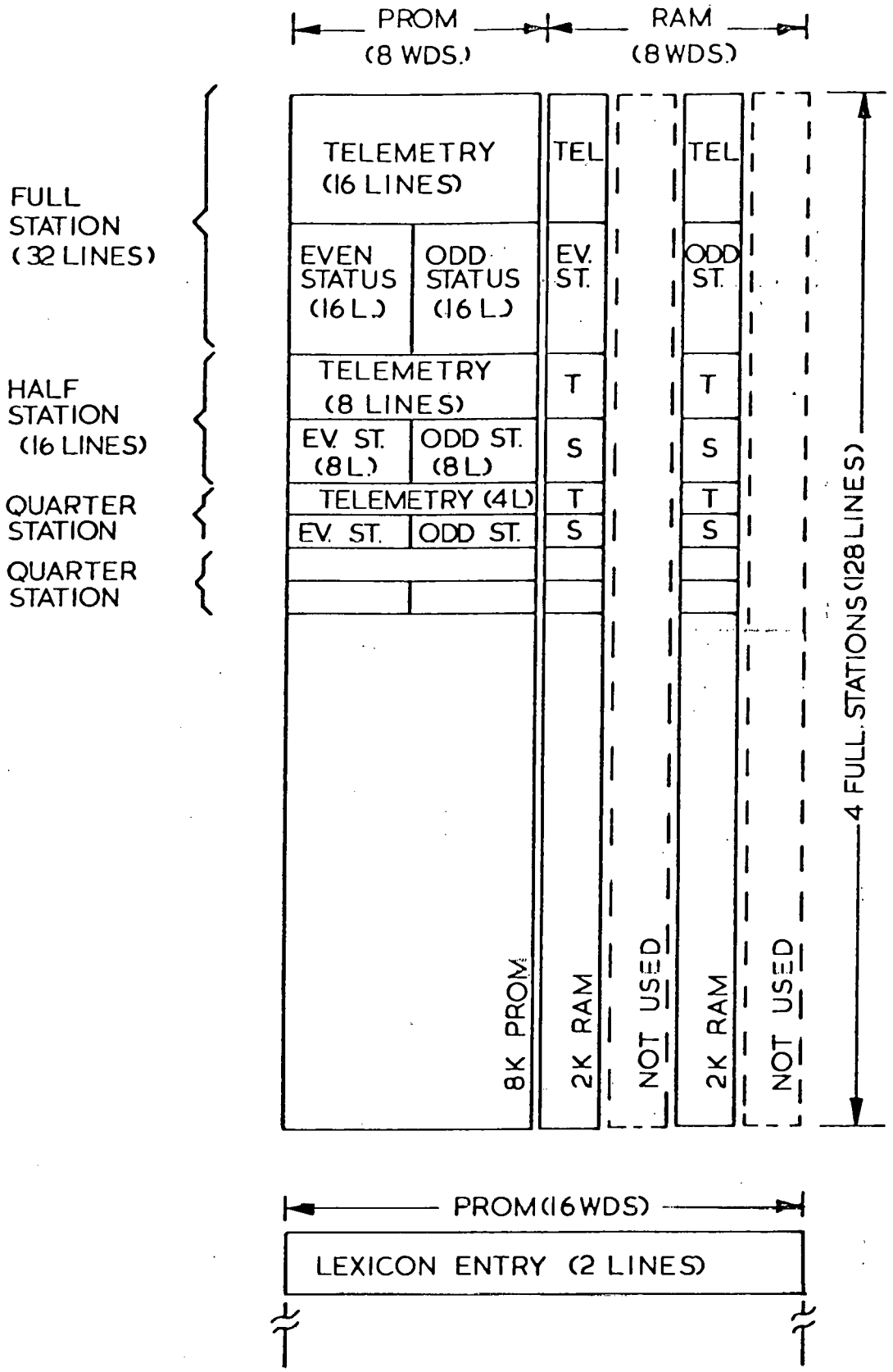


Figure 2. Organization of Main Memory

"A DIGTAL CONTROL STRATEGY APPLIED TO CONVEYOR-MOUNTED SHEARERS"

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ABSTRACT

An essential feature of automated longwall faces is a successful horizon control system and more recently effort has been directed towards controlling the steering of a Single Ended Ranging Drum Shearer (Ranger). Surface simulation work and underground trials experience has exposed the severe limitations of analogue control systems because of the Ranger's unstable geometry, the random effect of pan lift and the unstabilising influence of along face offsets. These effects have resulted in no exploitation of analogue control systems to Ranging Drum Shearers.

To surmount the above problems a new steering strategy has been developed which is based on a digital approach using micro processor technology. This will necessitate the development of new sensors, but the availability of memory gives greater freedom in the siting of sensors and removes all along face stability problems. Surface simulation work has verified the validity of the new approach.

The paper briefly describes the results obtained from a digital model of the underground process, which has been developed using analysis and underground data. A detailed description is given of the online control of a quarter-scale mechanical model of a Ranger, using a mini computer to simulate the steering strategy. The results demonstrate conclusively the viability of the new approach, and indicate that it is applicable to all types of conveyor mounted shearers operating in a variety of face conditions.

The paper further describes the implementation of the control system using micro computers mounted on the machine. A two way data link provides communication between the underground micro computers and a mini computer on the surface. In addition to the horizon control function the system will also cater for extensive machine health monitoring, thus enabling preventative maintenance and closer study of machine parameters under working conditions.

1. INTRODUCTION

Almost all deep mined coal output in the UK is derived from mechanised longwall faces where conveyor mounted power loaders are employed. To maintain high continuous yields from these faces it is essential that the profile of the path taken by the underground equipment as it moves through the seam is controlled in all three dimensions. Much work has been conducted in recent years to develop an automatic horizon control system, which would achieve control of the profile in two dimensions; for Ranging Drum Shearers which are becoming increasingly popular on NCB faces.

Analysis of the results obtained from preliminary underground trials conducted in 1974 identified the problems associated with steering Ranging Drum Shearers and led the way to a new approach to solve the horizon control problem (1). This is a digital control strategy and the work conducted to date to develop, test and implement the strategy on an underground machine is presented in this paper.

Digital computer simulation work has indicated that the predicted performance of the digital control strategy is promising. The control strategy has subsequently been applied to a surface based quarter-scale model of a Ranging Drum Shearer. The results obtained have indicated that the control strategy is able to maintain horizon control in many situations likely to be encountered underground, and that the limitations of the system are functions of constraints imposed by the mechanical limitations of the machine rather than constraints in the philosophy of the control strategy (2).

The success of these preliminary studies has led to the development of an underground system for implementing the digital control strategy. This has necessitated the development of a number of transducers and the system relies heavily on recent developments in computer technology. Micro computer systems mounted on the machine will implement the control strategy. They will also enable parameters associated with steering and the health of the machine to be continuously monitored under working conditions and transmitted to the surface for closer study. The system has been designed to be sufficiently versatile to ensure that it is applicable for control and monitoring of a wide range of machinery used within the industry.

2. REQUIREMENTS FOR A HORIZON CONTROL SYSTEM

The mechanical design of the Ranging Drum Shearer, which is now in use on the majority of coal faces of the NCB, is aimed at convenience and reliability from a mining viewpoint. The outboard drum is well suited to allow the cutting of stables at face ends and to obtain large extraction from a thick seam, but this geometry makes the machine inherently unstable.

A complete steering system should be capable of reliable performance on the majority of underground faces and to achieve this applicability it must be able to accommodate a wide variety of sensors.

The system should also be able to cater for different methods of mining operation and steer all types of conveyor mounted shearers correctly regardless of the general nature of the seam or local variations found within the seam.

Primary objectives for the performance of a steering system for a Ranging Drum Shearer for use in the mining environment encountered in the UK are to dynamically position the cutting drum to maintain a roof and floor profile coplanar to the seam, to leave a fixed amount of roof coal for roof stability, to maintain face advance tilt (FATILT) within certain limits, and to maintain a constant extraction. To achieve these objectives the cutting drum is required to remain at a specific horizon regardless of the disturbances imparted to the machine. These are often random in nature and are usually introduced when the semi-rigid armoured face conveyor (AFC) on which the machine rides is pushed over on to the newly cut floor. It is therefore apparent that a steering system must always possess sufficient information to compute the position of the cutting drum within the seam, regardless of the attitude of the machine body. However, the nature of the system is dependent on the data that can be measured.

Much work has been conducted at MRDE in recent years to develop suitable control systems and the means of measuring appropriate data. One of these methods, with particular emphasis being placed on Single Ended Ranging Drum Shearers (SERDS), is described in detail in the following sections.

2.1 Horizon Control Based on Previous Cut Data

The normal method of mining when using a SERDS is a two pass process. On the first pass the roof is cut with the drum trailing the machine body. On the second pass the machine returns with the drum down taking the floor cut and clearing the coal on to the AFC. The conveyor is then pushed over on to the new cut floor to take the next shear. However, owing to its semi-rigid nature the conveyor is not able to exactly follow the floor profile unless it is perfectly flat, and consequently vertical disturbances are always introduced into the conveyor to some extent. In the absence of steering action these disturbances would be imparted to the machine body and reflected into the roof as the shear commenced, and because of the machine geometry these would be both phase shifted and amplified and would soon lead to system instability (3).

To overcome this the measurement of the height of the roof in the previous cut shear at the drum centre line is required. The drum is then positioned so that the error between the heights of previous and present cut roofs is zero, consequently the drum always follows the previous cut roof regardless of vertical disturbances in the conveyor. Ideally this control action is implemented with 1/1 steering power and is termed roof follower action.

To position the drum at a particular horizon leaving a desired thickness of roof coal, a measurement of the previous cut roof coal thickness at the drum centre line is required. For practical reasons coal thickness data appertaining to the previous cut is most easily measured at a point located above the main body of the machine. Consequently, the coal thickness is not measured in the line of drum and a time delay exists in the control scheme, having a destabilising effect (3, 4). This can be overcome by storing the measured coal thickness as the machine traverses the distance between sensor and drum, and the horizon control system acts on coal thickness data appertaining to the drum centre line. This is termed horizon control action.

The requirement for both roof follower and horizon control signals to position the drum in the seam is necessary because the measured data used for control does not appertain to the drum centre line in the present cut. If it were possible to measure the current position of the drum relative to the seam, ie present cut data, drum control would become independent of the attitude of the machine body, provided the mechanical limits of the machine were not exceeded, and both roof follower and horizon control action would be inherent.

2.2 A Digital Control Algorithm for Previous Cut Data

It is possible to define the control algorithm from the philosophy and objectives of a steering system as discussed above.

2.2.1 Horizon Control Action

The horizon control action signal BPCONT is defined by the equation

$$BPCONT = G_{CT} * (Y_C - Y_{REF}) + G_{FATILT} * FATILT \quad \ldots\ldots\ldots\ldots 1$$

where Y_C = measured coal thickness in line with the drum for the previous shear (mm)

Y_{REF} = desired coal thickness (mm)

FATILT = machine face advance tilt relative to the seam (degrees)

G_{CT} = coal thickness gain (mm/mm)

G_{FATILT} = face advance tilt gain (mm/degree)

The first part of the equation calculates the control signal required to move the drum towards horizon using previous shear data, and the second part introduces correction for face advance tilt.

2.2.2 Roof Follower Control Action

The roof follower control action signal BPFOLL is defined by the equation

$$BPFOLL = G_{RF} * Y_H \quad \ldots\ldots\ldots\

SERDS whose steering is controlled by the digital system (5). The results obtained were very encouraging, and led to the quarter-scale model trials described in detail in Section 4. Consequently this work is not described in detail as many of the results would be duplicated. However the simulation has been used to derive suitable values for the control algorithm gains used in the quarter-scale model trials, and to illustrate that roof follower control action, described earlier, is necessary for system stability. These results are described below.

3.1 Some Results from the Simulation of a Digitally Controlled SERDS

The simulation programme was configured to include a SERDS fitted with the appropriate sensors to implement horizon control based on previous cut data, with the exception of the data required to implement roof follower control action. As an initial condition a half sinusoid disturbance of 100 mm magnitude was introduced into the conveyor profile in the along face direction. The simulation had predicted that if roof follower control action was enabled, this disturbance was effectively eliminated in one shear. However, with the absence of roof follower action this disturbance was sufficient to make the control system unstable, and results that illustrate this are shown in Figure 2.

The roof profile after the first shear is shown in Figure 2a. As the coal thickness data appertained to the previous shear and no roof follower action was present, the disturbance in the conveyor profile was imparted into the roof profile. It can be seen that on either side of the hole cut into the roof a small hump was cut. As the leading feet of the machine began to climb the disturbance, the pivoting action about the rear feet lowered the cutting drum. Owing to the dynamics of the horizon control loop there was a time delay before corrective action was applied to return the drum to horizon. The same process applied in reverse as the rear feet of the machine moved down the disturbance, cutting the second hump into the roof.

The method used to take the floor cut meant that a similar disturbance was cut into the floor profile, and consequently face advance tilt was introduced for subsequent shears.

The roof profiles for shears 4 and 7 are shown in Figures 2b and 2c respectively. It is evident that for subsequent shears the disturbance was becoming both greater in magnitude and offset along the face and that the control system was becoming increasingly unstable.

With the simulation configured to incorporate roof follower control action the controlled responses of the machine in the face advance direction, for a number of coal thickness and face advance tilt gains, to a demanded change in horizon of 100 millimetres are shown in Figure 3. If too high a FATILT gain was used the profile shown in Figure 3a resulted. A sawtooth profile was cut into both the roof and floor resulting in high values of FATILT in the first two shears which would be unacceptable underground. If too low a value of FATILT

gain was used Figure 3b illustrates that considerable overshoot was obtained which would also be unacceptable underground as the machine would cut into stone in the roof. By selecting a gain of 11 mm/degree a smooth response was obtained with no overshoot nor excessive face-advance tilts, as shown in Figure 3c.

3.2 Conclusions from the Computer Simulation

The results illustrated are not a rigorous test of the digital control algorithm as only two sets of experimental results are described. However the necessity to incorporate roof follower control action to maintain system stability is illustrated, and values for coal thickness and face advance tilt gains have been obtained which provide a satisfactory dynamic response in the face advance direction.

The simulation was used to thoroughly test the performance of the control scheme for a wide range of conditions. These results have not been described in detail, but they indicated that the control scheme could maintain stability and adequately control the cutting of the roof and floor in most situations that would be encountered underground.

4. DIGITAL CONTROL OF A QUARTER-SCALE MODEL OF A SERDS

As a result of the performance of the digital control system indicated by the simulation it was decided to test its performance with an existing quarter-scale model of a SERDS cutting polystyrene under the control of an online digital computer. Although the results from the digital simulation were promising, it was known that the models, particularly of the AFC, were not highly accurate representations of the real items and these trials would serve to test the control strategy under slightly more realistic conditions.

4.1 Description of Model and Computer System

The general arrangement of the equipment used in the trials is shown in Figure 4. All signals between transducers on the model and the digital computer were transmitted through an analogue computer via multicore cables.

4.1.1 Quarter-Scale Model

The main features are described below. All dimensions stated relate to a full sized machine.

(a) Haulage

The model was hauled along the conveyor by means of a small motor driving a haulage chain through a sprocket. The haulage speed was equivalent to 24.8 mm/s.

(b) Conveyor

The 23 pan conveyor structure on which the model

travelled had the same limited angular freedom as its full size counterpart. Each pan was equivalent to 1.5 m. Pushover was achieved manually.

(c) Cutting

The trials were conducted with the model system sited on a polystyrene floor, into which the floor profiles were cut. The cutting drum, which was driven by a small motor, was fitted with blades to achieve this.

(d) Steering

The cutting drum was sited at the end of an outboard boom which could be raised or lowered by means of a gear arrangement driven by a small motor. The motor could be controlled remotely.

(e) Sensors

The model was equipped with five sensors, giving measurements of face advance tilt, boom position (BP), along face position (AFP) and vertical height.

(i) Face advance tilt was measured by a tilt transducer whose mode of operation was similar to its full size counterpart.

(ii) The angular position of the boom was measured using a linear potentiometer fitted to the boom. The analogue computer converted the angular measurement to a linear displacement.

(iii) Along face position was measured by counting the pulses from a micro switch, which provided five pulses per revolution of the haulage motor shaft.

(iv) The vertical height reference was a narrow plane of light produced by three carefully sited collimated light sources. Two servo-driven light sensitive detectors were positioned on the model; these tracked the light plane to measure vertical deviations of the model from the reference. The transducers readings also enabled the along face tilt (AFTILT) of the machine to be derived.

4.1.2 Analogue Computer

The main functions that were performed by the analogue computer, an EAI TR-48, are listed below.

(a) Interface with Digital Computer

The analogue computer was used as an interface between

the model and the digital computer; it performed such functions as providing positive voltages for the unipolar analogue to digital converter (ADC) of the digital computer, and scaling of the amplitudes of the transducer outputs from the model so that the model appeared to be full size to the computer.

(b) Coal Thickness Sensor Simulation

At each point along the face the coal thickness above the drum was calculated by the digital computer by means of an algebraic equation and the outputs from the height transducers. The results were stored within the digital computer for the duration of one shear as the control algorithm used previous shear coal thickness data.

The dynamic characteristics of a nucleonic coal thickness sensor as used underground were approximated by a first order transfer lag of time constant 8 seconds. As the equivalent full size haulage speed of the model was considerably slower than that of a full size machine the time constant of the sensor had to be scaled. This was readily achieved on the analogue computer.

(c) Simulation of Boom Position Servo

The maximum possible rate of travel of the boom on the model was faster than the typical rate of 6 mm/s on a full size machine, and scaling had to be applied to the servo loop rate integrator to slow it down. The operation of the solenoid valves, and the associated dead band found in the full size system, were also simulated on the analogue computer.

4.1.3 Digital Computer

The digital computer controlling and monitoring the performance of the model was a pdp 11/10 computer with the RSX11M operating system An initialisation software programme was run first, allowing specific variables such as controller gains and display scaling factors to be selected. The model control programme was initiated by a pulse generated on the model at regular along face intervals. The computer then sampled the outputs from the model's transducers, computed the steering demand and passed the demand to the boom position servo to move the drum appropriately. All parameters required to analyse the performance of the model were stored on disk and were available for display.

If requested, a graphical display programme was also available. This was written for use with a Tektronix 4010 visual display unit (VDU), and enabled all data previously recorded during the steering trial to be displayed graphically in any combination defined by the operator. The programme could also be used to display current data dynamically as the model was running.

Five variables were chosen for monitoring and display: roof profile (ROOF PRO), face advance tilt (FATILT), along face tilt (AFTILT), roof follower control action (BPFOLL) and horizon control action (BPCONT).

4.2 Digital Steering Control Algorithm

The control equations described in Section 2.2 were programmed into the computer to enable steering of the model.

(a) Implementation

Owing to the physical limitations of the model, it could be used only to cut the floor. Consequently the roof cut had to be simulated by the digital computer. To ease the implementation of the control algorithm the simulated roof cut was computed for the same along face position as the floor that was being cut currently; consequently the roof profile was computed in the reverse direction with respect to time to that in which it would be cut in practice. Therefore the time dependent dynamic characteristics associated with the coal thickness sensor were not included in the roof cut simulation. However their exclusion meant that the unfiltered coal thickness signal seen by the control system was noisier than would normally be the case, and this imposed a more rigorous test on the stabilising ability of the control system.

(b) Evaluation

The objectives of the trials were as follows:

(i) Demonstrate the viability of the digital approach to horizon control. This is mainly a function of the controller design considerations (4, 6).

(ii) Assess the severity of disturbances with which the controller could cope. The rate of change of a disturbance will effect the performance, as the position along the face where steering action becomes effective is a function of the fixed speed of the shearer and the limited rate of boom movement. Although the inability of the boom to respond sufficiently fast does not necessarily denote instability in a mathematical sense (7), it is constraint in practical terms.

(iii) Determine the number of shears required before disturbances were eliminated.

(iv) Demonstrate that the performance of the control strategy predicted by the digital simulation facility was sufficiently representative of its behaviour when controlling a SERDS, albeit a model, to establish the usefulness of the simulation for preliminary control investigations.

4.3 Results

Results from two trials are presented. The illustrations were produced by the digital computer using the graphic display facility described previously. Data for display was collected at every fifteenth control interval, corresponding to approximately five data collection points per pan length over twenty pans. Distances and tilts have been scaled and expressed as full size, and are shown in the diagrams in millimetres and twentieths of a degree respectively.

The coal thickness gain (G_{CT}), face advance tilt gain (G_{FATILT}) and roof follower gain (G_{RF}) of the control algorithm were assigned values of 0.5, 11 and 1 respectively. These values were deduced from the computer simulation results described previously and gave a good predicted performance in both the along face and face advance directions.

Typical speeds for an underground SERDS are 61 mm/s during the cutting run and 102 mm/s during the flitting run, consequently during these trials the model was scaled to cut at one of these two speeds. The boom ranging rate was restricted to represent 6 mm/s. Therefore the maximum along face gradients which the machine could negotiate and also maintain drum position were 5^{o} at 61 mm/s and 3.3^{o} at 102 mm/s.

The roof and floor profiles were cut with reference to a horizon which was determined by the measured height of the plane of light. Because of mechanical vibration of the model, which is an inherent feature of its self-hauled mode of propulsion, the measured height signals were subject to superimposed noise. Tests indicated that the noise band could oscillate between $\pm$ 12 mm of horizon, and this effectively set the accuracy to which the machine could cut.

4.3.1 Trial 1

This trial was undertaken with the model scaled to cut at 102 mm/s and with initial vertical disturbances along the line of face introduced into the conveyor at two sections. The first disturbance was three conveyor pans in length incorporating pans 5 to 7: pan 5 formed a ramp rising to 116 mm above the conveyor datum, pan 6 was horizontal and pan 7 sloped back down to the conveyor datum. The second disturbance consisted of a 116 mm thick block lifting pan 13 at one point. Consequently the conveyor was able to pivot about this point, and the conveyor profile changed as the machine feet traversed it.

The values of the monitored parameters along the length of the first shear are shown in Figure 5a. As no initial disturbance had been specified in the roof profile, which was on horizon, no control action was taken and BPCONT remained at zero. However, roof follower action BPFOLL detected the floor disturbances and took instantaneous action as is evident from comparison with the AFTILT signal.

The gradient of the ramps constituting the first disturbance was 4.57° which exceeded the maximum gradient at which the drum position could be maintained when the machine was cutting at 102 mm/s. At this speed the quarter scale model took 15 s to climb the ramp, and in that time the drum could only be lowered 90 mm; consequently it could be deduced that a hole of approximately 26 mm depth would be cut into the roof. This was confirmed by observation of the roof profile ROOF PRO. After the front feet of the model had passed over the disturbance and the rear feet had started to mount the ramp the boom would have to rise in order to avoid leaving too much coal in the roof. However this situation was similar to that described above, and a hump was left in the roof profile as illustrated in Figure 5a.

Owing to the nature of the second disturbance, the weight of the model changed the effective profile of the disturbance as the model moved in its vicinity. As the leading feet of the model approached the disturbance the conveyor was held down until the feet moved on to the pan under which the disturbance was situated. The conveyor then pivoted about the disturbance and the model had to mount the disturbance within the distance of approximately half a pan. Because of the non-stationary nature of the process the gradient of the conveyor could not be accurately determined, however for most of the time it was greater than 4.57°. A similar mechanism operated as the rear feet passed over the disturbance. Because the average gradient of the second disturbance was greater than that of the first, the model as expected cut a greater amplitude disturbance into the roof, and Figure 5a shows this.

The cut floor profile was not generated for display by the computer as it could be observed in the polystyrene as the trial progressed. However, because of the nature of the control algorithm the cut floor profile was, as expected, similar in shape to the roof profile.

On completion of the first shear the conveyor was pushed over on to the cut floor, and because of the difference between the initial floor profile and the cut floor profile FATILT was introduced. Consequently during the second and subsequent shears control action BPCONT was introduced to counteract FATILT and the deviation of the drum from horizon, and roof follower action BPFOLL remained to compensate for vertical pan disturbance.

The results recorded for the fourth shear are shown in Figure 5b. Face advance tilt was now quite severe, particularly in the vicinities of the greatest roof and floor disturbances. The BPCONT signal was approximately complementary to the deviations from horizon in the roof profile ROOF PRO, variations in this relationship being due to FATILT for which BPCONT was also trying to compensate. Vertical disturbances in the floor were also quite severe, as indicated by the amount of correcting roof follower action. The propagation of the original disturbances from shear 1 to shear 4 can be seen by comparison of the ROOF PRO signals. The magnitudes of the disturbances increased, particularly for the second disturbance which possessed the higher rate of change; and both disturbances were

displaced along face because of the nature of the outboard drum. It is evident that disturbances sufficient to cause instability were introduced into the system and propagated (5).

These results indicate that the combination of the rate of change of the disturbances used as the initial conditions for this trial and the fast cutting speed of the machine were too severe for the constrained boom speed. The boom speed of underground machines is fixed at a typical speed of 6 mm/s. This is a function of machine design. Consequently to handle disturbances with a significant rate of change the machine cutting speed, which is not yet a feature that can be controlled automatically, must be reduced accordingly. This highlights an area in which machine design must be changed if the full benefits of automatic steering are to be realised.

4.3.2 Trial 2

The trial was conducted with the model scaled to cut at the slower speed of 61 mm/s. Three separate initial disturbances were introduced into the conveyor along the line of face. The maximum gradient provided by these disturbances was 4^{o}, sufficiently low to be accommodated by the boom at the cutting speed of the machine.

In addition to the disturbances in the conveyor an initial disturbance was incorporated in the roof profile. It consisted of one full-sinusoid of 75 mm amplitude, centred about horizon, and running from pan 2 to pan 16.

A diagram of these initial disturbances is shown in Figure 6a and the results recorded for the first shear are shown in Figure 6b. A gradual increase in FATILT can be observed along the length of the face. This was caused by deformation of the structure supporting the model due to pressure exerted by the gripped polystyrene. This is reflected also in the BPCONT signal which was attempting to counteract this tilt, however the effect is masked between pans 2 and 16 by the large sinusoidal excursion of BPCONT as it tried to counteract the initial disturbance in the roof profile. At the disturbance peaks the control signal saturated and this was reflected in the roof profile as at these positions disturbances still existed in the roof. However, within the ranging limits of the drum, the roof profile was reduced to within the noise band of the transducer signals.

The AFTILT disturbances for this shear were quite severe, and the control action taken to reduce them is reflected in the BPFOLL signal.

The parameters monitored for the second shear indicated a slight reduction in the magnitude of the deviations of the AFTILT signal, and a corresponding change in the BPFOLL signal. However, because of the difference between the initial floor profile and the first shear floor profile, considerable FATILT was introduced along the face. This caused small disturbances to be propagated into the roof profile.

Figure 6c illustrates the parameters monitored during shear 4. AFTILT, and the corresponding variations in BPFOLL, were reduced compared to their magnitudes in shear 1. However FATILT was still quite severe at most points along the shear. The roof profile was seen to be considerably worse than the profile for shear 1, however it was smoother than the profile for shear 2 and at this stage appeared to be improving.

The parameters continued to improve during each subsequent shear, and the results at the end of shear 7 are shown in Figure 6d. The roof profile was now quite flat, and within the noise band of the transducer signals; as was the observed cut floor along face. This is indicated by the AFTILT plot. However some FATILT existed in the second half of the shear, although its magnitude had been decreasing with every shear. It was envisaged that FATILT would be reduced to zero in the following one or two shears. A different value of G_{FATILT} may have improved the response of the control system to face advance tilt but system optimisation was not the objective of this trial.

Photographs of the model SERDS and the results of this trial are shown in Figure 7. Figure 7a shows the three initial conveyor disturbances described earlier, the model having completed the first shear. Figure 7b shows the floor profile at the end of the trial. The profile that was cut into the floor on the first shear as the model passed over the initial disturbances can be seen clearly on the left hand side of the polystyrene. In subsequent shears the profiles gradually improved, and by the end of the trial the conveyor was effectively horizontal.

4.4 Conclusions from the Quarter Scale Model Trials

As a result of these trials it is possible to conclude that:

(a) The digital control system, which incorporated roof follower action to compensate for vertical conveyor disturbances and used previous shear coal thickness data in line with the drum, is capable of maintaining stable steering control provided the rate of change of the disturbances propagated by the conveyor is not sufficient to require the boom angle to change at a faster rate than its maximum ranging speed. This is a function of the machine boom hydraulics and the machine cutting speed. (It is known that in exceptional circumstances, disturbance gradients of 7^{o} along face can be encountered underground (8). To accommodate disturbances of this magnitude at a cutting speed of 102 mm/s the boom must be able to range at a speed of 12.5 mm/s. Conversely to accommodate this disturbance with a boom ranging speed of 6 mm/s the cutting speed of the machine must be reduced to 49 mm/s.)

(b) The control system is capable of effectively smoothing disturbances introduced into the roof and floor profiles after approximately seven shears, subject to the provision stated above.

(c) The trials have confirmed that the results predicted by the digital simulation facility described in Section 3 are reasonably accurate, thus validating the digital models for further preliminary control investigations.

Previous work conducted using mechanical models of shearers (9) had indicated that a model was more difficult to control than an underground machine owing to the inability to accurately simulate conditions underground. Dust and fine coal produced during cutting tends to fill any depressions cut into the floor, and consequently reduces undulations in the conveyor profile, helping to reduce the possibility of instability.

Consequently the results of these trials suggest that provided the constraints detailed above are not exceeded the digital control algorithm should be satisfactory for controlling the steering of an underground Ranging Drum Shearer. A system to implement digital horizon control on such a machine is now in an advanced state of development and is described below.

5. REQUIREMENTS FOR DIGITAL ELECTRONICS FOR CONTROL OF COAL FACE MACHINES

Consideration was given to the design of computing equipment for mobile machine control in 1973. It was considered that equipment based on mini computer technology would be situated remote from the machine, and information and control signals would be transmitted between the controller and the machine.

The widespread introduction of LSI techniques has made it possible to situate a microprocessor based system of control on the machine, with transmission of machine data to remote stations which perform a monitoring function only.

The requirements of the system are as follows:-

(a) To provide sufficient computing facilities to enable the previously described control strategy to be fulfilled.

(b) To receive and process information required for the control strategy and to transmit useful information from a mobile machine to a remote point.

(c) To be mechanically and electrically capable of being integrated into a coal face machine and interfaced with the operational controls.

(d) To cope, from safety and reliability aspects, with the harsh underground environment.

(e) To provide simple first line fault indication which is suitable for the maintenance facilities on a coal face.

6. HORIZON CONTROL TRANSDUCERS AND MEASURING DEVICES

A control system is only as good as the quality of information it receives and the research and development of the measuring equipment has been at least as onerous as the design of the digital computing equipment. The R & D of measuring devices will be the subject of another paper, but it is necessary to briefly describe the devices and the nature of their output to specify the computing requirements.

It should be noted that it is not necessarily the intention to fit all transducers to a machine but to have available a number from which a selection can be made to meet the conditions of each underground installation.

6.1 Coal Interface Detectors

6.1.1 Natural Radiation Sensor

This device has been developed as a successor to the nucleonic sensor used on previous analogue control systems. It detects the natural gamma radiation from the stone above the seam and measures the proportional attenuation of this radiation by varying thicknesses of roof coal. Naturally radiating isotopes are present almost universally in UK strata and the major feature of the sensor is its measuring range of 0 - 45 cms of coal thickness. A standard transducer output range of 1.6 V dc is presented to the computing equipment.

Since this transducer can only be mounted at some distance from the drum and in the previous cut the information must be stored until the drum is at the point relating to the information and used in the horizon control method described in Section 2.1.

6.1.2 Pick Force Sensor

This transducer is in the course of development and is based on successful underground trials of the 'sensitised pick' carried out in 1972. Essentially the sensor consists of a pick instrumented by a piezo crystal or strain gauge which produces an electrical signal range of 1.6 V for the range of forces it experiences while cutting the strata. Since coal measures are sedimentary it is expected that the pattern of forces for each revolution of the drum will be statistically similar even though individual bands of varying hardness may not produce a corresponding force signal during each revolution. The sensor does not directly indicate the horizon of the drum. This information is obtained from processing of the sensor output data whose collection is synchronised by the measurement of the pick angular position round the periphery of the drum.

The technique used is to position the drum at the desired horizon and then to sample the sensor outputs as the drum rotates. From the average of a number of revolutions a master pattern is produced. Each successive revolution produces a pattern which is correlated with the master to indicate firstly, that there is a correlation and secondly, the displacement of the drum from the horizon of the master pattern. Examples of current and master patterns obtained from a surface trial are shown in Figure 8a. Although these patterns are apparently different, there is a significant correlation between them as indicated in Figure 8b. In this case the deviation of the drum from horizon is small. This deviation can be used to position the drum directly since the data is related to the present cut.

Some indication of the task imposed on the machine mounted computing equipment is that surface trials have indicated that up to 128 samples are required for the half revolution whilst the pick is cutting and the ensuing correlation will require 4000 multiplication operations during the remaining half second while the pick completes the revolution to its new cutting path.

6.2 Other Transducers Required for Horizon Control

6.2.1 Roof Height Sensor

This instrument is required to provide the roof follower information for horizon control systems using previous cut data. It is intended to use a non-mechanical device for operational safety and the most likely development is an air sonar sensor. This is reasonably immune to water and dust and will give a 1.6 V signal range for a measuring range of 20 cms to 150 cms in coal face conditions.

6.2.2 Along Face Machine Position Transducer

This is a key transducer which initiates the execution of the digital control algorithm, as the data sampling interval is based on distance travelled along the face. The machine position is derived from optical vane detectors, which for improvement of integrity generate three idential pairs of out-of-phase pulse trains, defining machine motion and direction of machine movement.

6.2.3 Machine Tilt Transducer

These are damped pendulums driving potentiometers, and measure tilt in longitudinal and transverse axes relative to the seam.

6.2.4 Boom Angle Transducer

This is a rotary potentiometer measuring boom angle relative to the axis of the machine body.

6.2.5 Machine Health Transducers

Although not required directly for horizon control, it is intended to aid system integrity and fault identification by providing conventional pressure, flow and temperature transducers to monitor the hydraulic system.

6.3 Arrangement of Transducers

The disposition of transducers on a Double-Ended Ranging Drum Shearer is shown in Figure 9. The position of the digital control electronics is also shown and a brief description of these follows.

7. FUNCTIONS AND DESCRIPTION OF THE DIGITAL CONTROL ELECTRONICS

The electronic modules on the machine compose three micro-computers and three other electronic modules which provide man/machine/remote information interfaces.

7.1 The system is shown diagrammatically in Figure 10 and is built round the executive micro computer. This is based on an Intersil 6100 CPU which has 4K of random access memory (RAM) and eight input/output channels. Underground application dictated both the choice of CPU and interconnection to other modules and these are discussed further in 7.2. The role of the executive is to provide intermodular communication and carry out certain data integrity checks. Up to 30 analogue transducer and 12 logic state signals can be accepted by the Data Access and Display Unit (DADU). This also has a 12-key ruggedised keyboard and liquid crystal display (LCD) enabling transducer or fault information to be indicated. All information is digitised at the DADU prior to being transmitted to the executive.

The Pick Force and Steering Analyser micro computers are identical in hardware and have 20K of RAM, hardwire multiplication circuits and an ADC to carry out the pick force correlation described in 6.1.2. When programmed as the steering analyser the module receives horizon control data via the executive, checks integrity of inputs and outputs and generates steering demands which are actuated through the machine interface module. The machine interface module essentially segregates 220 V solenoid driving circuits from the digital control signals by the use of triacs and toroidal transformers. This module also monitors the machine pilot circuit and the position of the control mode selector switches.

The method of data transmission to the adjacent roadway (gate) is a bidirectional time division multiplex system using the pilot core of the machine trailing cable. The data transmission rate is 50 Kilobaud. The data is received in the gate where an identical executive and DADU allow access to information within the system. Similar data transmission equipment will transmit over 10 Km to the surface, where information will be processed by a pdp 11/34 minicomputer and displayed for trials and managerial information.

7.2 Design Features to meet Hostile Environments

7.2.1 Underground Environmental Conditions

A summary of the environmental conditions indicates some of the problems which have been taken into consideration during design.

(a) Shock and Vibration

Machine mounted equipment will be subjected to continuous vibration of up to 1 g peak acceleration in the frequency range 10 - 2500 Hz and during transport the equipment may be subjected to shocks of up to 40 g.

(b) Electrical Noise

The maximum severity of transient noise on an underground mains supply is not known but transients of twice the magnitude of nominal supply voltage have been measured. The mains supply to the machine can vary by up to 20%.

(c) Explosive Atmosphere

Present regulations and practice in the UK restrict all underground electrical equipment to two categories. These are intrinsically safe equipment, whose power requirements are sufficiently low to enable them to be situated in the mine atmosphere, and non-safe equipment which must be situated in a flameproof box.

7.2.2 Choice of IC Technology

From consideration of temperature, ventilation, power consumption, noise immunity and intrinsic safety it was apparent that CMOS technology was preferred for micro computer design. This technology has the inherent feature of low power requirements and when powered at the design voltage of 10 V, will possess a high degree of noise immunity. The components used are of industry standard with an operating range of $-20^{\circ}C$ to $120^{\circ}C$. The Intersil 6100 CPU was chosen firstly because it was, at the time, the only CMOS microprocessor in widespread use and secondly it was software compatible with DEC pdp 8 minicomputers enabling the latter to assist with software development.

7.2.3 Power Supplies and Intermodular Connection

Power is derived from the 1100 V ac supply to the machine, which is initially transformed to 220 volt for the solenoids, and then reduced further via a constant voltage transformer (CVT) for the control system supplies. The solenoids are suppressed by varistors to avoid interference with the control equipment.

The executive, DADU, multiplex and machine interface modules are supplied from separate IS supplies with battery back up. The

pick force and steering analysers are supplied from a non IS supply when machine power is available. As the modules are supplied from separate supplies, segregation must be maintained between modules, and since the only connection between modules is through the data link, this link is designed to provide segregation.

To minimise wiring between modules data links are based on serial transfer using UARTs. The rate of intermodular transmission is 200 kilobaud, preventing the use of optical couplers. Therefore segregation is provided by passing the data through specially designed toroidal transformers which provide a galvanic break of 2 mm between the circuit inside a module and the lines connecting it to another module. Four wires provide two way data communication between UART pairs, and as the four wires are segregated from both modules all transmission lines can be loomed together.

The segregation between the machine to gate data link and the pilot core is also achieved by using toroid transformers.

7.2.4 Mechanical Construction

The modules have been designed to resist the effects of vibration and shock and all PCBs are either wired in situ or connected by two part connectors. The boards are securely clamped in position and covered with a 3 mm layer of semirigid epoxy resin which provides physical support for components. The resin also provides an insulating layer protecting the circuit from dust and moisture.

7.3 Fault Indication Facilities

Any module connected to the executive which fails to respond in a pre-defined manner will cause the executive to recognise a fault condition. The executive is then able to disconnect the UART output to the suspect module and connect it to its own input. Thus the executive can test its own input/output channels up to the point where they leave the module and decide whether the fault lies within itself or in the other module.

A fault at the machine is made known to the operator through the DADU. If it is the machine DADU that is faulty the fault will be reported on the gate DADU and on the surface. If the machine executive fails, the fault will be recognised by the other executive in the gate. Thus the system determines which module is faulty, and informs both the operator and the surface equipment accordingly.

Faults are defined as fatal or non-fatal and degrade or prevent automatic horizon control as required.

8. CONCLUSIONS

Conclusions for the results of the digital computer simulation predictions and the subsequent quarter-scale model trials described in this paper have been presented in the appropriate sections. These indicate that the digital control strategy will be capable of

satisfactorily achieving automatic horizon control when applied to an underground SERDS.

This has led to the development of a practical system to implement the strategy, and underground trials are expected to commence during 1979.

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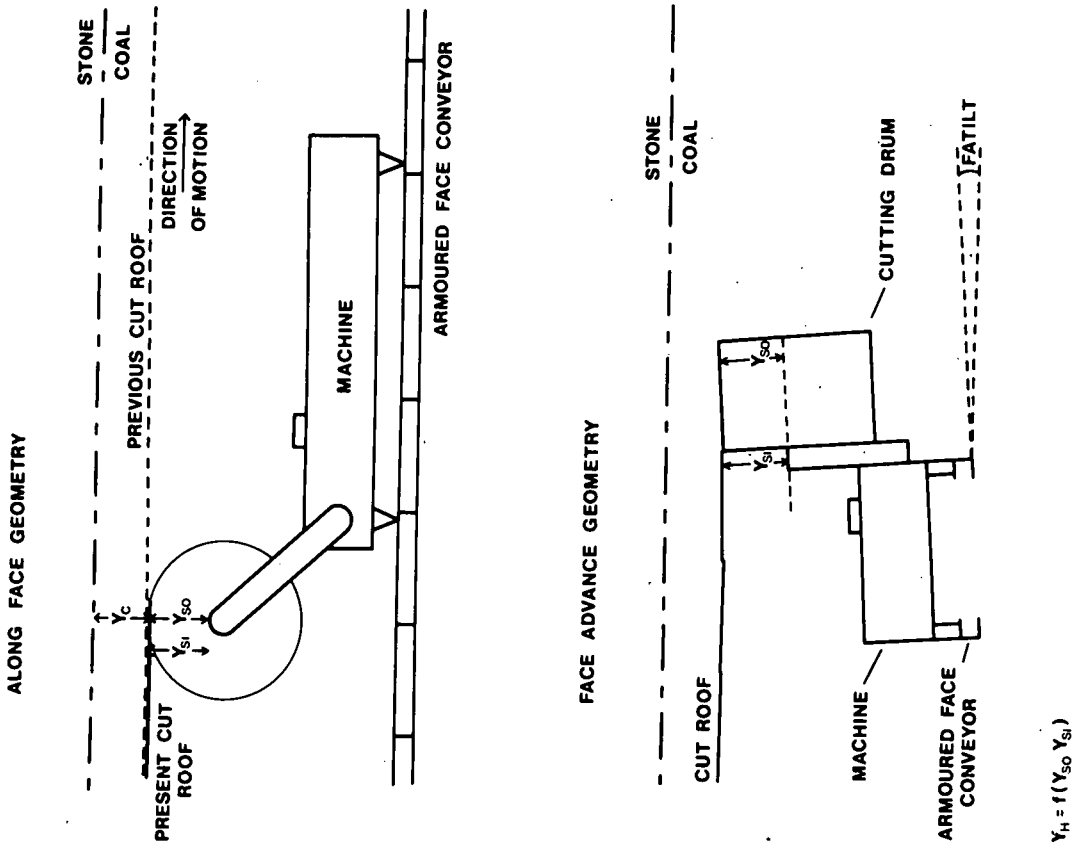


FIGURE 1 MACHINE GEOMETRY

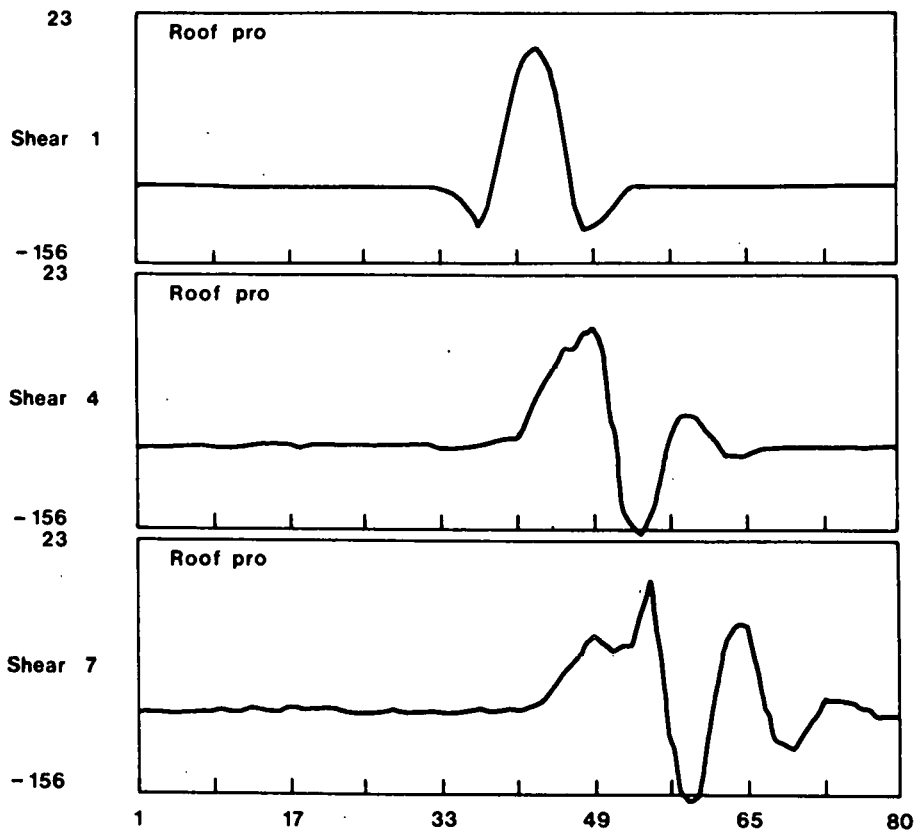
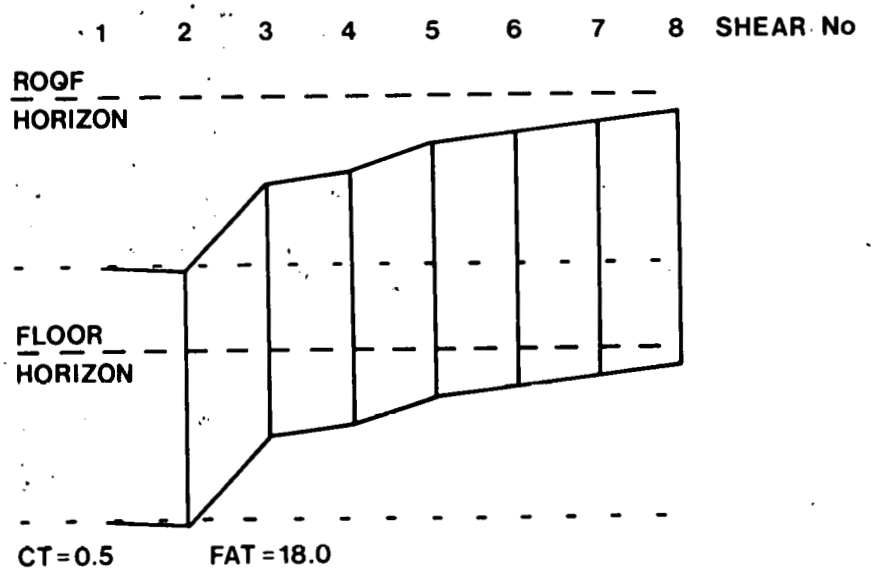
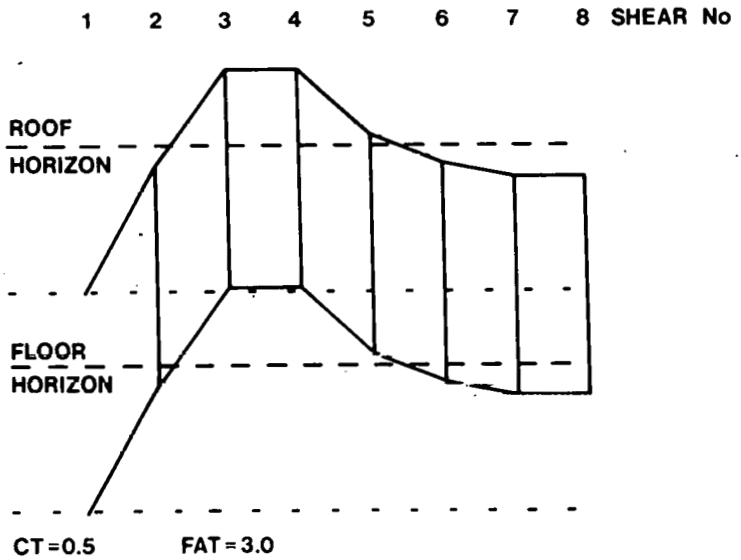


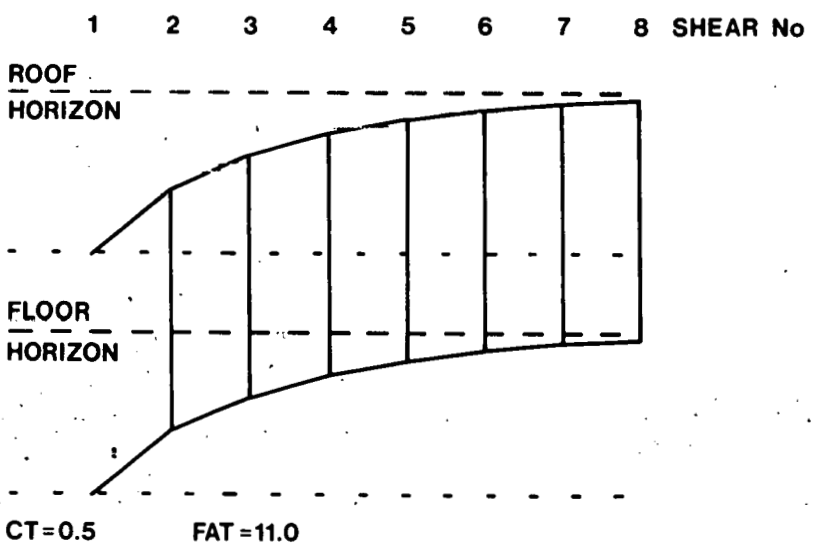
FIGURE 2 SIMULATED ROOF PROFILES PRODUCED BY DIGITALLY CONTROLLED SHEARER WITH NO ROOF FOLLOWER CONTROL ACTION



a



b



c

FIGURE 3 SIMULATED RESPONSES TO A HORIZON CHANGE IN THE FACE ADVANCE DIRECTION

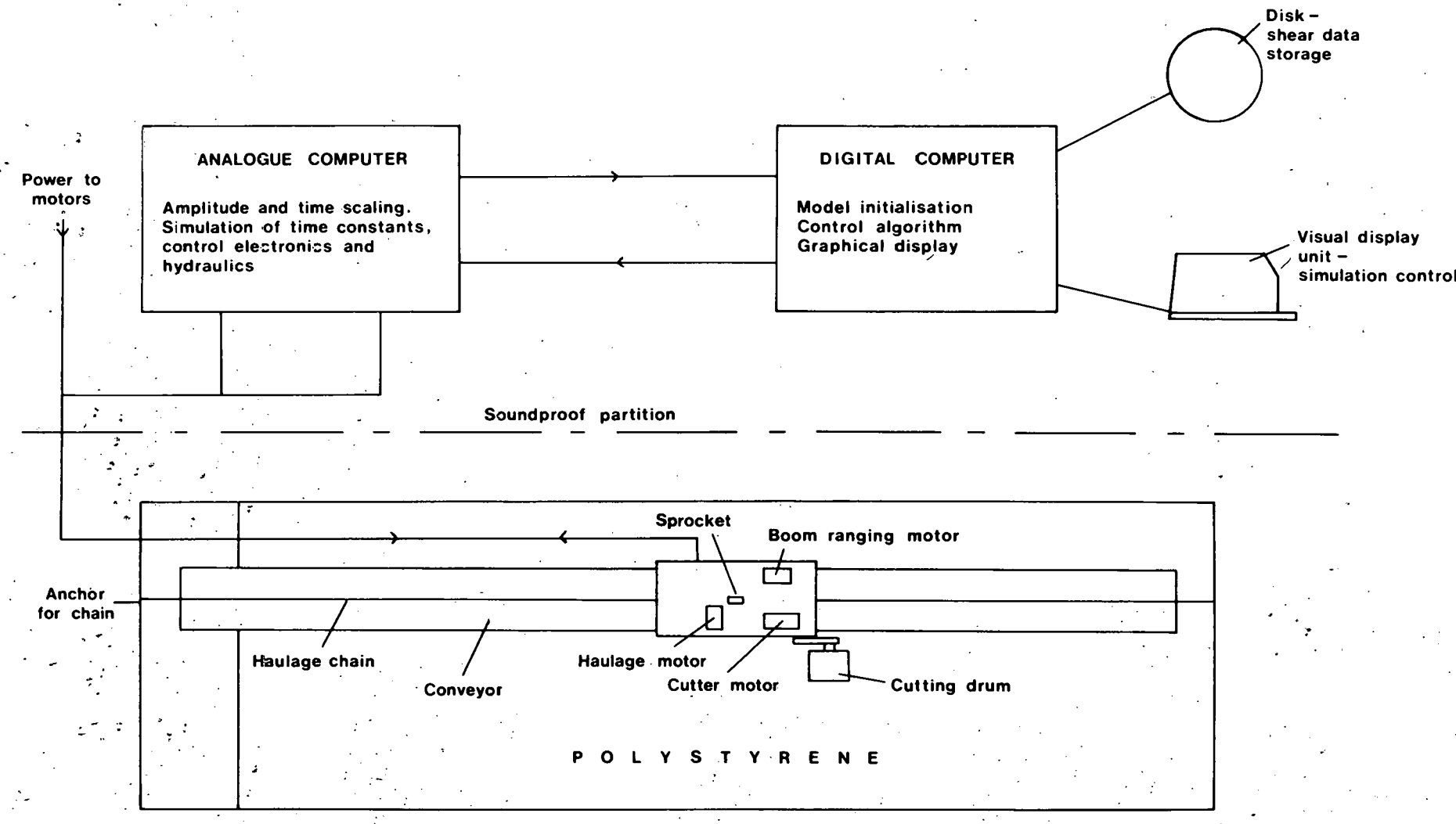
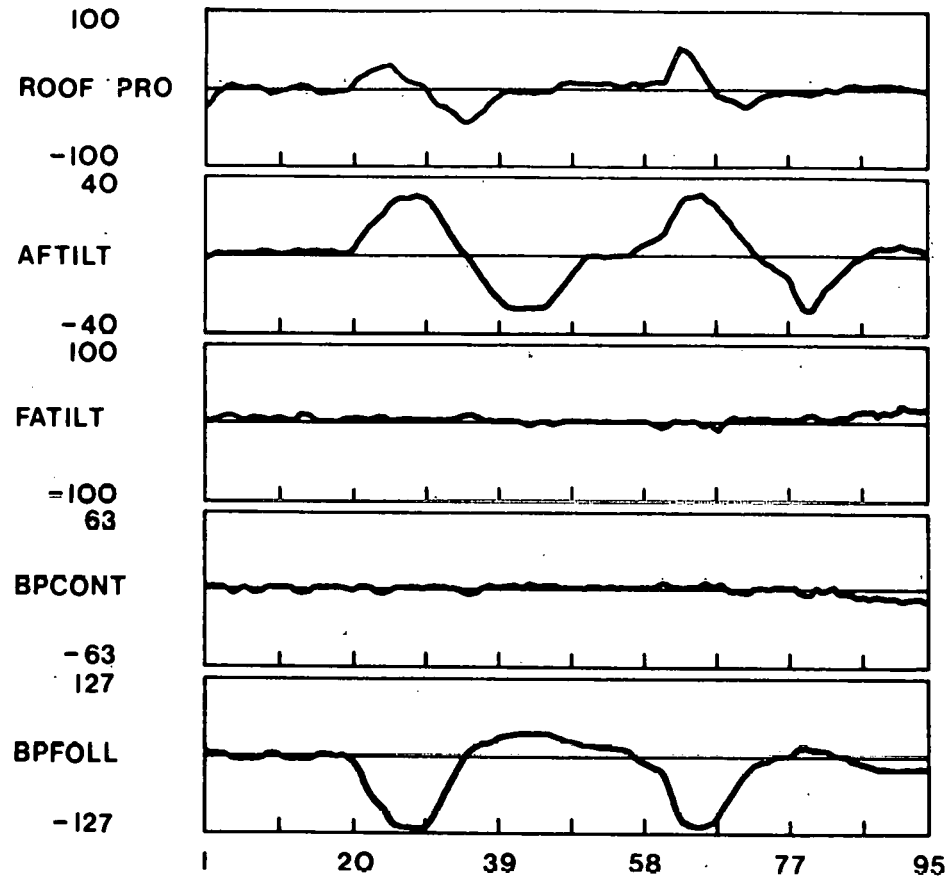


FIGURE 4 GENERAL ARRANGEMENT OF SIMULATION EQUIPMENT



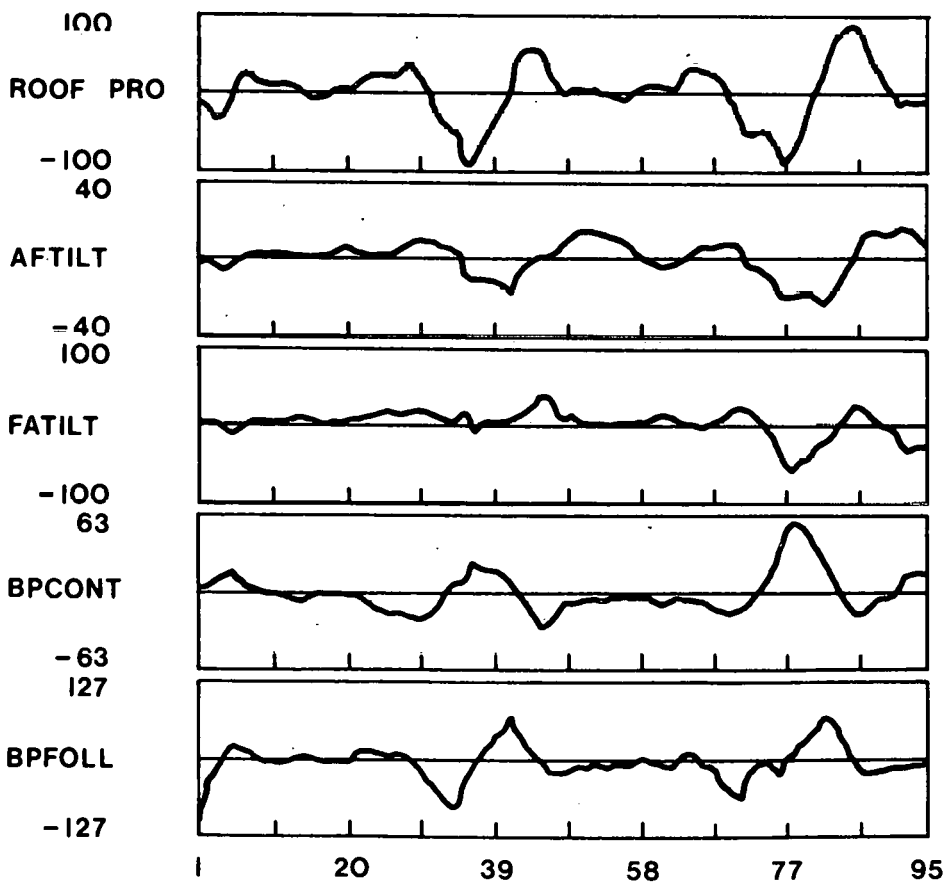


FIGURE 5 PARAMETERS MONITORED FOR TRIAL 1

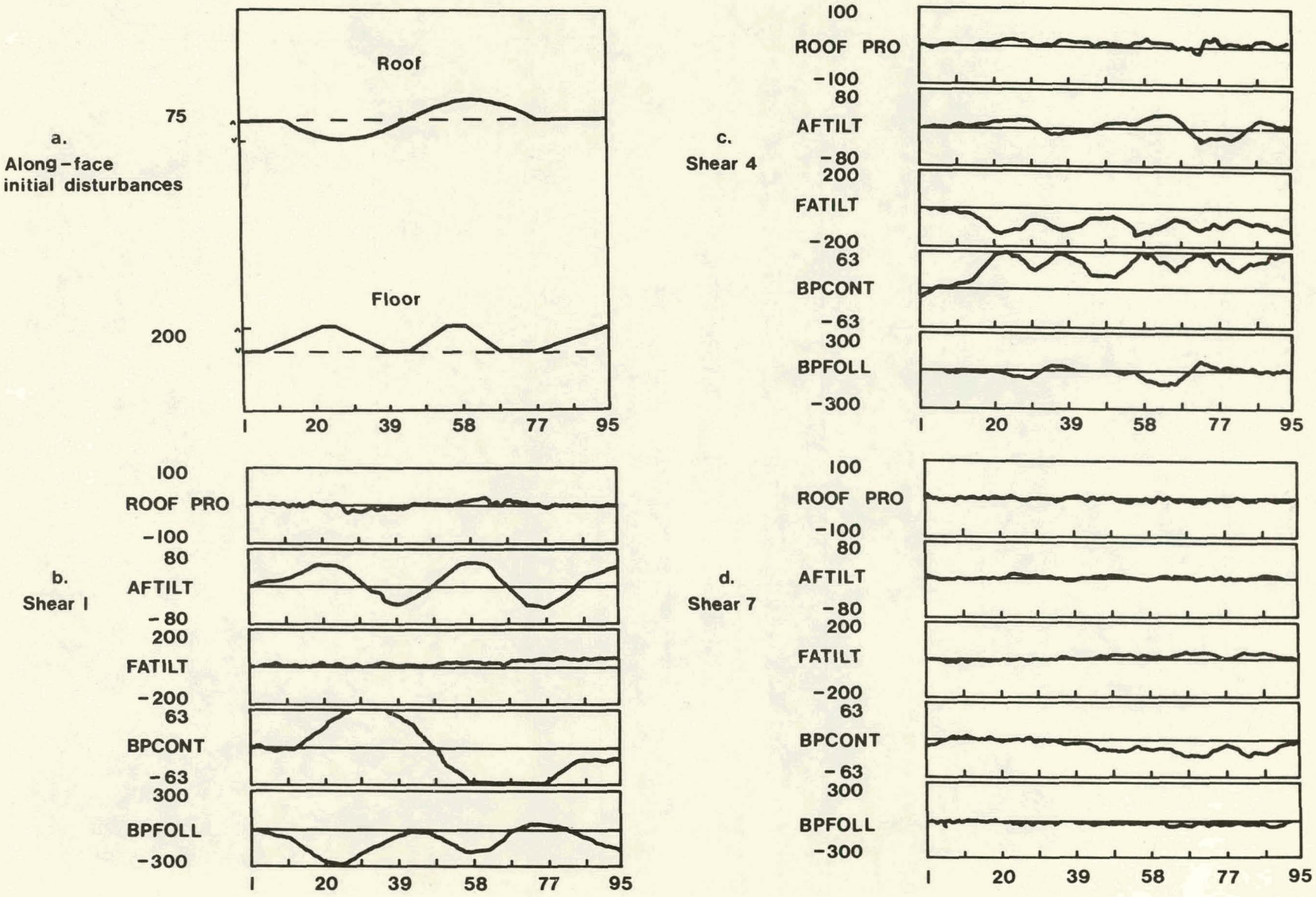


FIGURE 6 PARAMETERS MONITORED FOR TRIAL 2

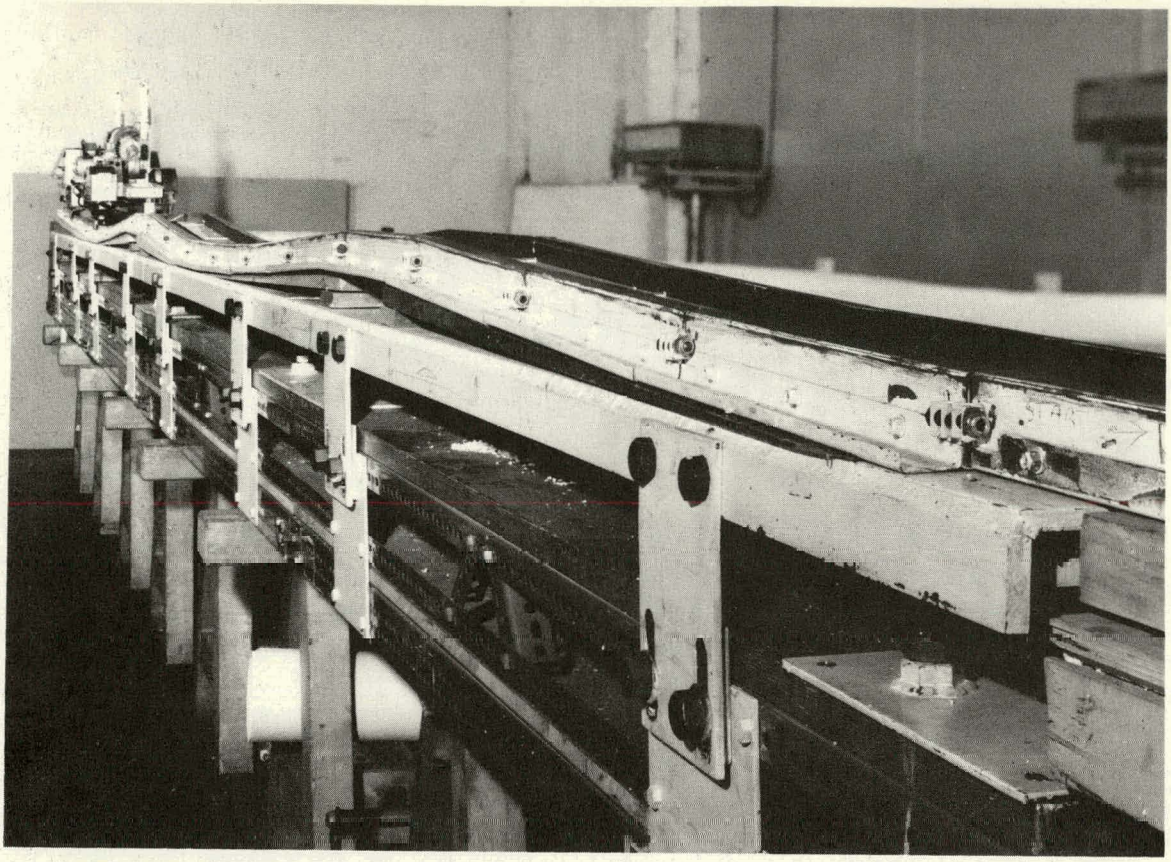


FIGURE 7a INITIAL CONVEYOR DISTURBANCE

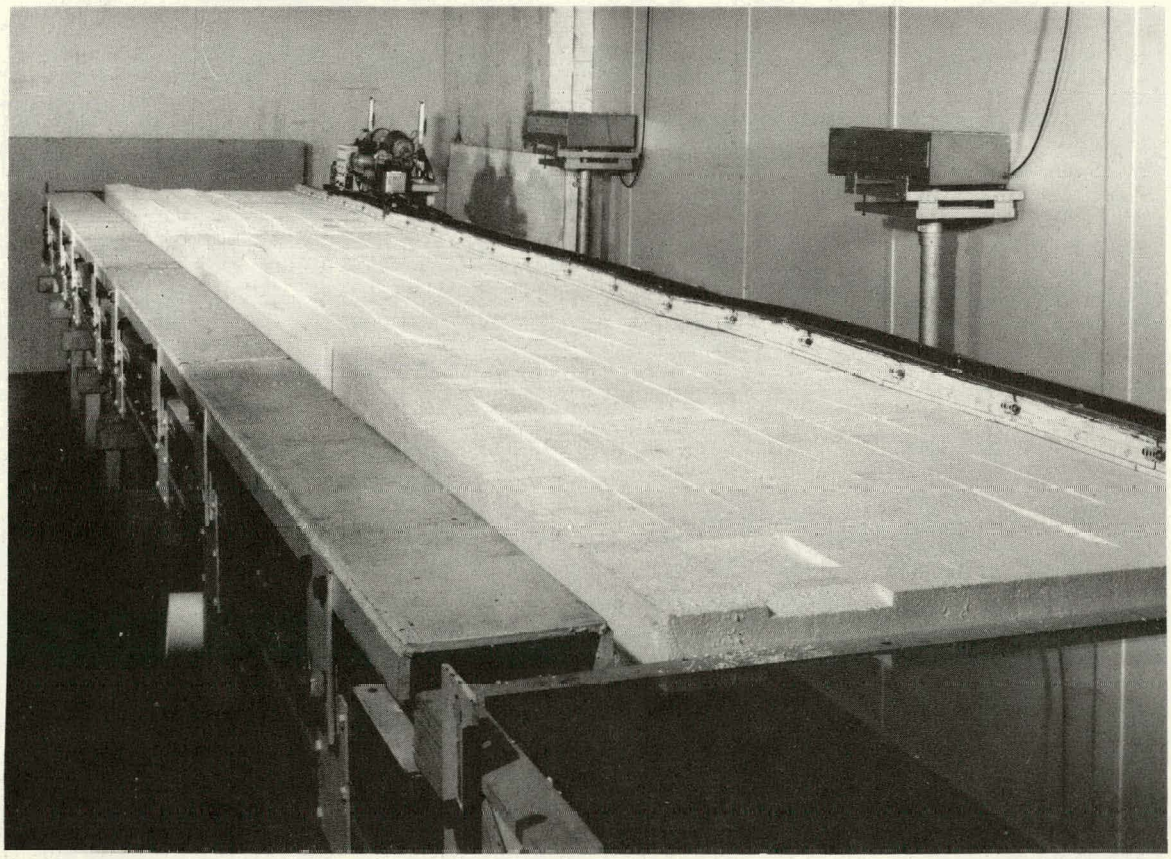


FIGURE 7b CUT FLOOR AFTER TRIAL 2

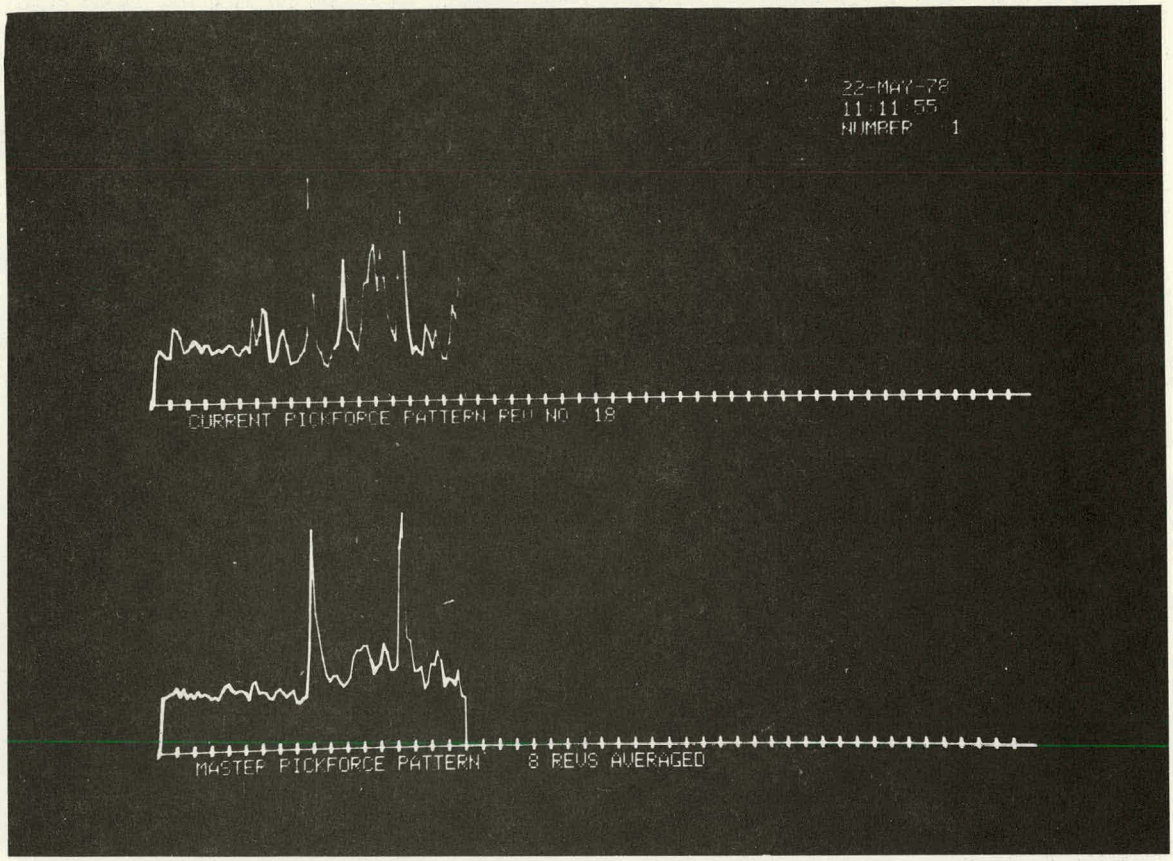


FIGURE 8a CURRENT AND MASTER FORCE PATTERNS

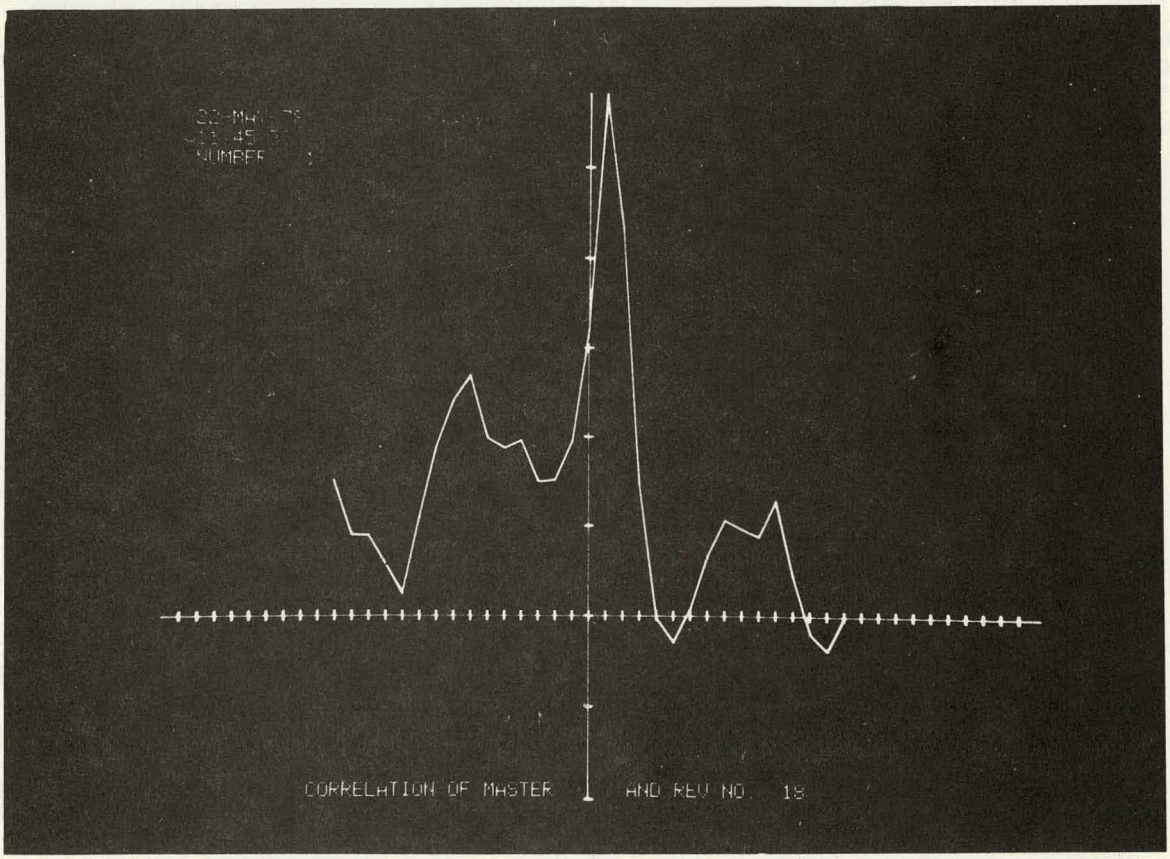


FIGURE 8b CORRELATION OF FORCE PATTERNS

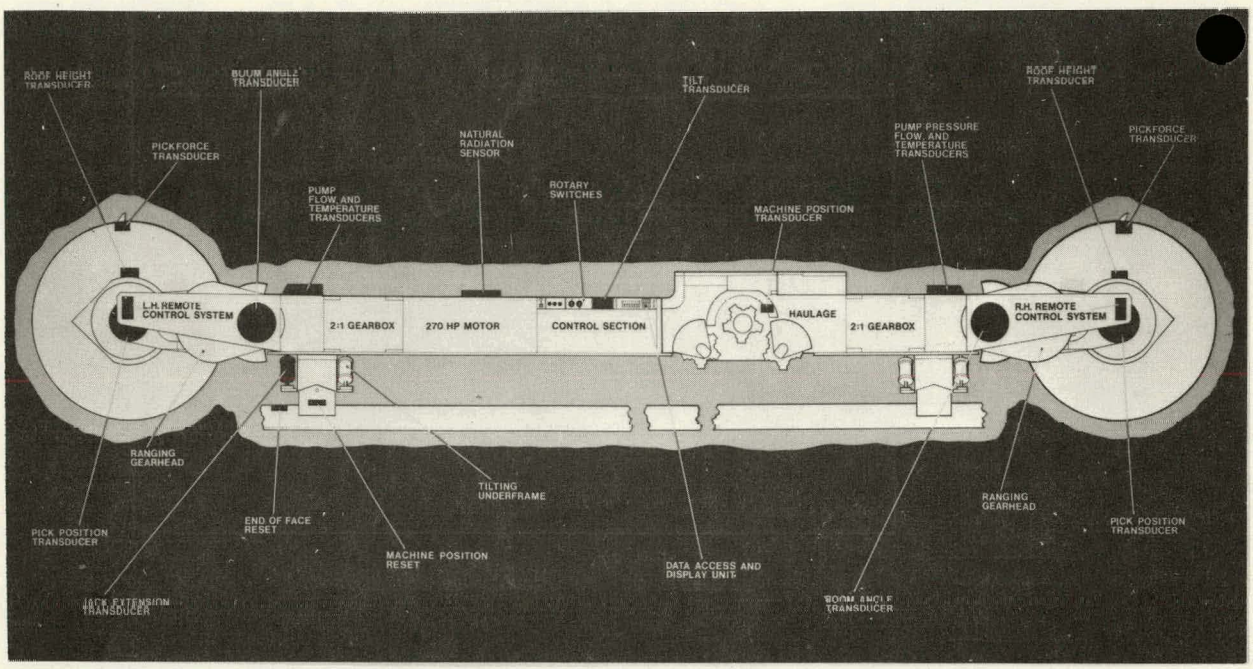


FIGURE 9 ARRANGEMENT OF TRANSDUCERS FOR AUTOMATIC STEERING SHOWN FOR A DOUBLE-ENDED RANGING DRUM SHEARER

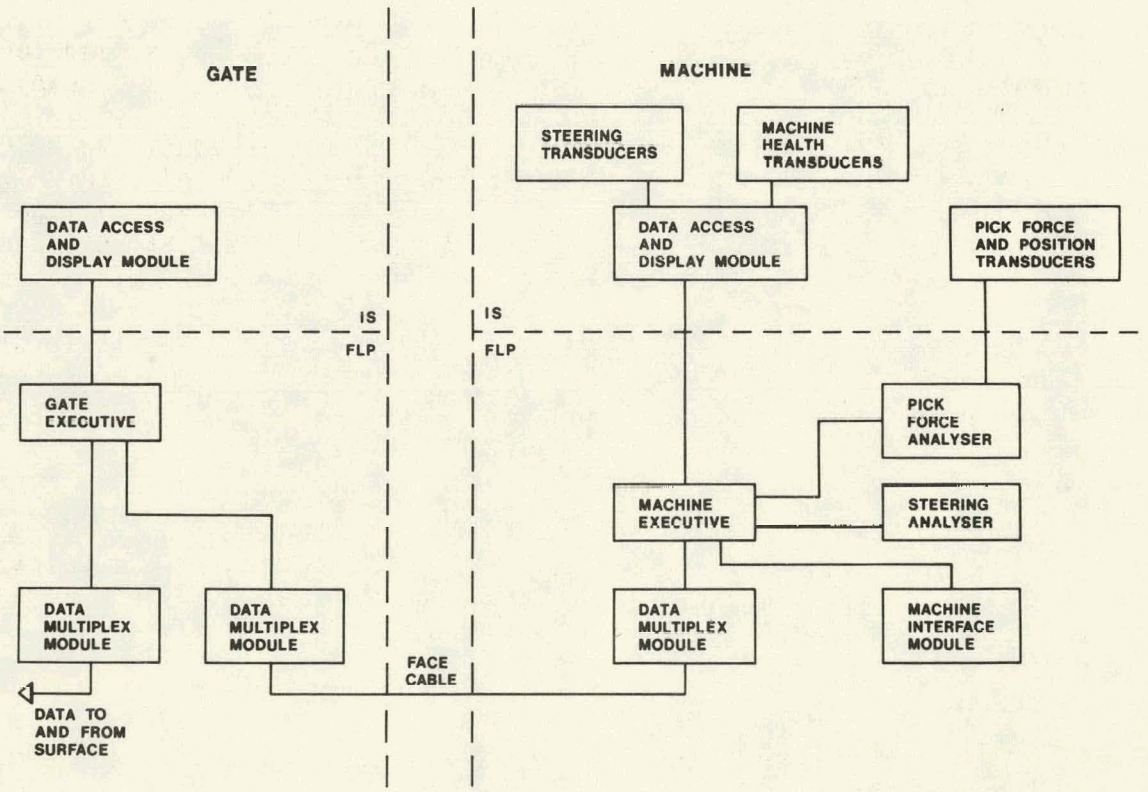


FIGURE 10 SCHEMATIC OF STEERING SYSTEM UNDERGROUND EQUIPMENT

A DISTRIBUTED MICROPROCESSOR MONITORING & CONTROL SYSTEM FOR COAL MINES

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ABSTRACT

A description is given of a distributed microprocessor system being designed under a USBM contract for use in underground coal mines. This system is based primarily on microprocessor technology and is useful for small as well as large mining applications. The system is designed to handle both environmental safety monitoring and for executing control and handling communications for other systems that will be installed in coal mines in the future. These include but are not limited to remote control, mine automation, production management and mine maintenance.

INTRODUCTION

Historically, West Virginia University has been doing design, implementation, and testing of a minicomputer-based mine-wide monitoring system. This work has been documented in past proceedings and in final project reports for the U.S. Bureau of Mines (1,2...). The minicomputer-based system utilizes a central minicomputer to pole approximately 13 coal mine sections and retrieve 20 points of data from each. The minicomputer then converts the retrieved voltage data to engineering units, checks each value against limit value, stores data into archival storage, and alarms based upon the limit comparison.

The intent of the microprocessor-based system being presently designed is to make the mine-wide monitoring system in general very modular and to distribute the system "intelligence" into each monitoring section for use by those on the section.

SYSTEM CRITERIA

The overall mine system was designed so that specific characteristics such as architecture, communications protocol, etc., were chosen in compliance with the following criteria.

This work has been funded by the U.S. Bureau of Mines under Contract No. H0144114.

Versatility

When installed, the system may be easily changed or altered when necessary. Versatility will exist for both hardware and software. In addition, movement of hardware in the system will require a minimum of software changes.

Expandibility

An operating mine system will be capable of being easily expanded as the user requires more mining sections and as new developments are applied in the mine complex. Although an upper bound exists on any installation, the fundamental system design will provide for any given system to be a subset of any new larger system. Any given system will be capable of expanding with no major retrofits of hardware, software, or personnel training. If new sensors or actuators become available, they will be easily added to the system without affecting other parts of the system operation.

Loopback and Stand Alone Areas

The system will be able to operate with loopback circuits if the primary communication circuit is broken. If all communication with central control is lost, the stations in the isolated areas will continue to perform their functions. Where control functions are involved in any communication failures, the controlled process should be automatically shut down or the control system automatically switched to a contingency mode.

Loss of Primary Power

If primary power is turned off to any station in the system, the remaining parts of the system will continue to operate. The affected station(s) should operate in their "power down" mode.

Monitoring and Supervisory Control

The system will be capable of monitoring and controlling both binary (on-off) and analog functions. The system possesses the ability to originate a control command at one station in the system and be executed at another station.

Central Control

The system is being designed to accomodate various levels of complexity including no central station, simple central read out and control station, and sophisticated computer based central control.

Multifunction Capability

The system should be able to serve the monitoring and control needs for the following types of operations:

1. environmental
2. remote control
3. production management
4. maintenance
5. automation

System Security

The system shall be made secure from unauthorized use by means of features such as passwords and other hardware/software checks.

Reliability

The system should be as reliable as possible consistent with the costs involved. System failures should yield a safe condition.

Cost

The design involves features that minimize initial installation costs and provide cost effective operation and maintenance on a continuing basis.

Now that the overall criteria of the system have been itemized, let us now consider the installation.

STATIONS

There exist five general types of stations in such a system. These are the Simple Monitor, the Portable Remote Readout, the Multifunction Remote Station, the Remote Control Subsystem and the Central Station. These are shown in Figure 1.

Simple Monitor (SM)

For the entry level system, the section foreman should be the first who should be informed of a problem on his section. The entry level monitor or simple monitor is shown in Figure 2.

The simple monitor is designed to accomodate the minimal monitoring needs for one mine section. This should be the kind of station a mine operator may first try as his initial monitoring attempt. Its size and cost were kept as small as possible while still allowing later expansion into larger systems.

This station accomodates at least four analog inputs and eight binary (on-off) inputs. Upon request from the bus or upon command of an internal program, this station will be able to perform at least all of the following functions:

1. read analog points and convert to engineering units
2. read binary inputs
3. read threshold levels

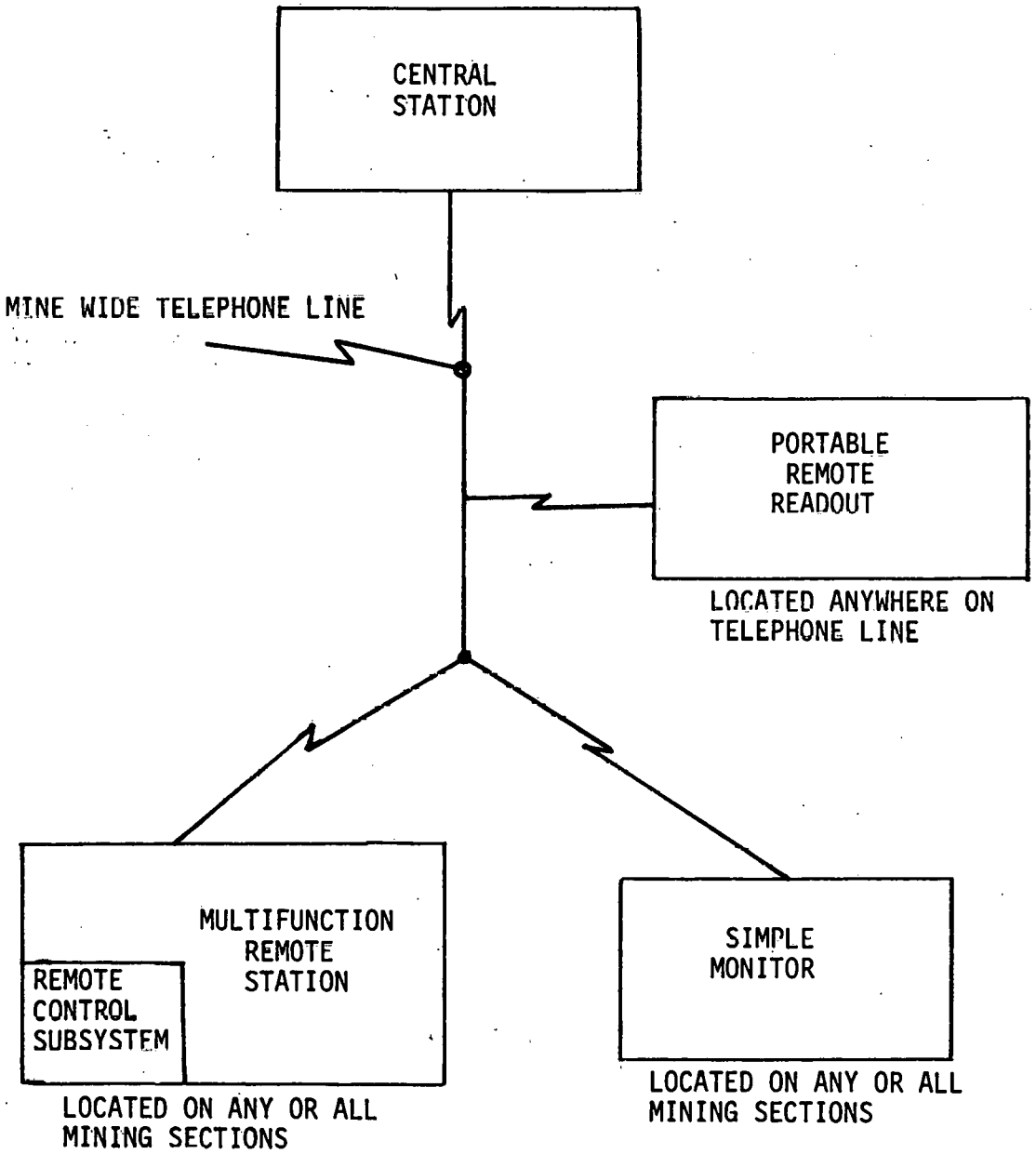


FIGURE 1: TYPES OF STATIONS

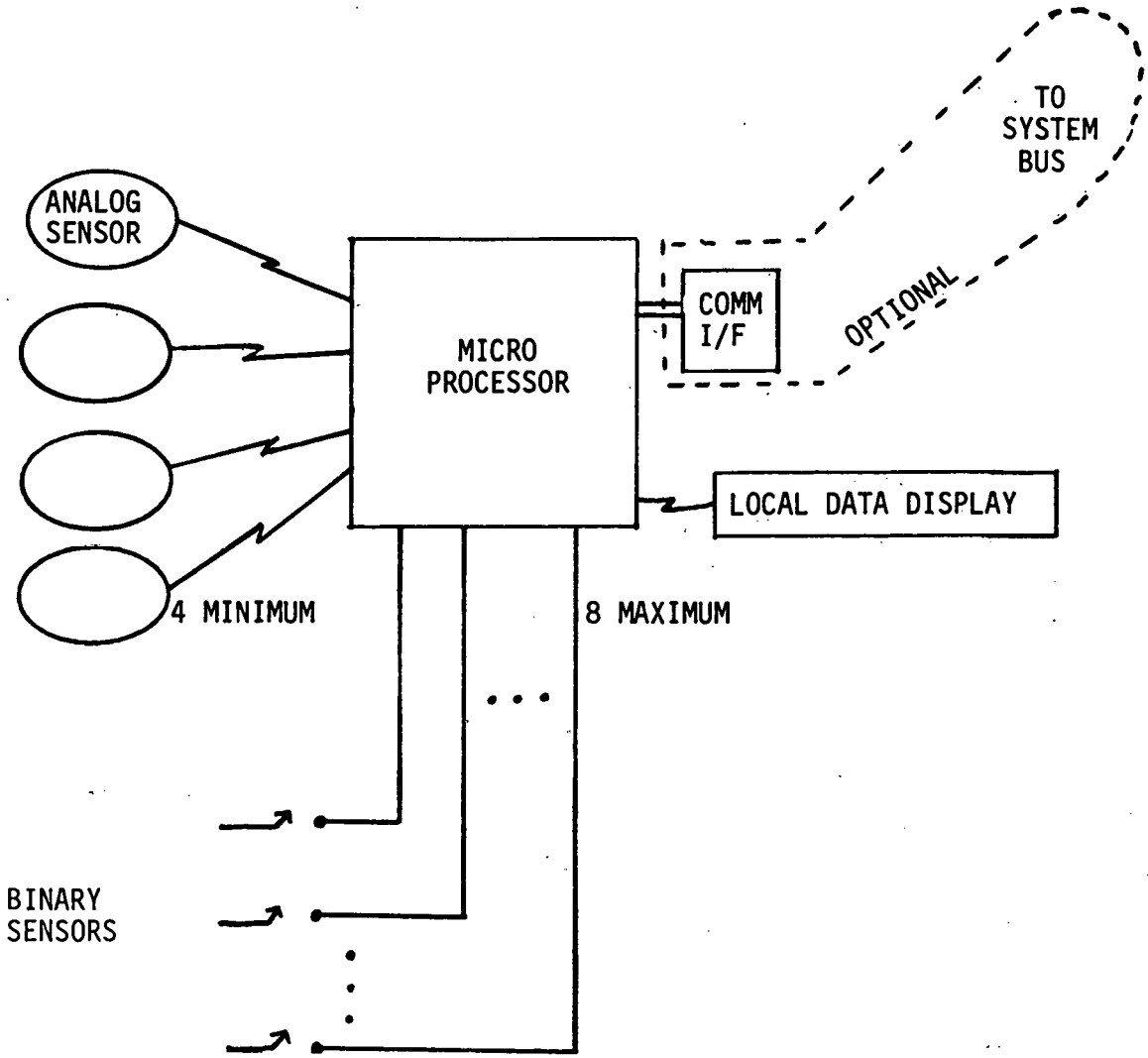


FIGURE 2: SIMPLE MONITOR

4. transmit analog and/or binary data onto bus
5. generate exception reports and transmit as required, and
6. generate local display.

The simple monitor as presently operating in the laboratory is microprocessor based (INTEL 8080) reading eight channels of analog data and displaying via a serial terminal. An alphanumeric display that is also 8080 based is operating separately. New software is being written in FORTRAN for the simple monitor and integration of the display and packaging will be undertaken in the near future.

Portable Remote Readout (PRR)

This station will ultimately be a handheld portable unit which can be attached to the system bus and provide passive readout of any data on the bus. This unit will monitor the bus and "pick-off" the desired information and process it for display. This station will display only one analog point and/or eight binary points (off/on data) at one time for one remote monitoring station.

Let one consider now the addition of this portable remote readout to the system (see Figure 3). This figure shows the first stage of system usage. The simple monitor is still as given in Figure 2 with its own local display. The PRR may now be attached to the telephone line attached to the simple monitor and monitor the data being continually broadcast by the SM.

The next step up in complexity is to obviously add more simple monitors and more PRR's. This presents a bus control problem (or telephone line control if one prefers) that is solved by designating each SM as "bus master" if no bus activity occurs within X seconds. SM No. 1 takes control immediately. SM No. 2 takes over if there is no activity in 15 seconds. SM No. 3 takes over if there is no activity in 30 seconds. This is continued so that if SM No. 1 should fail, for example, or if the telephone line should be broken, the system parts that are still connected still operate as independent sectors.

Note that Figure 4 shows one such configuration and Figure 5 shows another. These or any other configurations shall operate.

System Bus

A universal or global bus may exist throughout the whole system to which all stations connect. The bus will be a two-wire, balanced line circuit that accomodates full duplex, asychronous, serial communication. A bit oriented protocol will be used. In general, rate of data transmisson on the bus is not specified and can be chosen for each application. For the purpose of this demonstration, modems will be used that operate full duplex at 300 baud within a standard voice grade channel.

In a large system, the central station will normally be the "bus master". However, in the event the station fails, communications with

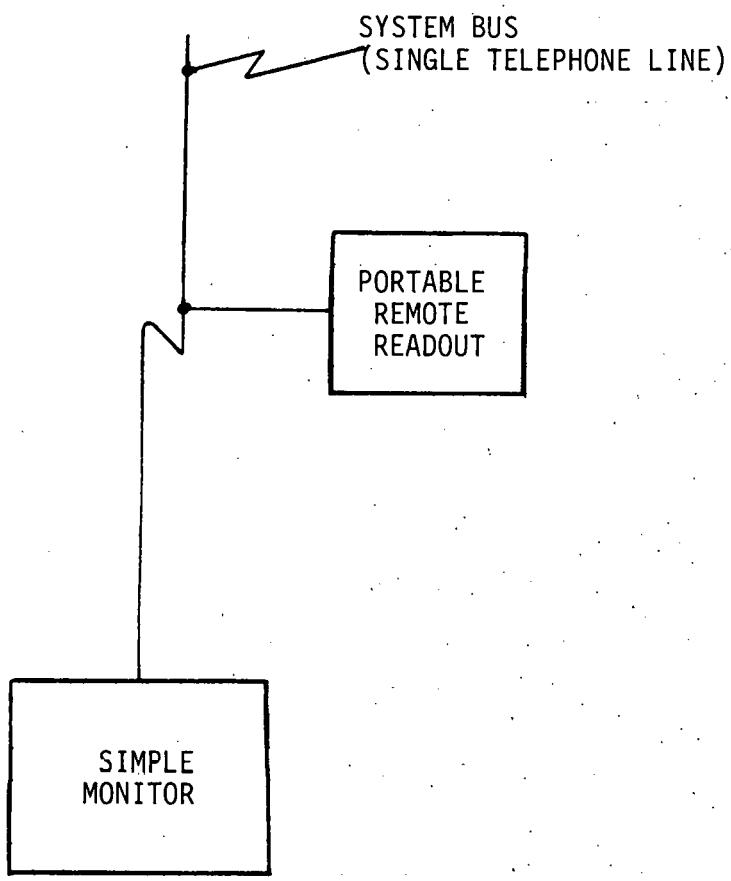


FIGURE 3. SIMPLE INTEGRATION MODE 1

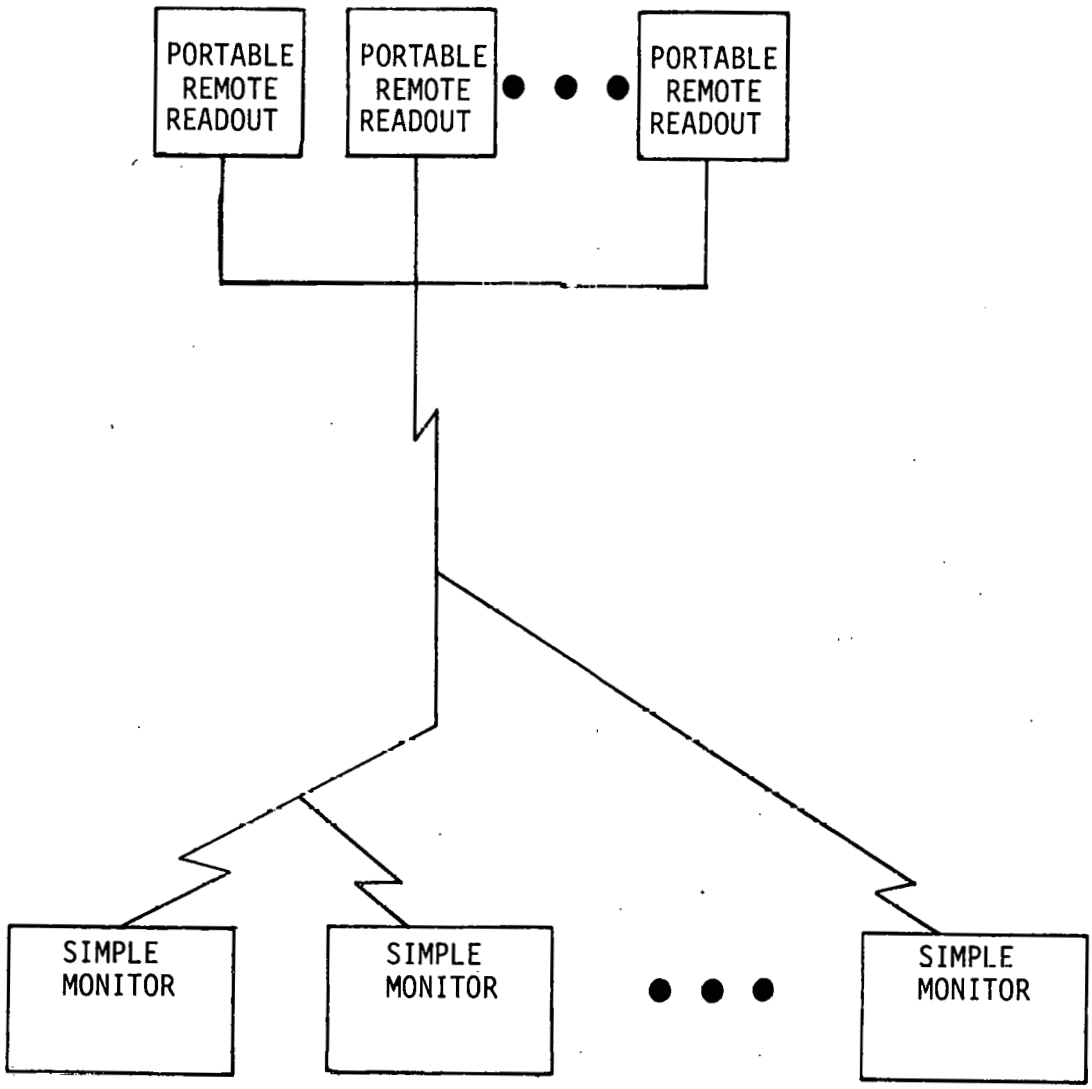


FIGURE 4. INTEGRATION MODE 2 POSSIBILITY

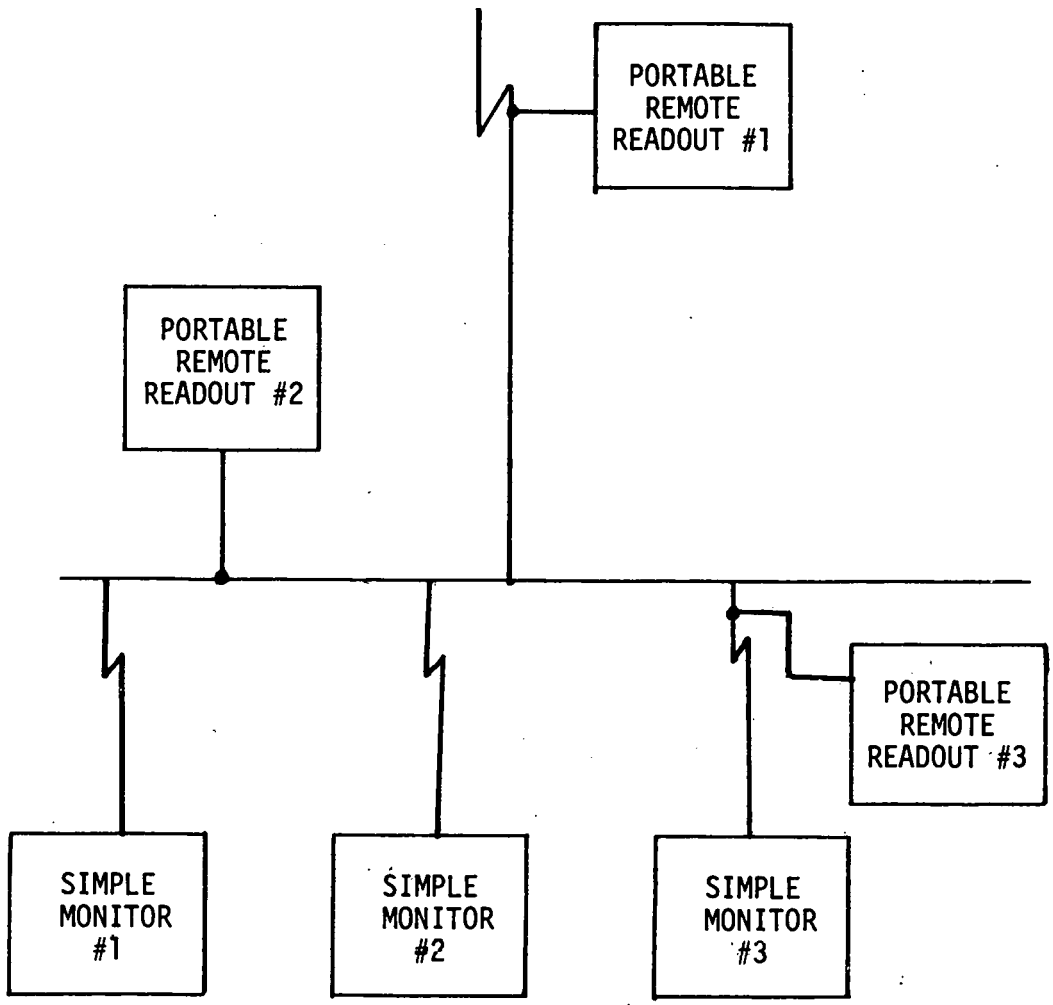


FIGURE 5: INTEGRATION MODE 3 POSSIBILITY

central is lost, or no central station is used, provisions will be made for proper operation of the remaining bus or bus segments as described above.

If possible, provisions shall be made to prevent a modem failure from "locking up" the entire bus. The bus will also be compatible to providing redundant "loopback" paths to the central station.

Multifunction Remote Station (MFR)

Let us now consider a larger system with more and more processors being added to a mining section (see Figure 6).

This station is the heart of the modular mine monitoring and control system. The multifunction feature is intended to provide a variety of possible configurations for various applications. In general, the multifunction remote station will be configured as shown in Figure 6. Each subsystem will be identified with each of the particular types of applications, i.e., 1) environmental, 2) remote control, 3) production management, 4) maintenance, and 5) automation. Each subsystem will be capable of communicating directly to the system bus through the required modem. However, in some cases several subsystems may be required to operate in the same vicinity and require communication directly between the subsystems. In the latter case, several subsystems can be tied together on a common subsystem bus which operates under the management of a subsystem controller which also manages communications with the system bus. This arrangement permits high speed communication between subsystems without interference to the main system bus.

Each subsystem will be designed for a specific type of application. Therefore, a separate control and display panel will be used for each. However, it should be noted that several circuit modules may be used in more than one subsystem.

Environmental Subsystem - This component provides for monitoring environmental related parameters and the corresponding data processing for local alarm and output requirements. The following apply:

1. designed to accomodate a minimum of eight analog and sixteen binary inputs,
2. analog inputs may have the following ranges (selectable at construction) - 0-1, 0-5, 0-10 VDC and 0-1, 0-2 mA,
3. binary inputs are TTL levels with provisions for interface as needed,
4. sampling speed will be under system or program control, but should never be faster than 100 kHz,
5. internal programs will convert samples to engineering units as required, check alarm levels, issue alarm information, generate exception reports, and process data for local readout and control,

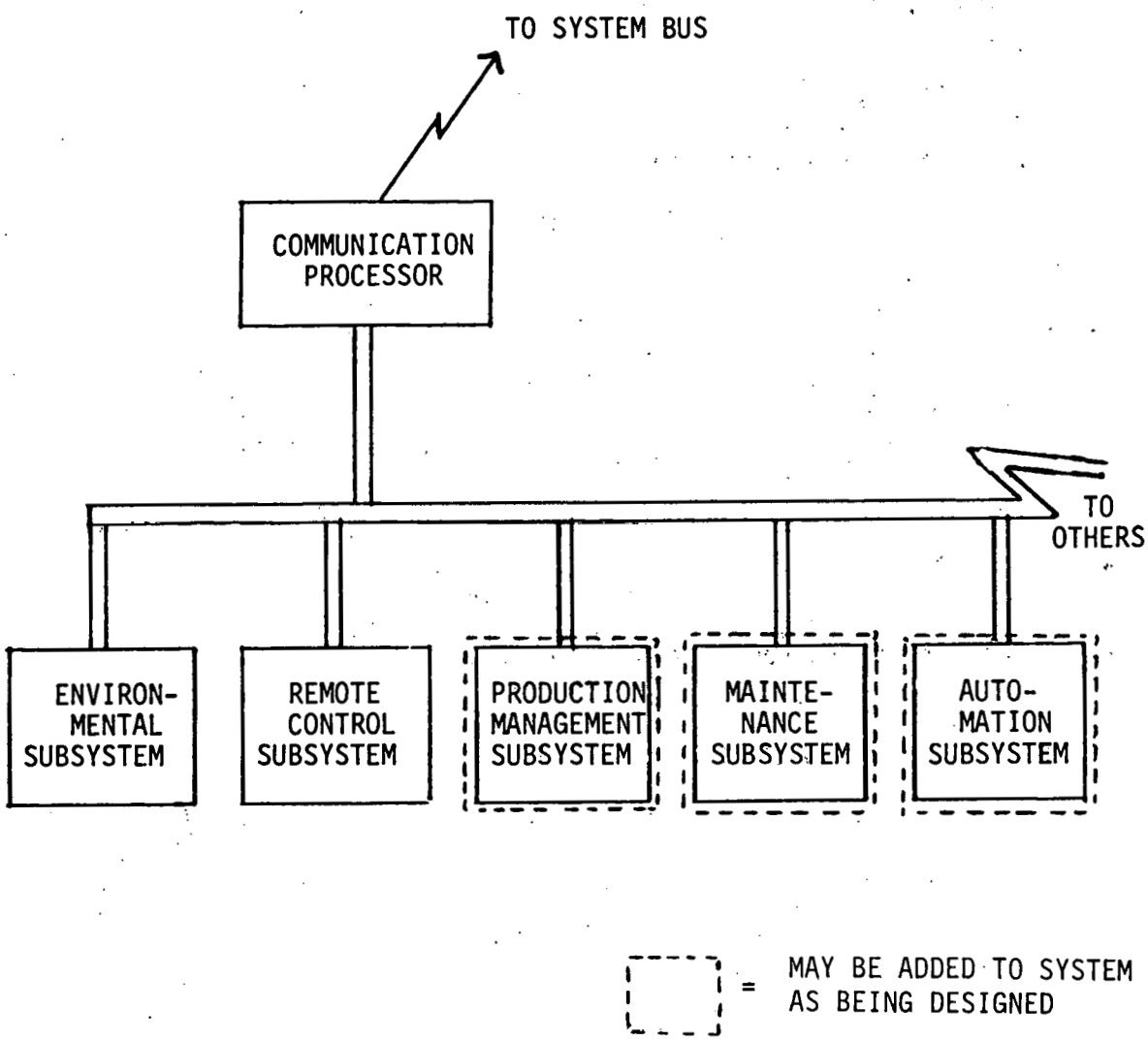


FIGURE 6: MULTIFUNCTION REMOTE STATIONS

6. will possess its own control and data display panel which will clearly show the alarm status of all environmental sensors, and
7. design will include option for A-to-D conversion to be at the sensor as well as at the subsystem.

It is intended that this subsystem be a simple hardware/software upgrade of the simple monitor.

Remote Control Subsystem - This station may be used specifically for executing control of external functions based on signals received from the system bus or generated locally. These features in the very simplest case may be a part of the environmental processor.

One should note that the monitoring requirements for a simple remote control may be met by the environmental monitoring subsystem. This, however, will depend upon the application. Nevertheless, data paths exist for the remote control to read monitored data from the environmental subsystem.

Other Subsystems - Provisions are also being made for other subsystems which can be dedicated to the following uses: production management, maintenance, automation, and others. It should be noted that each of these system types may not be unique. For example, several different versions of the automation subsystem may be required for systems of different manufacturers and/or types of mining systems.

No specific functional requirements are set forth for these subsystems. However, it is anticipated each subsystem may contain many of the same modules contained in other parts of the system. Thus, each subsystem will be capable of inputting and outputting information according to its own requirements as well as interacting with other parts of the system. By this approach each subsystem can possess a high degree of independence from the rest of the system and yet remain competitive.

It should also be recognized that a subsystem may be physically separated from the multifunction station location through a serial modem. This would permit control equipment on an automatic machine to be tied to the primary system via two wire circuit and an appropriately designed modem interface unit. This approach gives maximum flexibility for the developing area of automation which cannot be well defined now. However, it does permit the interaction of such systems to other subsystems that will no doubt be required for adequate safety and management control.

Central Station

This station may range from a collection of one or more PRR's to a minicomputer or even a collection of minicomputers. In between these extremes may also be a microprocessor and/or a collection of microprocessors.

For small mining operations or in the early stages of developing a monitoring system at a mine, it will be desirable to have available a "simple" central station that provides a minimum of features for a small number of remote stations.

The simple central station will provide for at least the following functions:

1. read out one analog point on a digital display in engineering units
2. show status of all binary points in system
3. provide for alarms by aural and visual means (from binary points)
4. poll status of all binary points in system on regular basis at least once each minute
5. support at least four remote stations, and
6. optional provision for executing on-off remote control at each station from commands at central station.

In the larger system as shown in Figure 7, the central station will serve as bus master, receive data from the system and issue commands to the system. It will be capable of operating in an automatic mode under program control or with human intervention. High level software will be used to minimize problems with changes in software and to meet a wide variety of needs.

Communications Processor (COMM)

Since the system is using a new more general communication protocol, the present version uses an independent microprocessor (8080) to preprocess and handle all communication to/from each station.

It is anticipated that within two years, a CMOS 8080 or 8085 will become available that will allow the communications processors to be powered by batteries for long time periods. The communication processor may then monitor the mine-wide system bus for commands or requests to its multifunction remote. If the proper command were received, the COMM would have the ability, during time when there is no primary section power, to turn on the environemntal subsystem and operate it from secondary power (batteries). The environmental subsystem would then collect the requested data and send it to the communications processor via the section bus. The communication processor would then relay the results to the central station and turn off the environmental subsystem to save batteries.

GENERAL EQUIPMENT SPECIFICATIONS

In general, primary power for all hardware is being designed to operate from 120 VAC circuits, but options for 220 and 440 VAC should be possible. Secondary power will be from batteries.

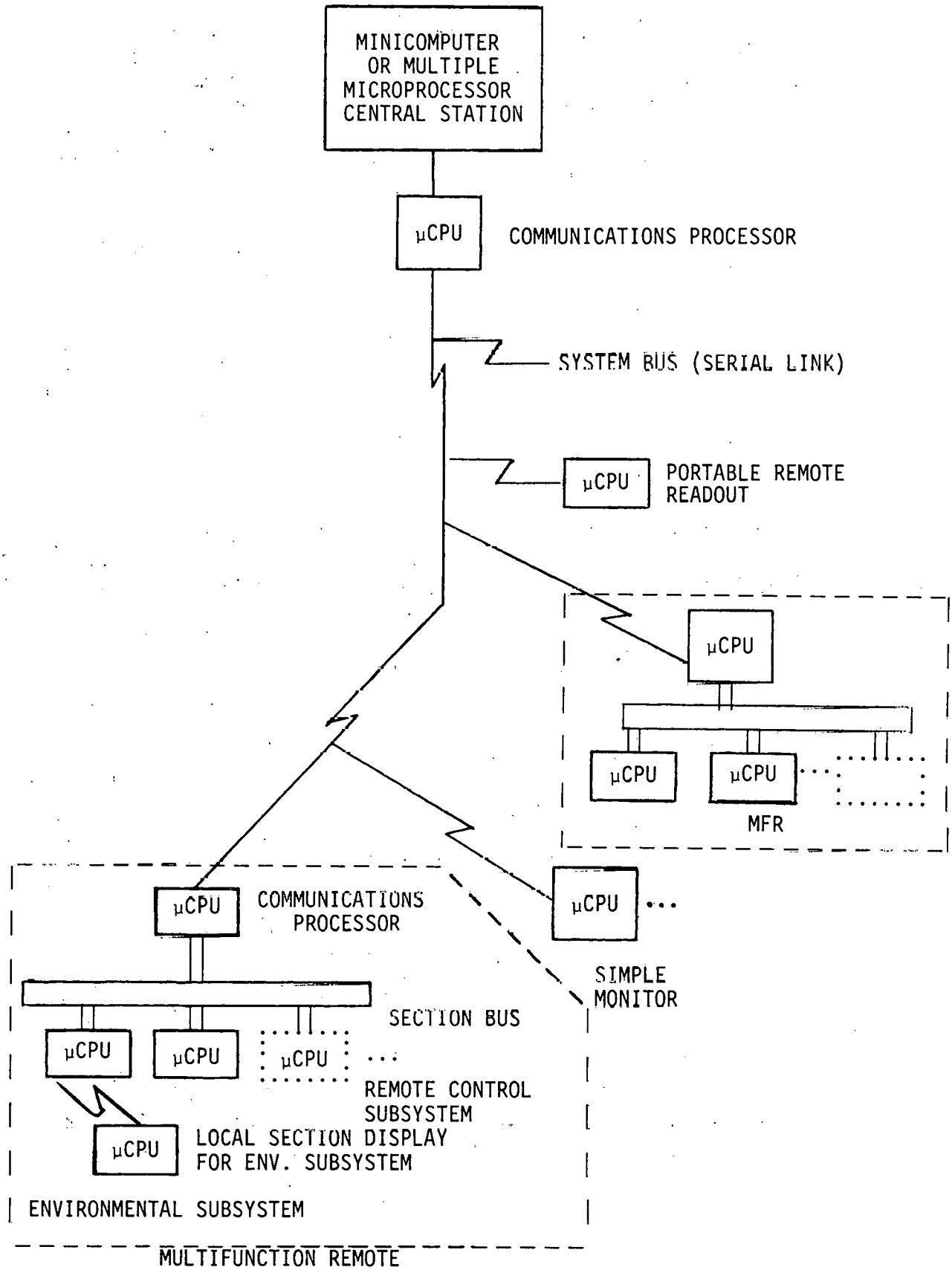


FIGURE 7. MINE WIDE DISTRIBUTED MONITOR AND CONTROL SYSTEM

All electrical circuits are being designed to be intrinsically safe when the primary power is off and equipment is operating from the secondary power source. The primary power will normally be located in "fresh-air" and therefore not require an explosion proof enclosure.

In general, the design goal will be to meet the environmental specifications set forth in Appendix B of Reference 8.

Each station will be electrically self-testing as much as possible. This should be relatively extensive considering the capability of the microprocessor.

The hardware and software is being designed to be modular in nature. That is, common "off-the-shelf" processors, memories, etc., are being used when possible so as to add to interchangeability among station parts. In addition, software is being written in a modular form so that changes and additions may be easily handled.

CONCLUSIONS

A general description has been given of a distributed microprocessor network presently being designed by WVU under U.S Bureau of Mines direction. This network is designed to allow an orderly low-cost entry and yet provide for orderly expansion of a mine-wide monitoring and control system. This system is microprocessor based and is shown in Figure 7 which indicates the various microprocessors involved.

It is anticipated that this concept will result in a low cost and long-lived system that will be highly desired by the coal industry.

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A STUDY FOR MONITORING

AND

SUPERVISORY CONTROL FOR AN

UNDERGROUND COAL MINE

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Electrical Engineer
Westmoreland Coal Company
Winding Gulf Division
Tams, West Virginia

ABSTRACT

We at Westmoreland Coal Company feel that continuous monitoring of various parameters in mines, particularly of the environment, could lead to increased safety and could, at the same time, be a definite aid to mine management. Continuous monitoring of a complete mine with all data available at a central location could show potential trouble developing, such as increased methane liberation, unexpected changes in ventilation, or the location of developing fires. With all of the information available at a decision-making site, intelligent, corrective action could then be taken.

Our project is a joint effort involving Westmoreland; Harris Corporation, Controls Division, Melbourne, Florida; and Pemco Corporation, Bluefield, Virginia.

Each organization is responsible for a portion of the project.

Westmoreland is providing the mine site, the Eccles #5 Mine at Eccles, West Virginia and the environmental transducers.

Harris Corporation is providing all the electronic equipment for the remote information processing equipment and the Computer Master Station.

Pemco Corporation is providing the power center, its transducers, ISE power supplies for the environmental transducers and the overall systems engineering.

We hope to determine the feasibility of continuously monitoring environmental conditions in the face area of a coal mine.

A STUDY FOR MONITORING AND SUPERVISORY CONTROL FOR AN UNDERGROUND COAL MINE

INTRODUCTION

The mining industry has been searching constantly for new methods of keeping the mine working areas safe, while at the same time producing coal as efficiently and economically as possible. Due to the often changing nature of the coal mine environment, it is humanly impossible to monitor and find meaningful patterns as quickly as is sometimes warranted. Electronic miniaturization and the subsequent widespread acceptance of microprocessor monitoring have paved the way for studies to be done with this type of data gathering equipment.

In June of 1976 Westmoreland Coal Company, Pemco Corporation, and Harris Controls Incorporated agreed jointly to test a microprocessor-based data gathering system in the Eccles No. 5 mine of Westmoreland Coal Company. Each organization was responsible for a portion of the project. Westmoreland Coal Company provided the experiment site, the Eccles No. 5 mine, and the various transducers. Harris Controls Incorporated built the master station, the remote station, and the information processing equipment. Pemco Corporation supplied the section power center, its transducers, and provided the system engineering for the project.

The monitoring system consists of a master station, a remote station, the remote environmental sensors, and all interconnecting cables.

The master station converts data received from the power center and the environmental transducers into signals that drive indication alarms and strip chart recorders. The devices may be monitored regularly to determine the mining face environment and equipment status.

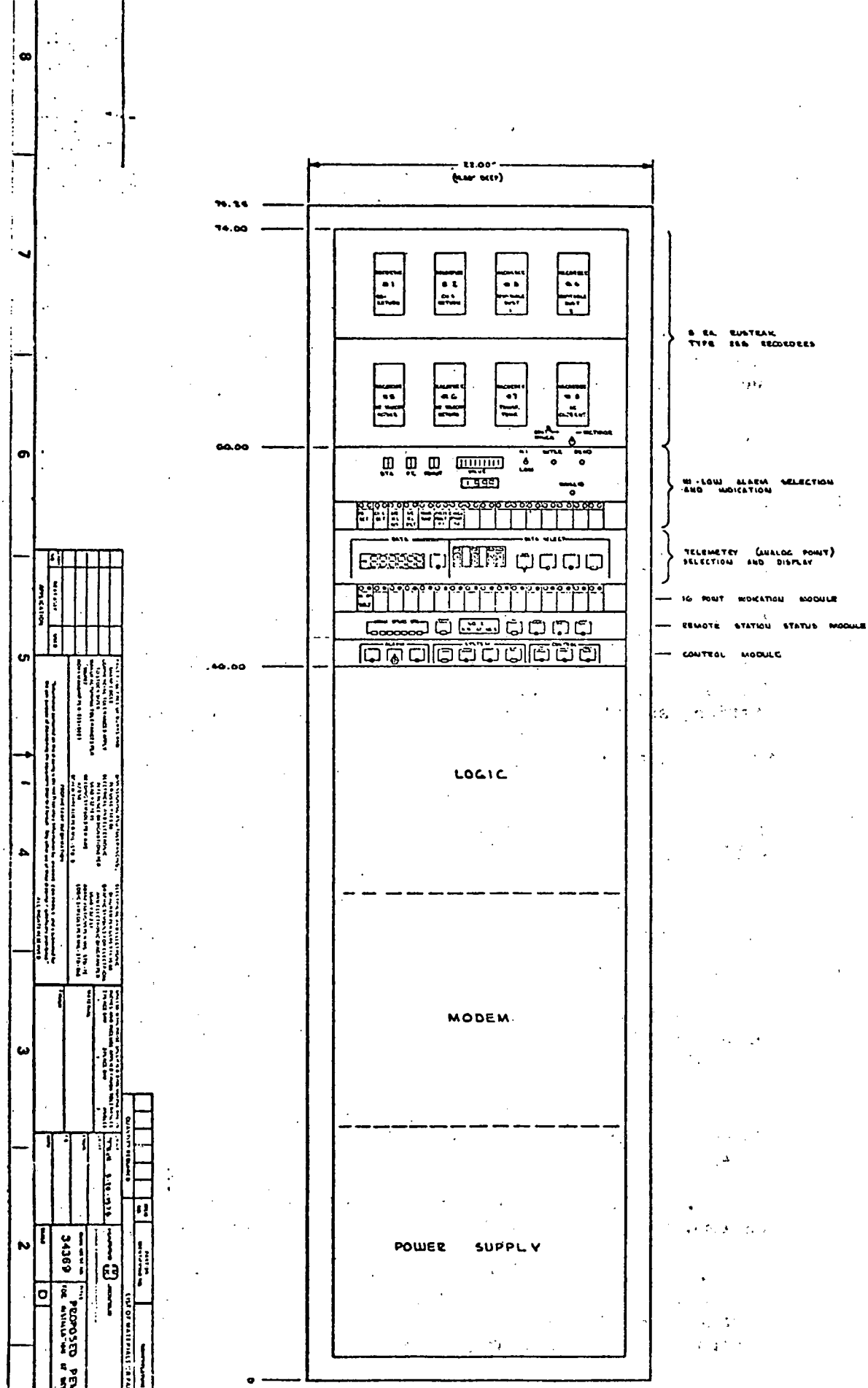
The master station consists of a 75 inch high equipment rack assembly. It contains five sub-assemblies that are listed below with their function (refer to Figure No. 1):

1. Power Supply - Converts 115 VAC available at the dispatcher's station into DC voltages required to power the remaining rack assembly equipment.

2. Modem - Provides an interface between the master station Logic equipment and the full duplex communication channel input.

3. Logic - Converts the Modem data input into signals that drive the indicators, alarms, readouts, and strip chart recorders. The Logic section also converts switch inputs from the control panel into data information that is processed and transmitted to the Power Center.

4. Control Panel - Contains lamp indicators, visual and audio alarms, digital displays, and selector switches that allow the operator to continuously monitor all remote data input points.


LOGIC
MODEM
POWER SUPPLY

5. Strip Chart Recorder - Provides a permanent record of the parameter being monitored. There is a strip chart for CO, CH_4, respitable dust 1 and 2, air velocity intake and return, power center transformer temperature, and either continuous miner current or rectifier current.

REMOTE STATION DESCRIPTION

The remote station was deployed on the No. 2 North Mains Section (refer to Figure 2) as part of the section power center. Besides the remote station the power center contains three other classes of equipment. They are all listed along with a description of their functions.

1. Mine Machinery Power Supply - This equipment develops the appropriate power level needed to drive the various pieces of mining face machinery. The mine machinery power supply equipment will not be discussed in detail later, unless this equipment relates or interfaces with the EMS.

2. Power Center Performance Transducers - There are five power center performance transducers located in the monitor system equipment section of the power center. They are as follows:

a. Continuous miner current
b. Rectifier current
c. Secondary bus voltage
d. Secondary bus power
e. Transformer temperature

These transducers develop analog output voltage proportioned to the particular parameter being measured. The analog signals are processed by the power center monitor system modules along with the external mining face environment transducer analog inputs. A summary of the power center performance transducers is provided in Table 1.

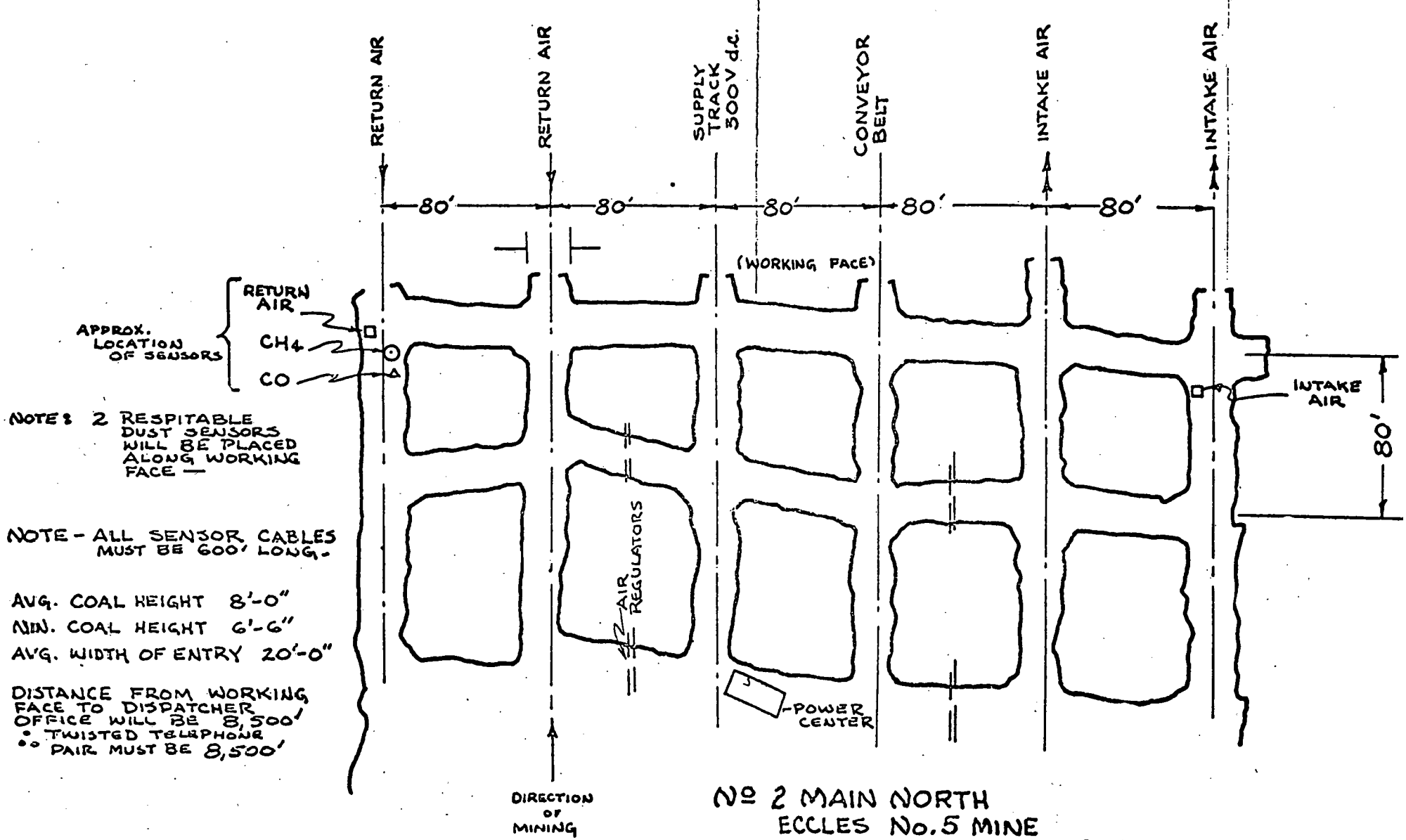
3. Mining Face Transducer DC Power Supplies - Two DC power supplies provide power to the mining face transducers. They are listed below:

a. + 12 VDC at 0.8 amp. This power supply provides + 12 VDC at 0.8 amp. to the CO transducer. The power supply is designed to be intrinsically safe and it features redundant circuitry.

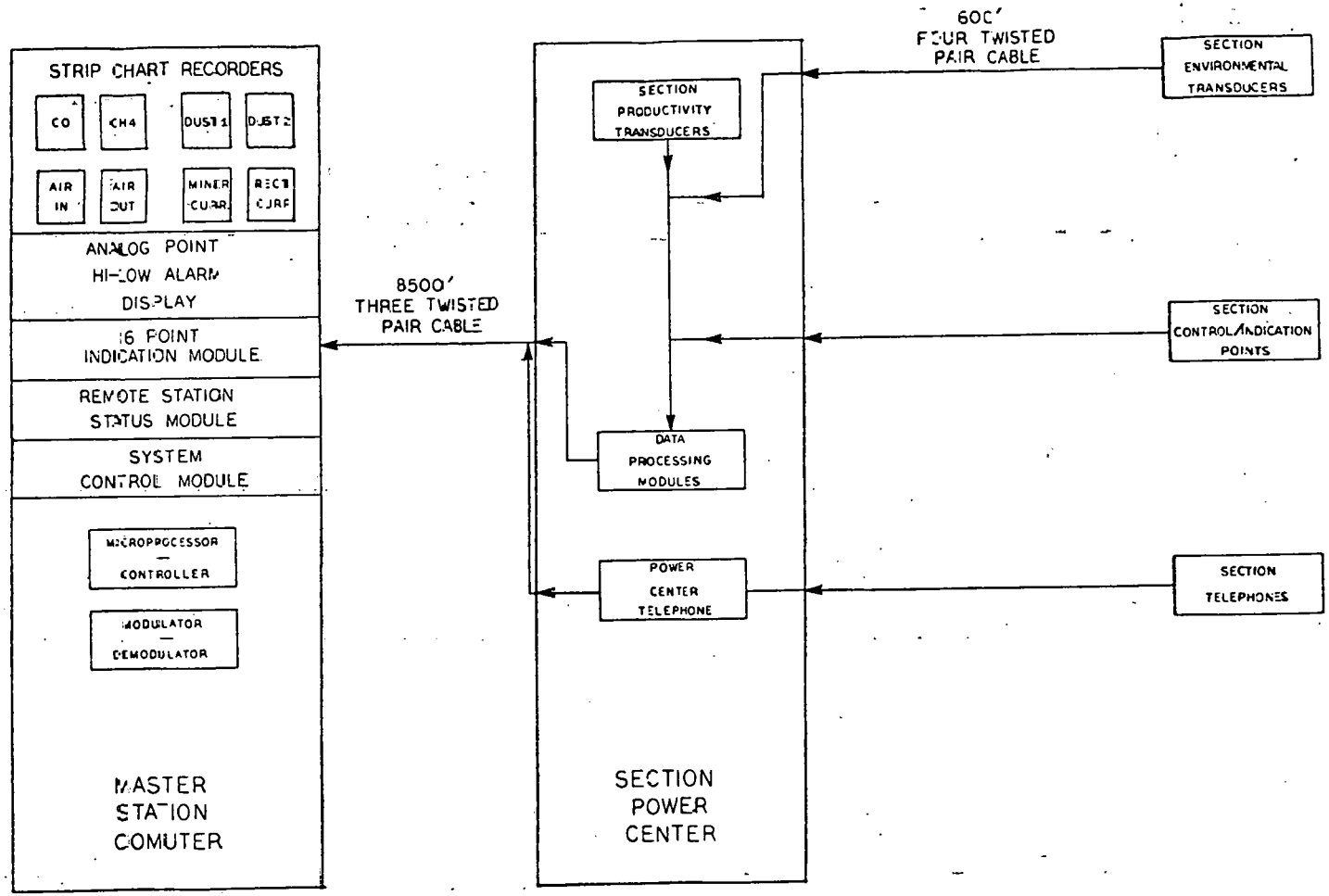
b. + 12 VDC at 0.1 amp. This power supply provides + 12 VDC at 0.1 amp. to the two air velocity transducers. The power supply features redundant circuitry and it is designed to be intrinsically safe when used with the air velocity transducer.

4. Transducer Information Processing Modules - Several information processing modules convert the transducer analog inputs into data suitable for transmission over an unconditioned voice grade communication channel.

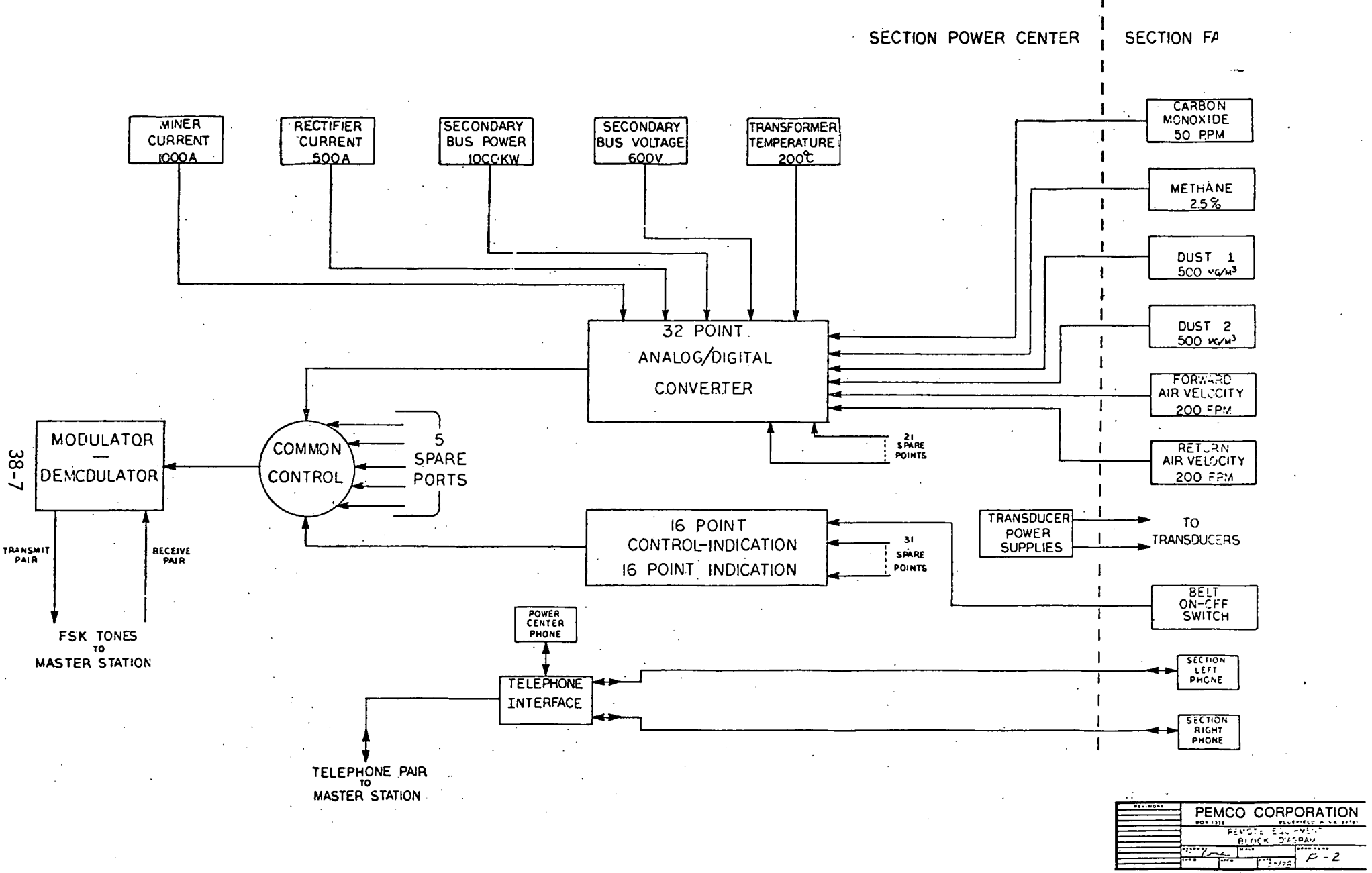
The communication data is transmitted from the power center via two pairs of twisted pair telephone cable. The communication cable is 8500 ft. long and is routed along side a trolley from the power center



No 2 MAIN NORTH
ECCLES No. 5 MINE
WESTMORELAND COAL CO.
ECCLES, W. VA. 25836



PEMCO CORPORATION
BLUEFIELD, W. VA.
MONITOR SYSTEM DIAGRAM
P-1


SECTION POWER CENTER
SECTION FA
MINER CURRENT 1000A
RECTIFIER CURRENT 500A
SECONDARY BUS POWER 1000KW
SECONDARY BUS VOLTAGE 600V
TRANSFORMER TEMPERATURE 200°C
CARBON MONOXIDE 50 PPM
METHANE 2.5%
DUST 1 500 µG/M³
DUST 2 500 µG/M³
FORWARD AIR VELOCITY 200 FPM
RETURN AIR VELOCITY 200 FPM
32 POINT ANALOG/DIGITAL CONVERTER
21 SPARE POINTS
MODULATOR — DEMODULATOR
COMMON CONTROL
5 SPARE PORTS
16 POINT CONTROL-INDICATION
16 POINT INDICATION
31 SPARE POINTS
TRANSDUCER POWER SUPPLIES
TO TRANSDUCERS
BELT ON-OFF SWITCH
TRANSMIT PAIR
RECEIVE PAIR
FSK TONES TO MASTER STATION
POWER CENTER PHONE
TELEPHONE INTERFACE
SECTION LEFT PHONE
SECTION RIGHT PHONE
TELEPHONE PAIR TO MASTER STATION
38-7
PEMCO CORPORATION
P-2

to the computer master station located in the dispatcher's office. The 4-wire communication channel provides for full duplex (dedicated transmit pair and receive pair) data transfer between the master station and the power center.

MINING FACE ENVIRONMENT TRANSDUCERS

Six transducers gather information related to the mining face environment and send this information via 500 ft. cables to the power center. The six transducers consist of two air velocity transducers for intake air and return air, two dust monitors which measure respirable dust concentration along the working face, a carbon monoxide (CO) sensor, and a methane (CH_4) sensor. In addition, a conveyor belt ON/OFF signal is provided to the power center. A summary of the transducers and their characteristics is shown in Table 1.

INTERCONNECTING CABLES

The transducer cables are 500 ft. long on 18 inch diameter reels. Each cable contains three triple twisted pairs of No. 18 AWG conductors with No. 22 AWG drain wire for each pair of conductors. The communications cable consists of three twisted pairs of No. 14 AWG standard mine telephone wire which is surrounded by a continuous copper shield. Only two pairs are used in monitor communications. The other pair is used for communication via standard mine pagers between the master station and the power center. Figures 3 and 4 show the complete mine monitoring system.

PROBLEMS ENCOUNTERED

It was decided to put the master station underground in the dispatcher's office since this is the center for mine communications. Several days into the project, it became apparent that the master station would have to be moved to another location. One of the reasons for this was that the dispatcher's office was too small to operate and test the master station while the dispatcher carried out his daily duties. Since the dispatcher's office is the center for mine communications, it attracted quite a bit of personnel traffic who could not resist flipping the switches and pressing the buttons on the master station. The master station was finally moved to the underground mine foreman's office which was considerably better for testing and trouble shooting purposes. There was still some problem with the mining personnel bothering the master station. The master station will eventually be brought to the surface pending the construction of a small building to house it.

The first major problem encountered upon activation of the monitoring system was that the communications cable was picking up interference in the form of the 100KHZ carrier on the DC trolley system. This interference would activate the "No Reply" and the "Invalid Reply" alarms indicating that something was wrong with signal processing equipment. This problem was finally solved by installing a bandpass filter on the receive and transmit lines of both the master station and the remote station.

TABLE 1 - SUMMARY OF TRANSDUCER CHARACTERISTICS

Transducer No.	Monitored Parameter	Analog Output	Power Input	Manufacturer & Model	MESA Status
1.	CO	0-4V	+ 4 to 6.8 VDC	Mine Safety Appl. Model 3	No application yet
2.	CH4	0-1V	115 VAC	General Monitor Model 180	Conditional use Letter of approval
3.	Dust #1	0-100MV	Battery	Bureau of Mines	Permit No. 332
4.	Dust #2	0-100MV	Battery	Bureau of Mines	Permit No. 332
5.	Air Velocity ± 1	0-5V	+ 12 VDC at 40 MA	J-TEC Assoc. Model VA-215	Application in process
6.	Air Velocity ± 2	0-5V	+ 12 VDC at 40 MA	J-TEC Assoc. Model VA-215	Application in process
7.	Miner Current	0-5V	----------	Scientific Columbus Model CT-510AZ	----------
8.	Rectifier Current	0-5V	----------	Scientific Columbus Model CT-510AZ	----------
9.	Sec. Bus Power	0-5V	----------	Scientific Columbus Model XL31K5A2	----------
10.	Sec. Bus Volts	0-5V	----------	Scientific Columbus Model VT-110AZ	----------
11.	Transformer Temp.	0-5V	115 VAC	Scientific Columbus Model 9019C	----------
12.	Belt On - Off				

A problem was also encountered in deploying the six environmental transducers from six separate cable reels located at the end of the power center. This was a very awkward situation when it came time to move the power center thus creating some confusion for the face production personnel. The six cable reels were replaced by two reels of cable with enough conductors to take care of four transducers each. The environmental transducers were connected into junction boxes located on the right-hand side and on the left-hand side of the section. This greatly simplified move-up of the equipment and reduced the maze of cables at the power center.

Obtaining reliable and accurate environmental transducers for gas and dust is a problem that has not been completely solved. We only recently obtained a carbon monoxide transducer from MSA. The reason for waiting so long for the transducer was that it would not remain stable for a practical amount of time. We hope that this problem has been corrected with this new transducer.

The two dust transducers are the type MSHA uses to obtain dust samples from working sections. These units have their own rechargeable power supply which lasts eight hours. The units must then be recharged. Our problem has been in finding a place to recharge them where they won't be tampered with. Also, there is no standardized method for them to be calibrated underground. We hope to eventually solve these problems as we go deeper into this field.

The methane transducer and the two air velocity transducers have given us no problems to this date.

These are the major problems we have encountered at this point of operation.

CONCLUSION

The continuous satisfactory operation of a mine depends upon knowing what is happening in every area of the mine. This project proves that the hardware is available to perform the job of informing management of what is happening from an environmental and production standpoint. There must be continued work on the sensors if practical devices to collect accurate information are to be readily available and convenient to use. A careful selection of the monitoring and supervisory control equipment will then become a major decision for mines in the future as the economic feasibility of a system becomes more realistic.

EARLY WARNING FIRE DETECTION USING

LOW LEVEL CARBON MONOXIDE MONITORS

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U.S. Department of Labor
Pittsburgh, Pennsylvania 15213

ABSTRACT

The Ventilation Branch of Pittsburgh Technical Support Center, Mine Safety and Health Administration (MSHA), has demonstrated an early warning system capable of monitoring the mine air, determining whether this air contains trace amounts of carbon monoxide from a developing fire or heating and providing alarm before the fire or heating has progressed beyond the initial stages. This system, developed using commercially available equipment, has operated successfully underground and has demonstrated that the remote monitoring of carbon monoxide at low levels is a feasible approach to early warning fire detection in underground coal mines.

INTRODUCTION

The danger of underground coal mine fires has been recognized since the inception of underground mining and because of this hazard, extensive precautions are taken to lessen the probability of fire and to protect and evacuate underground personnel should a fire occur.

The Ventilation Branch, Division of Health Technology, Pittsburgh Technical Support Center, MSHA, is engaged in a project which will provide the underground miner with increased protection from fire. This project has resulted in the development of an early warning system capable of monitoring the mine air, determining whether this air contains carbon monoxide from a developing fire or heating, and providing alarm before the fire or heating has progressed beyond the initial stages. By identifying a fire during the very early stages, before significant fire contaminants result, the system protects the miners against the toxic products of combustion, gives additional time to safely withdraw personnel from the affected area and permits direct fire fighting techniques to be used.

The uses for this or similar systems are many and diverse. Similar systems have already gained acceptance throughout Europe and in Canada for the early detection of spontaneous combustion (1-2).[1] In this country, proposed uses include main mine fan monitoring, monitoring of air which has ventilated gob areas and monitoring of air in belt entries as either the primary fire detection system or as an additional safeguard in mines where it is necessary to use air which has traveled the belt entry to ventilate the working face.

THE EQUIPMENT

The early warning system developed by MSHA utilizes off-the-shelf components to monitor the carbon monoxide content of the air at a series of remote stations and transmit this information to a central location (master station). It was determined that at the state of the art, carbon monoxide is the most sensitive detector gas for indicating the onset of coal heating (3-4). The CO monitor used is an Energetic Science Inc., Ecolyzer Carbon Monoxide Monitor[2] having a range of 0-50 ppm CO. This instrument is specific for carbon monoxide and uses the electrochemical method for detection. It has progressed from a laboratory instrument to an industrial model suitable for use in a relatively clean and dry environment to a model which is suitable for use in an underground coal mine. The model used for initial field testing of a prototype early warning fire detection system was a modified version of the industrial unit, designed to MSHA's specification and intended as a first step towards development of a mine quality instrument. Experienced gained with this monitor in an operating underground coal mine assisted in developing the Series 4000 monitor.

The principal of monitor operation is relatively simple. Air is drawn into the monitor by an internal pump at the rate of 700 cc/min. The pump has sufficient capacity to draw a sample through 3/16 inch diameter tubing for distances exceeding 100 feet without appreciable reduction in flow. This capability permits monitor placements remote from areas where they may be damaged. The sample is first filtered, and scrubbed to remove both particulates and interfering gases. Table 1 is a list of interfering gases which have been checked, showing the gas concentration tested and the resultant monitor reading.

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[1]Underlined numbers in parentheses refer to items in list of references at the end of this presentation.

[2]Reference to specific brands, equipment, or trade names in this report is made to facilitate understanding and does not imply endorsement by the Mine Safety and Health Administration.

TABLE 1. - Monitor response to Interfering gases

Interfering Gas	Symbol	Concentration Checked	Monitor Reading (ppm)
Methane	CH_4	5%	0.0
Carbon Dioxide	CO_2	1%	0.0
Ammonia	NH_3	500 ppm	0.0
Nitric Oxide	NO	50 ppm	0.0
Nitrogen Dioxide	NO_2	25 ppm	0.0
Nitrous Oxide	N_2O	100 ppm	0.0
Acetylene	C_2H_2	5 ppm	15.0
Ethylene	C_2H_4	10 ppm	10.0
Ethane	C_2H_6	10 ppm	0.1
Propane	C_3H_8	100 ppm	1.0
Hydrogen	H_2	50 ppm	0.5
Hydrogen Sulfide	H_2S	50 ppm	0.5
Sulfur Dioxide	SO_2	100 ppm	1.0
Natural Gas		100%	2.5

The sampled air enters the electrochemical sensor where the actual carbon monoxide concentration is determined. Within the sensor carbon monoxide is electro-oxidized to carbon dioxide at a platinum electrode in an aqueous electrolyte. A second electrode serves as a cathode for the electro-reduction of oxygen. The current flowing between these two electrodes is proportional to the concentration of carbon monoxide in the gas sample.

The equations which govern these reactions are:

a. $CO + H_2O \xrightarrow[\text{anode}]{} CO_2 + 2H^+ + 2\bar{e}$

b. $1/2O_2 + 2H^+ + 2\bar{e} \xrightarrow[\text{cathode}]{} H_2O$

c. $CO + 1/2O_2 \longrightarrow CO_2$ (overall reaction)

Early versions of this monitor, including the prototype models used for underground testing, required a prehumidifier to maintain the sampled air at a constant humidity. This presented a serious problem with condensate forming in the cell during summer months when the relative humidity underground approached 100 percent. New instruments are now equipped with a sensor which operates at relative humidities ranging from 60 to 100 percent without a prehumidifier. If lower humidities are anticipated the standard cell having an operating range of 10 to 85 percent relative humidity can be used.

The monitor has a detection range of 0 to 50 ppm CO with a minimum detectible sensititvity of 1 percent of full scale. Power for the Series 4000 monitor may be supplied by either 110 volts AC or 12 volts DC. The monitors used during the underground test were powered by batteries continually charged by mine power. These monitors are capable of continued operation for approximately six hours after an interruption in the primary power source.

The second component of the fire detection system is the data acquisition and alarm monitoring equipment. Although the monitors used for underground testing were equipped with both audible and visual alarms it was felt that central monitoring of alarms and carbon monoxide concentrations was essential. With the assistance of the Pittsburgh Mining and Safety Research Center, United States Bureau of Mines, a system was obtained which represents or approaches the state-of-the-art in this field. The demonstration telemetry/data retrieval system used by MSHA consisted of:

1. 1 - Fluke Datalogger
2. 12 - Sonex FM Discriminators
3. 1 - Sonex Power Supply
4. 3 - Sonex 4 channel Transmitters
5. 3 - Ocenco Power Supplies

The first three items comprise equipment used at a master station while one each of items 4 and 5 were installed at three remote monitoring locations.

A brief description of the operation of this system is presented to facilitate understanding, however, any telemetry/data retrieval system, including a hard wire system, capable of transmitting the monitor output to a remote location would be applicable. This was successfully demonstrated during the early stages of this test when a hard wire system was used (5). The carbon monoxide monitors produce a 0-1 volt DC signal proportional to the carbon monoxide concentration. This direct current signal is converted to a frequency modulated signal which is transmitted on twisted pair phone wire to the master station where a discriminator separates the various frequencies (each monitor having its own separate and distinct frequency) and converts them to a 0-10 volt direct current signal proportional to the carbon monoxide concentration. The output from the discriminator is then fed into the Datalogger which automatically scans through the operating discriminator channels according to a preprogrammed format. Should any of the monitors indicate a carbon monoxide concentration exceeding a preset level, a visual alarm is activated and, depending on the nature of the program, the Datalogger will either automatically print the number of the channel and the CO level or this information can be requested by depressing a switch on the front panel of the Datalogger. Unfortunately the Datalogger was not suitable for underground use and was replaced with a simpler unit during the test. This will be discussed in greater detail later in this paper.

UNDERGROUND PERFORMANCE

In March and April of 1976, underground installation of ten (10) carbon monoxide monitors was completed. This installation followed approximately 45 days of laboratory operation during which initial shakedown and environmental tests were completed. During environmental testing, the monitors operated satisfactorily through a temperature range of 50°F to 77°F with a relative humidity range of 43 to 94 percent.

Of the ten (10) monitors installed, three (3) were purchased by MSHA for testing in an experimental early warning carbon monoxide detection system. The three monitors were installed in the belt entry of the No. 3 Unit of the Island Creek Coal Company's, Virginia Pocahontas No. 4 Mine. The remaining seven (7) monitors, purchased by the Island Creek Coal Company, were installed in the belt entries of the remaining active sections of the mine. Figure 1 is a schematic showing the approximate monitor locations used.

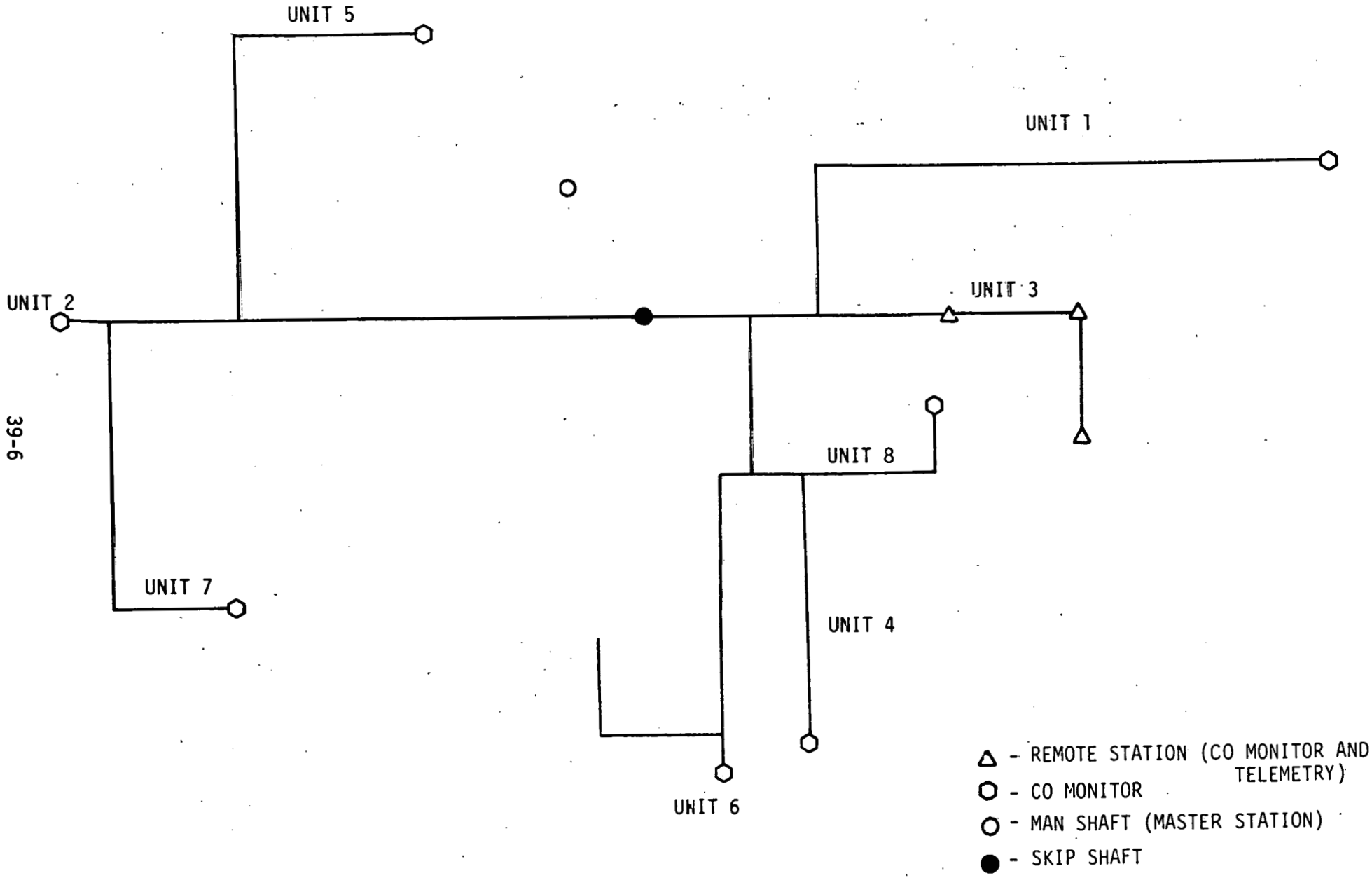


FIGURE 1. - Schematic of underground fire detection system installation.

During the first eight months of operation numerous instrument malfunctions occurred. Malfunctions can be classified into two groups; electrochemical cell malfunctions and electronic malfunctions. The electrochemical cell malfunctions which occurred were of three types and each type was addressed by MSHA and the manufacturer as it appeared.

The first type of cell malfunction which occurred, termed "cell leakage," was the product of a design flaw resulting in microscopic holes in the cell diaphragm which permitted the liquid electrolyte to escape from the cell. Redesign of the cell eliminated this problem.

The other types of cell malfunction, dehydration and cell flooding were directly related to the relative humidity of the sampled air. Cell dehydration occurred when the relative humidity of the sampled air was less than approximately 90 percent and the sample air was not humidified prior to entering the cell. This problem was corrected by maintaining the water level in the prehumidifier (water bottle) when the relative humidity fell below 90 percent. Cell dehydration caused a loss in sensitivity and increased the time for the instrument to respond to a change in the carbon monoxide concentration.

Cell flooding, on the other hand, results from condensation formed in the electrochemical cell during periods of high relative humidity underground. The condensed moisture in and around the cell results in erratic output from the monitor. The physical appearance of a cell which is flooded is similiar to one which is leaking, and generally a laboratory evaluation is required to distinguish between the two. To solve this problem, the manufacturer redesigned the electrochemical cell to partially compensate for relative humidity changes. The electrochemical cells used after August of 1976 require the use of the prehumidifier when the relative humidity at the sampling station falls below 90 percent and bypassing the prehumidifier when the relative humidity exceeds 90 percent. Because cell dehydration and cell flooding require time to develop, short term fluctuations in the relative humidity is not a problem.

Continued development has resulted in an electrochemical cell capable of operating without the use of a prehumidifier within a relative humidity range of 60 to 100 percent. This improved cell is currently available for use with the Series 4000 monitor.

The electronic malfunctions experienced during underground development and testing were of the type expected when any new piece of equipment is taken into a coal mine for the first time, especially when a laboratory grade instrument is extensively modified for DC power by rechargeable batteries rather than AC power. As problems arose with the monitors, modifications were made and the monitors were maintained in operating condition. The last modification was made in December 1976, and the monitors have operated satisfactorily since that time.

During the initial months of monitor operation frequent (twice weekly) operation checks of each monitor were made. As experience with the monitors was gained the number of checks were reduced to once a week and eventually to much longer periods.

Table 2 shows the results of calibration checks of the MSHA monitors during the period of December 14, 1976 to April 11, 1978.

TABLE 2. - Results of calibration checks
December 14, 1976 - April 11, 1978

Date	Calibration Gas ppm-CO	Monitor Reading ppm-CO		
		1	2	3
12/14/76	24.5	24.0	24.0	24.0
12/21/76	24.5	24.0	23.0	*
12/28/76	24.5	24.0	22.0	*
1/04/77	24.5	24.0	23.0	*
1/11/77	24.5	23.0	24.5	*
1/18/77	24.5	15.0	20.0	24.0
1/28/77	24.5	20.0	**	23.0
2/01/77	24.5	24.0	24.0	19.0
2/08/77	24.5	21.0	24.0	23.0
2/15/77	24.5	23.5	26.0	26.0
3/22/77	27.0	27.0	25.0	24.0
5/04/77	27.0	24.0	***	24.0
6/14/77	27.0	36.0	20.0	25.0
8/09/77	23.0	23.0	**	24.0
11/14/77	25.0	20.0	****	25.0
4/11/78	25.0	25.0	*****	****

* Monitor Damaged Due to Short Circuit in Telemetry, Monitor Repaired 1/11/77
** Power Off to Monitor - Could Not Check Calibration
*** New Sensor Installed
**** Monitor Removed for Maintenance
***** Monitor Reinstalled Underground

On June 14, 1977, all of the monitors (both MSHA's and the company's) were checked and calibrated using a 27-ppm calibration standard. The monitors were then permitted to operate until August 9, 1977, at which time the calibration was again checked using a 23-ppm calibration standard. The only checks made during this 8 week period were visual checks to insure that the monitors were receiving power. Table 3 shows the results of the calibration checks of August 9, 1977.

TABLE 3. - Results of the calibration checks of August 9, 1977

Monitor No.	Reading (ppm) (23 ppm calibration gas)	Zero	Remarks
MSHA owned			
1	23	-1	
2	---	---	Power off
3	24	1	
Company owned			
4	20	-1	
5	35	-1	
6	21	0	
7	26	+3	
8	---	---	Power off
9	24	-1	

As indicated by the results presented in Tables 2 and 3, the monitors appear to be capable of operating underground for extended periods with a minimum of care. Of the nine (9) monitors in operation on June 14 and August 9, six (6) were operating within manufacturer's specifications; two (2) were without power and could not be checked and one (1) had been maladjusted.

Each monitor was equipped with a strip chart recorder which gave a continuous record of the monitor output. By reviewing the recorder tracings for each monitor, it was possible to evaluate performance and instrument response to changes in the carbon monoxide concentration at the measuring station. It was also possible to determine the number of occasions an alert or an alarm condition was indicated, had the alert or alarm been set at 10 and 15 parts per million (ppm) respectively.

These alert and alarm levels would represent increases of 5 and 10 ppm above the maximum ambient established previously for this mine. In most instances, it is also possible to determine whether the increase was caused by a rise in the carbon monoxide level at the remote station or some other source (interruption of the signal between the transmitter and receiver, restoration of power following a prolonged power outage, or equipment malfunction). In like manner, it is possible to determine which outputs would have activated alert and/or alarm at both the remote and master stations and which would have activated alert and/or alarm at the master station only. Table 4 shows the results of a review of strip-chart recorder tapes for the monitors during the period December 7, 1976 to March 24, 1977.

TABLE 4. - Results of review of strip chart recorder tapes December 7, 1976 - March 24, 1977

Alarms caused by monitor output equivalent to 15 ppm CO or greater

Monitor Location	Apparently caused by		
	Carbon Monoxide	Loss of Signal	Power
Tail	0	1	1
Drive-1	0	5	3
Drive-2	0	2	7

Alerts caused by Monitor output equivalent to a CO concentration between 10 and 15 ppm:

Monitor Location	Apparently caused by		
	Carbon Monoxide	Loss of Signal	Power
Tail *	0	0	1
Drive-1	1	0	0
Drive-2	3	0	0

* Due to a malfunction in the telemetry receiver recorder output for this Monitor is not available after January 28, 1977. Calibration checks, however, indicate satisfactory operation.

As indicated, Table 4, there would have been no alarms and only 4 alerts at the master station resulting from increases in the carbon monoxide concentration. The 4 alerts were the result of short term increases in the CO level probably caused by surface activity. Because individual monitors were not equipped with alert indicators, alert would not have been given on the section. On eight (8) occasions, alarm would have been activated at the master station due to a loss of signal between the transmitter and the receiver. These alarms would not have been activated on the section. The remaining eleven (11) alarms listed under the general heading "power" on Table 4 can be broken into three types: (1) four appear the result of prolonged power outages resulting in a loss of signal from the transmitter; (2) three appear the result of restoring power following an outage (outage not long eneough to cause loss of signal); and, (3) four short duration alarms appear the result of electrical interference. The alarms caused by restoring power following outages can be expected with this monitor and serve to point out the need for a reliable power source. As indicated, Table 3, calibration checks during the period indicate satisfactory monitor performance.

In addition to the three Series 3000 monitors installed in the belt entry of the No. 3 Unit, a fourth monitor, a prototype Series 4000 monitor was installed on December 7, 1976. Unlike the Series 3000, the Series 4000 has no back-up batteries and monitoring capability is lost when mine power is interrupted and an increase in output results everytime power is restored. The Model 4000 was a "bare bones" monitor in that it had no meter for direct readout, no alarms, and no relay capability. This monitor was connected to the telemetry system and the output recorded at the master station. Table 5 gives the results of a review of the strip chart recorder tracings from December 7, 1976 to March 24, 1977, for this monitor.

TABLE 5. - Results of review of stripchart recorder tapes for Series 4000 Ecolyzer December 7, 1976 - March 24, 1977

Total number of alarms	77	
Total number of alerts	5	
Alarms caused by CO	0	
Alerts caused by CO	0	
Alarms caused by power outages/restorations	71	
Alerts caused by power outages/restorations	5	
Alarms caused by loss of signal	5	
Alerts caused by loss of signal	0	
Miscellaneous Alarms	1	(Electrical Interference)
Miscellaneous Alerts	0	

As indicated, Table 5, the Series 4000 monitor would have activated 77 alarms during the period December 7, 1976 to March 24, 1977. Of these, 71 were the result of restoring power following an outage, five appear to be the result of signal loss and one appears the result of electrical interference. In addition, five alerts were caused by short power outages. This monitor was installed at the No. 1 Belt Drive and was powered from the same source as one of the Series 3000 monitors. Because of the backup batteries used in the Series 3000 monitor, only three of the 76 power outages indicated by the Series 4000 monitor affected the Series 3000. This once again indicates the need for a highly reliable power source for this or any other monitoring system.

While the operation and performance of the carbon monoxide monitors since December 1976 has successfully demonstrated the capability of this instrument to function underground, one component of the data retrieval system did not prove satisfactory. Initial plans were to locate the master station in a building on the surface. These plans were modified however when a dispatcher's office was built at the bottom of the intake shaft. This office provided a location underground which was relatively clean and was manned at all times when personnel were in the mine and eliminated the difficult work of installing transmission cable in the 1300 foot deep shaft. However, a malfunction occurred with the Datalogger in October 1976 which required its removal from the mine. After being repaired, it was reinstalled at the master station in January 1977. Subsequent malfunctions however, have demonstrated that this particular component of the system is too delicate to withstand the rigors of the underground environment even when protected within the confines of a sheltered area such as an underground office. Therefore, the Datalogger was removed from the mine and replaced with an alarm/monitoring panel constructed by personnel of the Ventilation Branch using "off-the-shelf" components. This panel is capable of monitoring and recording the output from four remote stations and is equipped with a single audible alarm and multiple visual alarms for each monitor. The MSHA constructed alarm/monitoring panel has functioned satisfactorily since being installed in December 1976, and has demonstrated that while sophisticated equipment is available a simpler method may be more appropriate and less expensive.

CURRENT WORK

In developing an Early Warning Fire Detection System for underground coal mines, MSHA selected equipment which represented state of the art at that time. The state of the art in this field, however, is advancing rapidly as new and existing technology is applied to the problem. For this reason, MSHA is continuing to investigate alternative

monitoring techniques and equipment as well as additional applications for the existing system. One application which is currently under investigation due to requests from various mining companies is the replacement of existing fire detection systems using point-type heat sensors with a system using product of combustion detectors. Under the provision of 30 CFR, Section 75.1103 such a system must provide protection equivalent to point-type sensors. It is in this area of equivalency that MSHA is currently working.

A 60-foot long section of conveyor belt has been constructed in a fire gallery located at MSHA's facilities in Bruceton, Pennsylvania. Electrical heaters installed in a simulated return roller located at one end of this belt and partially covered with coal are used to slowly develop a coal/belt fire. Using four thermocouple arrays installed along the belt a temperature profile of the entry can be drawn and compared with the carbon monoxide generated during the development of these fires. Although tests are incomplete and results as yet not fully analyzed, initial indications are that it will be possible to provide the miner with protection against fire equivalent to that afforded by point-type sensors through the proper spacing of low level carbon monoxide monitors. The spacing of these monitors will be a function of several variables including: belt entry air velocity, physical characteristics of the entry, and carbon monoxide alarm levels. Due to the complexity of the problem and the limited resources and manpower available to MSHA for this type of work, current plans are to request the assistance of the Bureau of Mines in further studying this problem.

CONCLUSION

The successful operation of an early warning fire detection system using "off-the-shelf" low level carbon monoxide monitors has clearly demonstrated that the state of the art in this field is sufficiently advanced to warrant the use of such a system on a large scale basis. This system, if properly utilized and maintained, may be capable of affording to the underground coal miner a degree of protection against mine fires which here to fore has not existed. In addition, by detecting fires in their very early stages mine management may more often be in a position to apply direct fire fighting techniques rather than fighting a fire remotely or even sealing the mine.

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