Information Circular 9228

Surface Testing and Evaluation of the Multiple-Unit Continuous Haulage System

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UNITED STATES DEPARTMENT OF THE INTERIOR
Manuel Lujan, Jr., Secretary

BUREAU OF MINES
T S Ary, Director

Library of Congress Cataloging in Publication Data:

Jaspal, Jasinder S.
Surface testing and evaluation of the multiple-unit continuous haulage system.

(Bureau of Mines information circular; 9228)

Includes bibliographical references.

Supt. of Docs. no.: I 28.27:9228.

1. Mine haulage. I. Erhard, Lee A. II. Mayercheck, William D. III. Title. IV. Title: Multiple-unit continuous haulage system. V. Series: Information circular (United States. Bureau of Mines); 9228

TN295.U4 [TN331] 622 s [622'.66] 89-600120

CONTENTS

ILLUSTRATIONS

TABLES

UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

A	ampere	kW·h/st	kilowatt hour per short ton
°C	degree Celsius	kW·h/(st·ft)	kilowatt hour per short ton per foot
dBA	decibel, A-weighted	lb	pound
°F	degree Fahrenheit	min	minute
ft	foot	mm/s	millimeter per second
ft/min	foot per minute	pct	percent
gal/min	gallon per minute	psi	pound per square inch
h	hour	r/min	revolution per minute
hp	horsepower	s	second
Hz	hertz	st/min	short ton per minute
in	inch	V	volt
in^2	square inch	V ac	volt, alternating current
kV·A	kilovolt ampere	yd^3	cubic yard
kW	kilowatt		

SURFACE TESTING AND EVALUATION OF THE MULTIPLE-UNIT CONTINUOUS HAULAGE SYSTEM

By Jasinder S. Jaspal,[1] Lee A. Erhard,[2] and William D. Mayercheck[3]

ABSTRACT

Most of the underground coal in the United States is mined via room-and-pillar mining methods with continuous miners. These machines operate intermittently because they have to wait for shuttle cars to interchange positions. To overcome this discontinuity in shuttle car haulage and to realize the full production potential of continuous miners, the U.S. Bureau of Mines developed a multiple-unit continuous haulage (MUCH) system through a research contract with Jeffrey Mining Machinery Div. (JMMD). The MUCH system consists of 12 rubber-tired vehicles and a bridge conveyor. The rubber-tired vehicles are connected by a unique mechanical linkage system to form a 250-ft train. The mechanical linkage permits the vehicles to track-retrack the preceding vehicle in both inby and outby directions. The cut coal cascades from one vehicle to another until it is discharged on the section conveyor belt. The MUCH system provides continuous haulage to the continuous miner. The MUCH system was initially surface tested by JMMD. Subsequent extensive testing was conducted by the Bureau at its test facilities at Bruceton, PA. Deficiencies found during testing were corrected. This report summarizes initial tests by JMMD and subsequent extensive tests and evaluation by the Bureau.

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INTRODUCTION

In room-and-pillar mining, a continuous miner cannot operate continuously because it must halt production-loading operations to permit a loaded shuttle car to move out and an empty shuttle car to move in before the production-loading operation can be resumed. This intermission precludes realization of full production potential and shows a weakness in a shuttle car haulage system.

For the mining of 90° or other angle pillar entries or crosscuts, no haulage system was available that could remove the production coal continuously from a continuous miner. This situation led the Bureau to develop the Multiple-Unit Continuous Haulage (MUCH) system through a contract with Jeffrey Mining Machinery Div. (JMMD) of Dresser Industries, Inc. JMMD designed and fabricated the system and surface tested it at its plant. During these tests, numerous modifications were made to the MUCH tracking-retracking and conveying systems. After these modifications, the MUCH system was tested at the Ohio Transportation Research Center (OTRC). Additional modifications were made to the system to improve its performance.[4] The modified MUCH system was brought to the Bureau's Mining Equipment Test Facility (METF) in 1983 for more comprehensive surface testing with the purpose to evaluate the system performance and operational characteristics, correct any faults found during tests, and prepare the system for an in-mine trial, including an experimental permit approval from the Mine Safety and Health Administration (MSHA).

Most mine operators prefer not to risk the installation and operation of prototype equipment because any failures would be expensive in terms of operating costs and loss of production. Because of this risk, there is a strong tendency in the mining industry to resist the introduction of new equipment or reject promising equipment after short duration of unsuccessful trials. The MUCH system was, therefore, subjected to rigorous surface tests to improve its reliability in performance and eliminate or reduce any major design deficiencies. Modifications and surface tests on the MUCH system were completed in December 1986 at the METF. The results of tests indicate that the MUCH system will perform successfully in an in-mine trial.

The MUCH system is scheduled to be used in 1989 at a highwall operation with a remotely controlled Jeffrey 102HP continuous miner. Necessary changes are being made on it for its operation with a remote controlled system.

ACKNOWLEDGMENTS

The authors want to extend their sincere appreciation to Robert J. Evans, civil engineer, Pittsburgh Research Center, for technical guidance during surface testing at the METF.

DESCRIPTION

The MUCH system consists of 12 rubber-tired vehicles in which there is one lead vehicle, 10 intermediate vehicles, and one discharge vehicle with a bridge conveyor to form a 250-ft-long train. Intermediate vehicles can be added or removed from the train to suit the section requirement. Each MUCH vehicle has a chain conveyor mounted on a transporting vehicle that features four-wheel steering and two-wheel drive. There is a hopper on each vehicle to receive coal from an inby vehicle. Except for the lead vehicle, the hopper of each vehicle is sized and shaped to accept coal in all steering positions.

The vehicles are connected by a patented[5] self-tracking steering system that connects adjacent vehicles into a train with automatic mechanical tracking and retracking (fig. 1). Coal cascades from conveyor to conveyor at the rate of 12 st/min until it reaches the section belt. The train of vehicles is steered inby by the MUCH system operator in the lead vehicle, and outby by the discharge vehicle operator located outby near the section belt. The mechanical steering linkages on each vehicle enable all vehicles to sequentially track the path of the preceding vehicle through a mine at 80 ft/min. System specifications are given in appendix A.

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[4]Hundman, G. J., and P. W. Meisel. Development of a Multiple Unit Continuous Haulage System (contract JO333941, Dresser Industries, Inc.). BuMines OFR 101-84, 1983, 326 pp; NTIS PB 84-188630.

[5]Voight, E. T. Steering System for a Train of Rail-less Vehicles. U.S. Pat. 4,382,607, May 10, 1983.

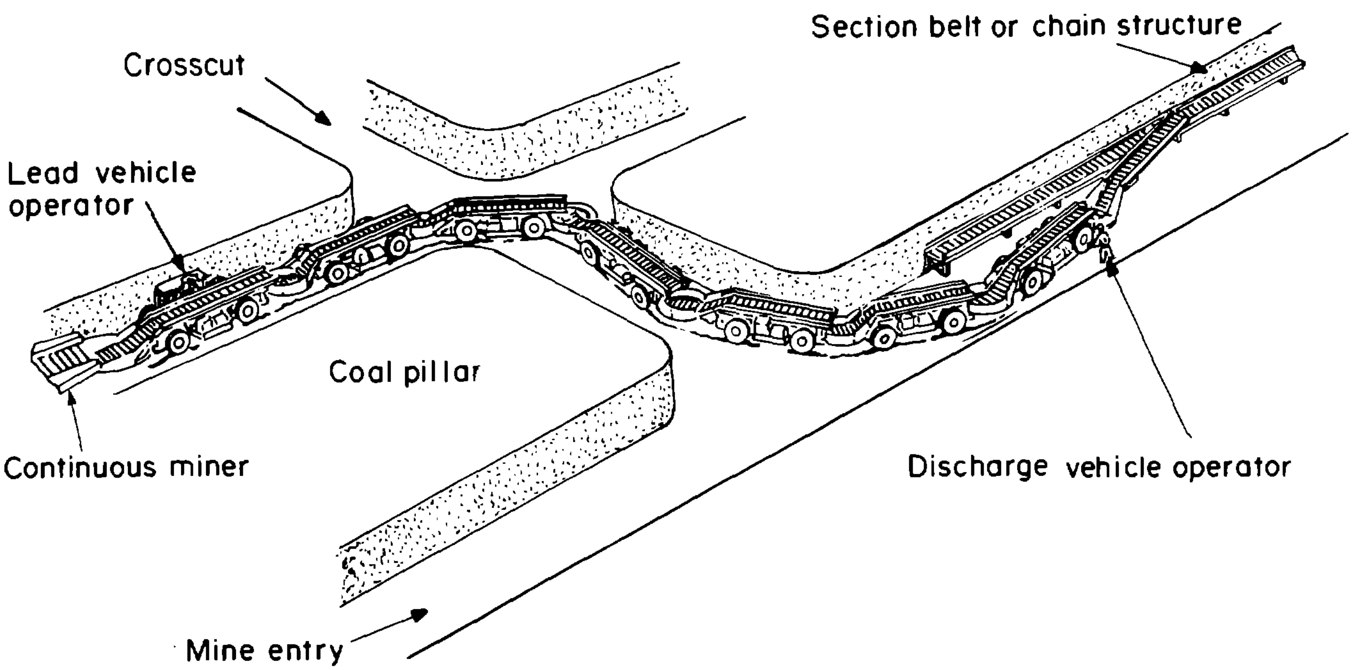

Figure 1.–Multiple-unit continuous haulage system.

LEAD VEHICLE

The lead vehicle is the first vehicle in the train and it receives coal directly from the tailboom of a continuous miner. An operator compartment with canopy is positioned on the right side of the vehicle. The lead vehicle contains a hydraulic power system to power the hydraulic front-wheel steering and the hopper lift cylinders. The lead vehicle also has headlights, a communications system, a panic bar emergency shutoff system, and hydraulic brakes (fig. 2).

The MUCH system is controlled by an operator from the lead vehicle. The operator sits on a padded seat, the back of which adjusts to a reclining position for operating in lower seams.

A tram control stick is located to the left of the operator, near the left knee. Movement of the stick forward provides forward tram, while movement to the rear causes reverse tram. The right foot pedal is a deadman's control that commands tram movement of the train. Simultaneous deflection of the control stick and depression of the right foot pedal are required to tram. This simultaneous action cannot be accomplished from outside the operator compartment.

Lateral movement of the left control stick operates a hydraulic valve located behind the operator's seat. This hydraulic valve commands lead vehicle front-wheel steering through a pair of push-pull hydraulic cylinders. A control lever is also located to the right of the operator. This lever controls a hydraulic valve to raise and lower the lead vehicle's coal receiving hopper.

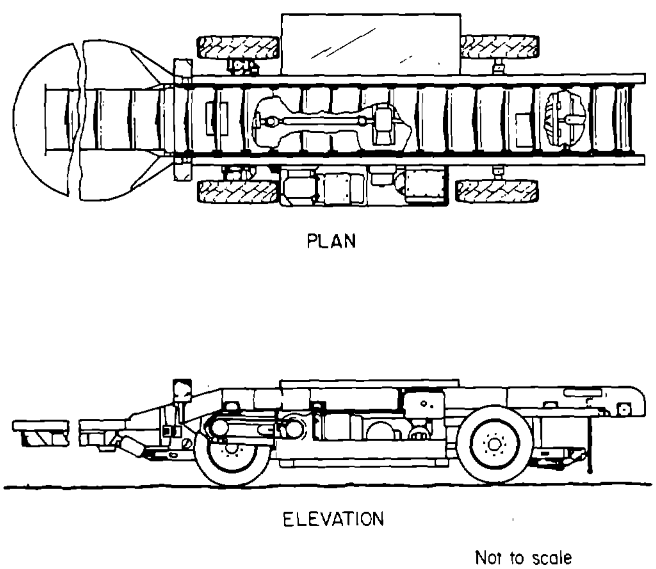

Figure 2.–Lead vehicle.

In addition to the right foot pedal for tram control, a left foot pedal operates a disk brake through a hydraulic master cylinder. This brake is on the lead vehicle rear wheels only, and serves as a braking assist to the tram motor brakes. An electrical control case is mounted to the left of the operator. The control panel functions, starting at the outby end, are as follows:

Lead vehicle motor control breaker.

Hydraulic pump and headlight breaker.

Conveyor systems start switch.

Conveyor systems stop switch.

Lead vehicle conveyor reverse switch.

Lead vehicle conveyor run switch.

Start hydraulic pumps switch.

Stop hydraulic pumps switch.

Lead vehicle headlight on switch.

Lead vehicle headlight off switch.

An emergency stop bar, which activates an emergency stop switch, runs along the upper edge of the control panel. The emergency stop switch activates the main line circuit breaker (CB-1) shunt trip, causing power to be interrupted over the entire train.

Two headlights are mounted next to the conveyor at the front of the lead vehicle. The headlights are operated from the lead vehicle operator station. During normal operation, the headlights would remain on. When the system is positioned behind the continuous miner, receiving and transporting coal, the MUCH operator can switch the headlights off if desired.

Tram Drive

The horsepower of the tram motor is greater on the lead vehicle than on the intermediate vehicles. The higher horsepower motor is required because of the additional weight of the lead vehicle. The lead vehicle carries the system operator, operator compartment, system controls, and a larger front-receiving hopper, and it must travel at the same speed as the intermediate vehicles. Therefore, the lead vehicle tram motor is 7.5 hp instead of 5 hp, as used on the intermediate vehicles.

All tires used on the MUCH system are the same size–8.25 by 15. However, the additional weight of the lead vehicle requires higher tire inflation pressure. The lead vehicle tire pressure is 100 psi, while the intermediate vehicles have 75-psi foam-filled tires.

Hydraulic System

The lead vehicle utilizes hydraulic power for steering and to raise and lower the coal receiving hopper. The hydraulic package located on the left side of the lead vehicle includes a 1-hp electric motor, hydraulic pump, hydraulic reservoir, and function controls. The hopper of the lead vehicle is 18 in longer than the hoppers of the intermediate vehicles. This additional length gives the MUCH system a capability to remain stationary while the continuous miner goes through one cycle of sumping, cutting, and loading.

Conveyor Drive

The lead vehicle and all other vehicles of the MUCH system have a 15-hp, electric, flange-mounted, conveyor drive motor attached to a speed reducer. A chain drive with a 1.16:1 sprocket speed reduction is used from the speed reducer to the conveyor drive shaft. The conveyor consists of 2-in-pitch, double-strand chain, 506 in long, and 29.5 in wide. The majority of flights are on 10-in centers. The conveyor deck is 30 in wide and has 9-in sideboards. The lead vehicle conveyor is the last to start in the MUCH system train. It is, however, the first vehicle from the continuous miner that starts conveying coal outby. Because it is the last inby vehicle of the train, it is not provided with a speed switch. With the chain running at 280 ft/min, a conveying capacity of 12 st/min is achieved.

INTERMEDIATE VEHICLES

The intermediate vehicles constitute the bulk of the system, and consist of conveyor deck plates, conveyor sideboards, receiving hoppers, axles, speed reducers, drive motors, electrical control boxes, and frames (fig. 3).

Tram Drive

Each individual intermediate vehicle has a 5-hp tram drive motor with a spring-actuated, electrically released brake. The tram motor is face mounted to a speed reducer, which is coupled to the front-driving axle via a drive shaft. If tram power to an intermediate vehicle is lost, the system can continue to operate by disconnecting the tram drive universal joint on the drive shaft until the problem can be corrected.

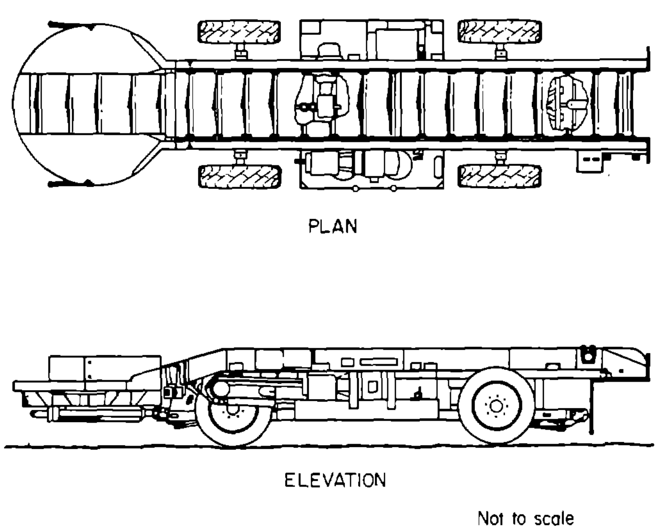

Figure 3.–Intermediate vehicle.

The brake is part of the electric motor and is spring-actuated and electrically released. The brakes are automatically applied with any electrical power interruption. The brake is used for parking and emergency applications only, and is controlled by the main breaker switch, panic bars, or any planned or unplanned power interruption.

There are two 15:1 speed reducers mounted on each vehicle in the system. The tram motor speed reducer is mounted on the lower frame under the conveyor; the conveyor drive speed reducer is mounted on the left side motor mount shelf.

Conveyor Drive

The chain conveyor drive system of the intermediate vehicles is similar to the lead vehicle. A speed switch is mounted to the tail shaft of each conveyor on each intermediate vehicle. The speed switches are designed to close when the conveyor speed reaches 85 pct of normal operational speed. When the conveyor start switch is activated, only the bridge conveyor starts. When the bridge conveyor reaches 85 pct of normal speed, the speed switch closes and the electrical circuit logic supplies power to the discharge vehicle conveyor drive motor. When this conveyor reaches 85 pct of normal speed, its speed switch closes to activate the next inby conveyor drive motor. As this action continues, the conveyors of the train sequentially start from outby to inby until the lead vehicle conveyor is actuated. The conveyor drive switching logic, in conjunction with the speed switches, is designed to prevent coal from being dumped onto a stopped conveyor.

DISCHARGE VEHICLE

The discharge vehicle, patterned after the intermediate vehicles, differs in a number of respects. This vehicle is designed to support the forward half of the bridge conveyor, which is connected to the vehicle by a hitch. The bridge conveyor increases the load on the discharge vehicle rear axle and decreases the load on the front axle. Therefore, the discharge vehicle (fig. 4) is the only vehicle with rear-wheel drive.

The electrical control box on the discharge vehicle is larger than those of the other vehicles because it contains the switching controls for both the discharge vehicle and bridge conveyor; the main electrical power enters the train at the discharge vehicle. The control case has a main line circuit breaker (CB-1) at the main power entry point. A 480-V ac to 120-V ac stepdown transformer is mounted on the discharge vehicle and supplies 120-V ac control power to the system.

The primary MUCH system operator is located at the lead vehicle while the secondary operator, or helper, is located at the discharge vehicle. A pager phone is provided at both the lead and discharge vehicle operator stations, as well as at the midpoint of the system.

The discharge vehicle operator controls all steering of the system when tramming in the outby direction. Because the discharge vehicle is connected to the panel belt by the bridge conveyor, steering is needed to keep the system from running into the mine rib or panel belt. The steering is controlled by a hydraulic unit that includes a 1-hp electric motor and a hydraulic reservoir. The hydraulic unit is mounted on the side of the conveyor sideboard. When applied by the operator, the brake operates on the rear axle and is used to provide tension in the train when tramming inby or when on a slope.

Two headlights are mounted on the outby end of the discharge vehicle. The headlights assist operation of the system in the outby direction.

The discharge vehicle operator has no operator's compartment or platform. Therefore, for operator protection, rear-wheel fenders are installed over the rear wheels.

An emergency stop switch is mounted on each side of the discharge vehicle. These emergency stop switches activate the CB-1 shunt trip coil, causing power to be interrupted over the entire train.

Tram Drive

The horsepower of the tram motor on the discharge vehicle is similar to the lead vehicle because of the additional weight of the bridge conveyor. The discharge vehicle tram motor is 7.5 hp instead of 5 hp, as used on the intermediate vehicles.

BRIDGE CONVEYOR

The bridge conveyor (fig. 5) mounts on a hitch on the outby end of the discharge vehicle, and transports mined material to the section panel belt. The outby end of the bridge conveyor rides on a bridge dolly that rides the panel belt frame. The bridge dolly is built to accommodate a 30-in-wide panel belt. The bridge conveyor has an active length of 22 ft. The bridge conveyor chain is powered by a 15-hp electric motor mounted on the bridge conveyor. A supply unit is positioned on the bridge conveyor for the discharge vehicle hydraulic steering power unit.

The bridge dolly frame contains four wheels that ride, two on each side, on angle iron guides attached to the side of the panel belt frame. The dolly frame also contains a crescent ring, in which a matching collar is anchored and allowed to angularly rotate. The outby end of the bridge conveyor pins to the collar. The bridge dolly is designed to run on a special belt tailpiece built by Long-Airdox,[6] which is described in Long-Airdox bulletin 5-973-1. Twenty-nine of the 9-ft sections are connected together to permit the bridge dolly to travel approximately 260 ft over the section panel belt. The 260-ft distance is required to permit penetration of the vehicle train through lateral and parallel entries during advances of the tailpiece. The tailpiece would consist of 28 increment sections and 1 tail section.

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[6]Reference to specific products does not imply endorsement by the U.S. Bureau of Mines.

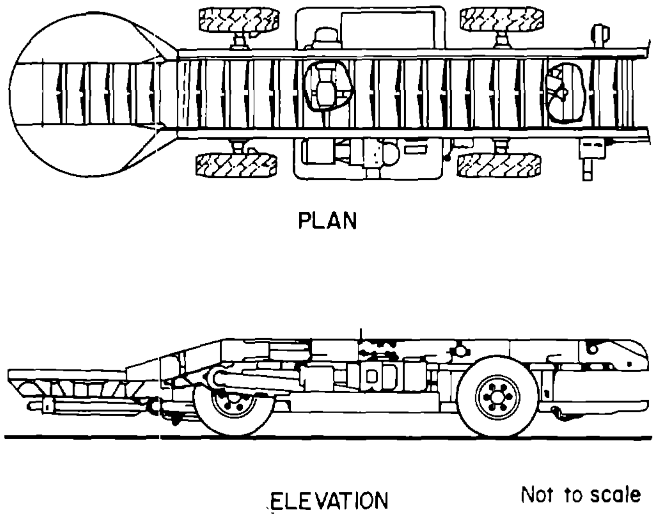

Figure 4.–Discharge vehicle.

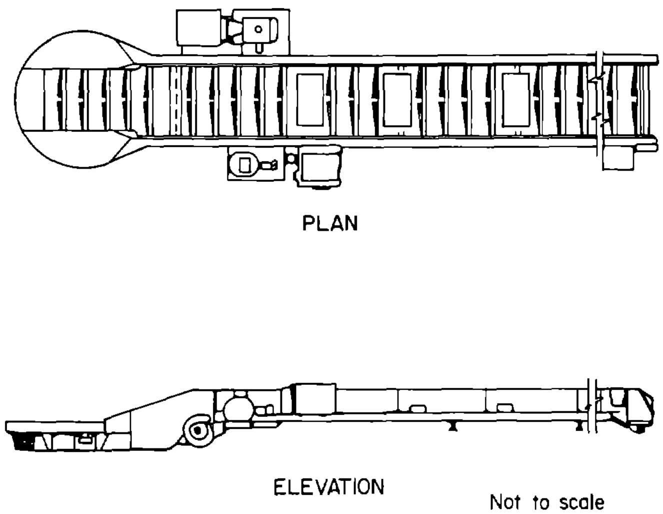

Figure 5.–Bridge conveyor.

SURFACE TESTING

JEFFREY MINING MACHINERY TESTING

During the 1977-79 period, JMMD conducted a series of tests on the MUCH system at its facility. The objectives of these tests were to verify the tracking-retracking ability of the system in 90° room-and-pillar configuration in an aboveground environment and ascertain the conveying capabilities of the system.

These tests revealed that each vehicle would overshoot the path of preceding vehicle and the MUCH system train would not track parallel to the section belt. The overshooting problem was solved by increasing the steering bar length by 2 in. The problem of the MUCH train not tracking parallel to the section belt was corrected with addition of hydraulic power for steering the discharge vehicle. The hydraulic power pack was added to the bridge conveyor, and the control valve and steering cylinder were added to the discharge vehicle. Other problems, such as jackknifing, were not solved although many modifications and/or changes were made in the system. The results of 1977-79 testing thus remained inconclusive.

In 1980, JMMD was asked to conduct additional testing on the MUCH system. The objectives of additional testing were to find out the MUCH system limits, its ability to operate and stop on inclines, and how it would track through S-turns on various slopes, over the rolls, and over loose and mud bottom. These tests were conducted at the Ohio Transportation Research Center (OTRC).

A three-entry mine plan was laid out at the OTRC. The simulated walls of entries and crosscuts were constructed with snow fence. The system would not track-retrack in the same manner at the OTRC as it had at the Jeffrey facility. It required modifications and changes for steering and wheel alignment, as well as addition of turnbuckles to all vehicles and brakes on the lead and discharge vehicles. After these modifications and changes were made, there was an improvement in track-retracking and conveyor discharge performance.

In 1982, additional design changes were made prior to sending the system to the Bureau's test facility at Bruceton, PA. These changes included installation of a chain between hopper and conveyor to limit travel between the adjacent vehicles, installation of rubber belting on hopper sideboards to stop spillage, and relocation of the conveyor speed switch.

METF TEST PROGRAM OVERVIEW

The MUCH system, consisting of 12 vehicles, bridge conveyor, and numerous spare parts, was received at the Bureau's METF in July 1983 for assembly and surface testing. Upon completion of assembly, all functions and safety devices were checked out and made operational; operator familiarization and training was undertaken prior to starting the test program.

Surface tests were conducted at the METF to verify and evaluate the performance of the MUCH system. Tests were divided into sequences to evaluate a particular subsystem or machine function. Modifications were made to the MUCH system to correct deficiencies noted during surface testing. A description of each test sequence is given in the following sections.

TRAM AND MANEUVERABILITY TRIALS

Tests were conducted in the METF equipment maneuverability trial area (EMTA) to determine the tramming capability of the system in a simulated mine environment, to define and correct any observed tramming or steering problems, and to demonstrate the reliability of the overall tram system.

Early in the surface test program, it became obvious that tramming and maneuvering the 12-vehicle system was no simple matter, especially within the confines of the EMTA and with inexperienced operators. The system was trammed into a continuous loop numerous times for haulage system trials, demonstrations, and noise level tests, and was also trammed in and through the simulated mine workings of the EMTA.

Figure 6 illustrates a number of operational procedures and maneuvering sequences that were undertaken early in the tramming trials. Initially, these maneuvers were difficult to perform successfully. Vehicles were contacting the ribs at corners as well as midrib. After operators performed a number of maneuvers, they were better able to gauge how an entry or crosscut must be entered, how tight a radius to turn, when to begin straightening the lead vehicle in a turn, how much rib clearance was needed, etc. Figures 6*B* through 6*F* illustrate typical maneuvers performed to place change the system from a far-right entry to the left crosscut in a simulated three-entry section. Figures 6*F* through 6*H* illustrate the return of the system to the far-right entry. In the step shown in figure 6*D*, it was necessary to reposition the outby end of the system to provide sufficient room to accommodate the limited turning ability of the discharge vehicle.

During these early tramming trials, more problems were experienced with the system making rib contact near the center of the pillar (area 1, fig. 6*H*) than at the pillar corners (area 2, fig. 6*H*). At the pillar corners, when making a 90° turn, the system tended to move away from the corner, but at midpillar, the system worked toward the rib and sometimes made contact. In general, the overall tracking of the system seemed inconsistent, but the tracking seemed better when turning sharp 90° turns than when making more gentle, larger radius turns.

Figure 6*I* shows the system in a gentle S-curve being trammed through the EMTA and out the east equipment door. This maneuver proved to be one of the most difficult encountered during the early tramming trials. Three pillar corners and the exit door frame were contacted numerous times while the system was being trammed out of the building.

It was much more difficult to move the system away from the rib or corner when the train was relatively straight than when the train was in a tight turn. This was expected, because the force available to pull a vehicle away from a rib or corner is proportional to the sine of the angle between the jammed and adjacent vehicle, and dependent upon the radius of curvature of the train.

The typical method used to unjam vehicles from contact with a pillar was only marginally effective when the train was in a gentle curve. The method consists of shutting off outby vehicles and tramming the remaining vehicles inby, then pulling vehicles away from the rib. Likewise, unjamming vehicles from contact with a pillar corner by removing tram power from outby vehicles and tramming remaining vehicles in the outby direction to push jammed vehicles away from the rib was ineffective when the train was in a gentle curve. These methods did work quite well, however, when the system was in a sharp turn.

A coal ramp approximately 2.5 ft high (fig. 6*J*) was built to observe system performance as vehicles trammed over it. The lead vehicle and first two intermediate vehicles were successfully trammed over the coal ramp and down the other side despite deep wheel penetration. No serious mechanical interferences were observed. Steering bars provided the necessary degrees of freedom in roll, pitch, and yaw axes. As the overall test program continued, a vehicle-to-vehicle interference problem that led to major vehicle frame modifications became obvious. This problem and subsequent modifications are described in the "System Modifications" section of this report.

As a result of these initial tramming trials, the steering-tracking system was examined to better in defining the theoretical capabilities of the system and correct any mechanical problems which might hinder tracking ability.

Steering System

A physical layout of the mechanical steering system (fig. 7) was constructed that reflected the linkage geometry of one steering axle and drawbar. This model led to a better understanding of the steering system and helped in defining the theoretical limits of the system. Figure 8 shows an overall layout of the system making two turns of different radii. During tramming of the system, it had been observed that the tracking seemed to be better when making tight turns than when making more gradual turns. The tracking layout drawing supports this observation. It can be seen that when the system is making a 12-ft-radius turn, the center of the turn is coincident for all vehicles making the turn.

A 43-ft turning radius is also shown in figure 8. It can be seen that the center of the turn wanders as the turn is entered by successive vehicles. It moves approximately 14 in with each new vehicle entering the turn. An enlarged view (fig. 9) shows the tracking error of each vehicle as the vehicles pass the same reference point. A tracking error of 5.25 in per vehicle occurs when turning at a 6° drawbar angle, which yields a 43-ft turning radius.

Once the tracking was defined from a theoretical standpoint, it was obvious that the system was not tracking as well as could be expected; therefore, the mechanical steering components of each vehicle were examined. The steering system freeplay was measured with a dial indicator on the end of each steering bar where it attached to the tie rods. Freeplay ranged from 0.03 to 0.050 in. The causes of excessive freeplay were found to be two loose tie rods, which were tightened, and 26 loose steering plates on 14 axles. The steering plates are bolted to the steering knuckles by four bolts on each plate. Once the bolts loosen, the plate is free to rotate as much as the clearance in the boltholes will allow, thus allowing excessive freeplay in the system. All loose steering plate bolts were torqued to specifications.

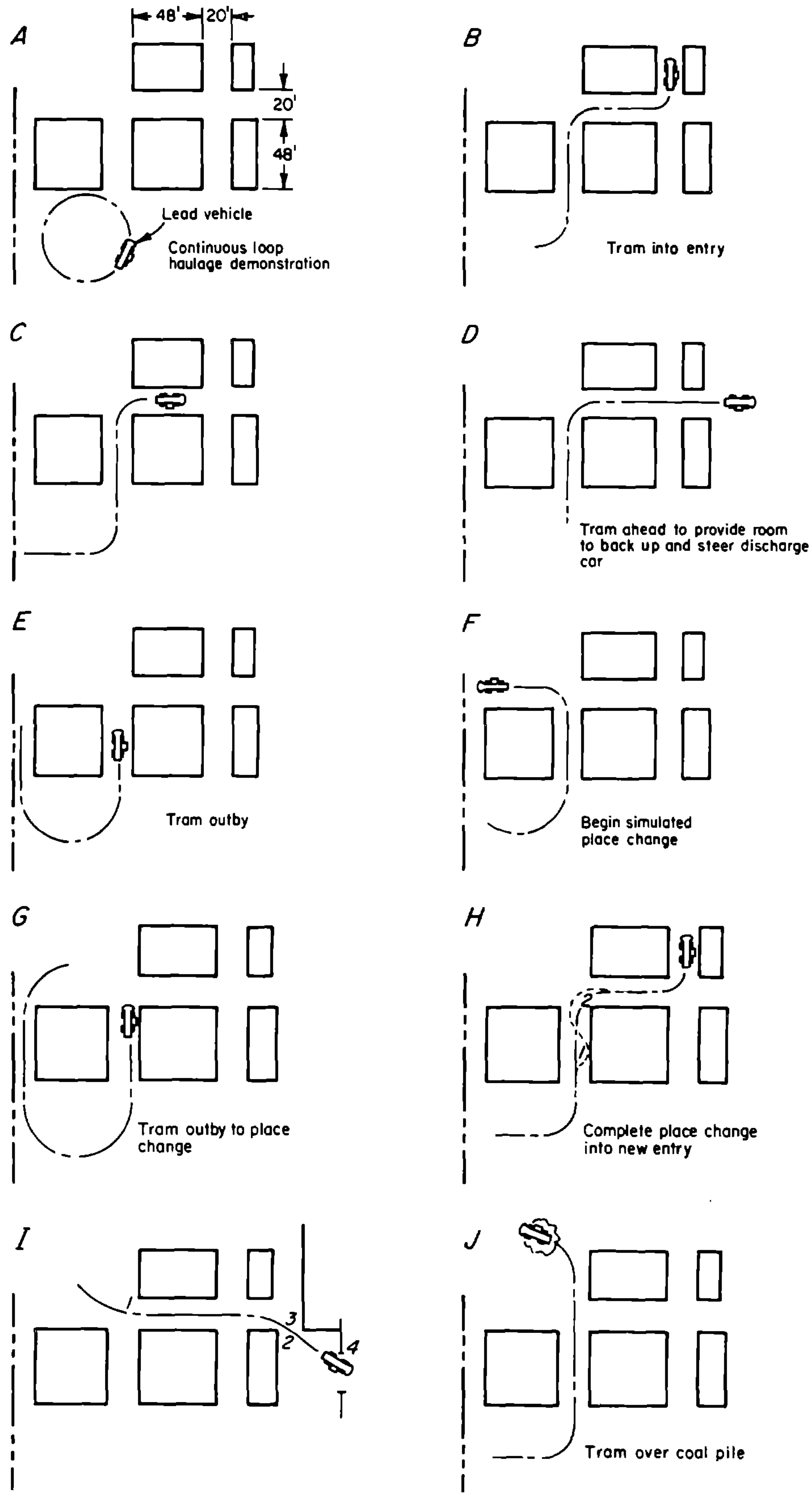

Figure 6.–MUCH system operational maneuvers in equipment maneuverability trial area.

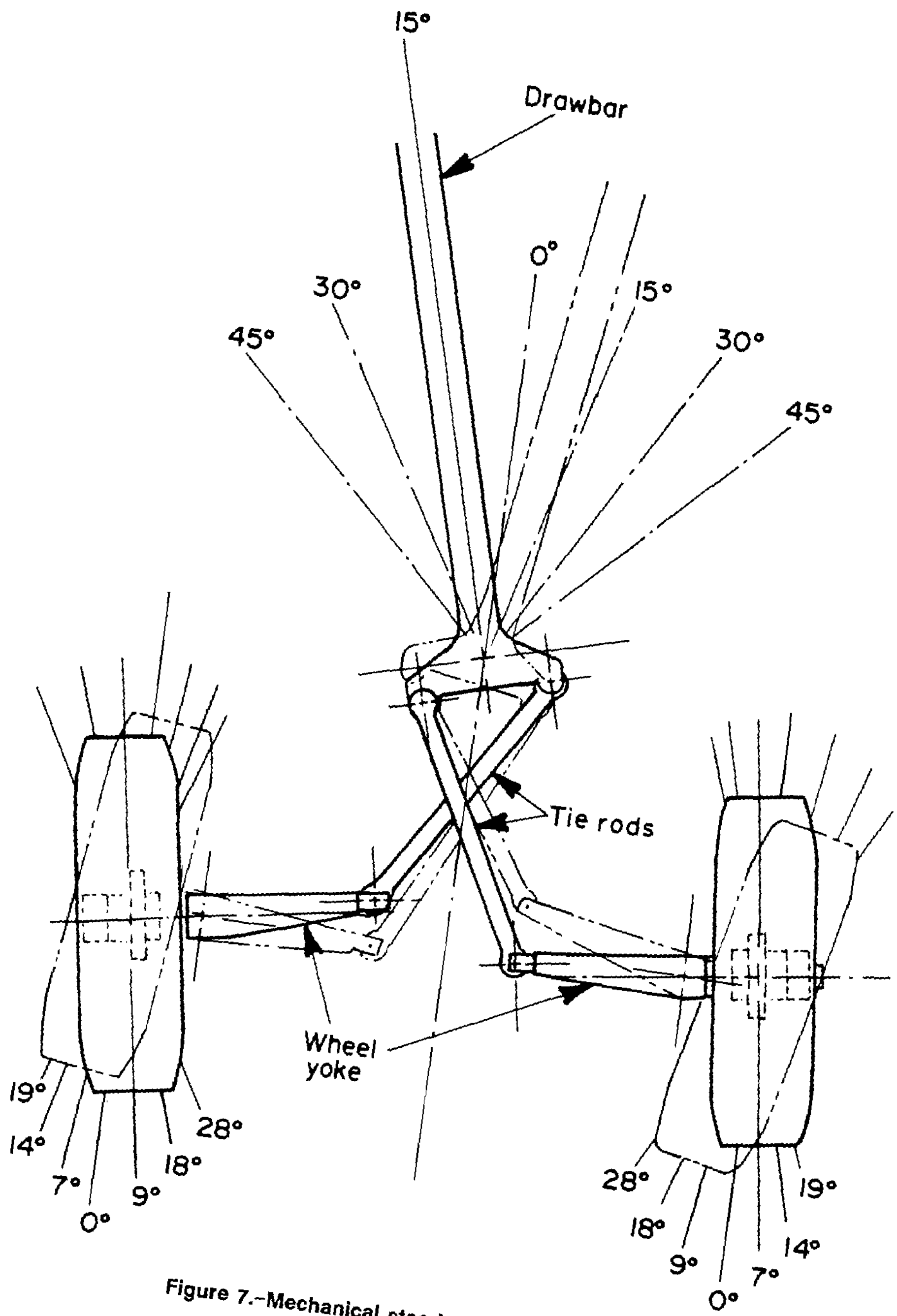

Figure 7.–Mechanical steering linkage geometry.

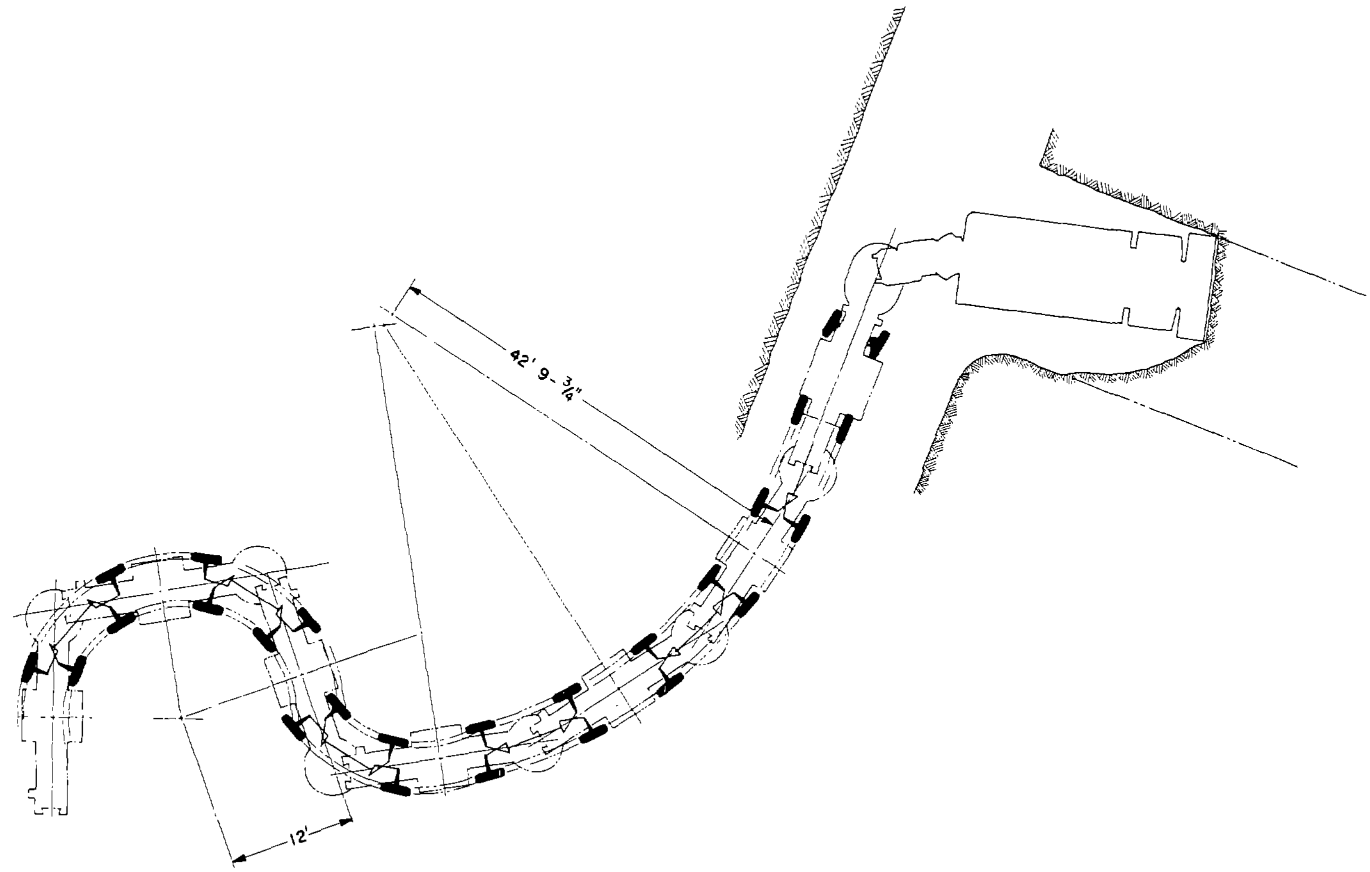

Figure 8.–Tracking layout, 12- and 43-ft turning radii.

The mechanical steering system on each vehicle was properly aligned per instructions in the operator's manual. Adjacent vehicles were trammed into a straight line and the drawbars were aligned along the axis of the vehicles. The tie rods were then adjusted so that each wheel was parallel to the drawbar center line.

The elimination of excessive freeplay in the steering system and the alignment of the individual wheels had a positive effect on the system tracking, but some problems were still evident, especially when tramming in the outby direction. Tramming outby was difficult because of the severely limited steering range of the discharge vehicle and the tendency of the vehicles in the trailing half of the system to track poorly and jackknife. Modifications were made to the discharge vehicle steering system to increase the available steering angles and improve component location. The modifications are described in the "System Modifications" section of this report.

The tendency of the trailing half of the system to track poorly and jackknife while tramming in the outby direction was investigated and the problem was corrected. The tram system is designed so that the electrical power to the tram motors in the last two trailing vehicles is automatically cut off during tramming to provide tension in the train. When the tram control in the lead vehicle is actuated, all tram motors receive power, but after approximately 1.1 s, the motors in the two trailing vehicles are automatically powered down to create drag on the vehicle train.

Upon investigation, a malfunctioning time-delay relay was found in the lead vehicle outby tram circuit that did not power the motor down after 1.1 s of operation. The unit was disassembled and repaired. Subsequently, tracking in the outby direction improved and jackknifing when tramming outby was reduced, because additional drag was provided by the lead vehicle.

Tracking-Retracking Tests

Objective

The objective of the tracking-retracking tests was to determine the ability of the MUCH system to successfully tram in both the inby and outby directions within the constraints of the EMTA simulated workings.

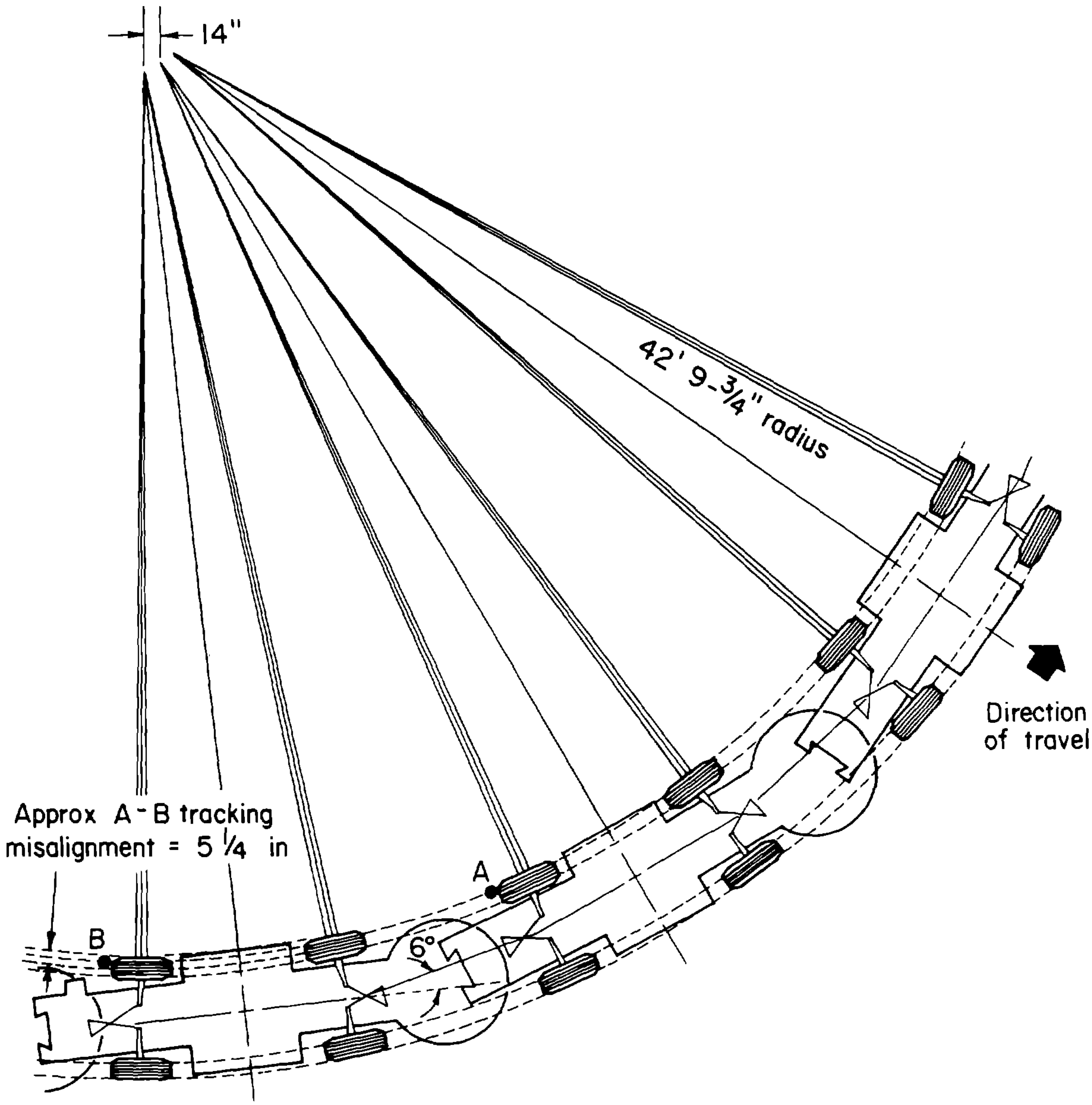

Figure 9.–Tracking capability, 43-ft turning radius.

Procedure

The EMTA was utilized to simulate an underground working area with 48- by 48-ft pillars and 20-ft-wide entries and crosscuts, as shown in figure 10. Ten data stations were located on the test course. Each data station consisted of a heavy string pulled taut across the entry approximately 5 ft above the floor. A 20-ft-long by 1-ft-wide strip of heavy kraft paper was suspended from each string. Uncapped felt-tip markers of assorted colors were attached to each of the 12 MUCH system vehicles, one per vehicle, at the same location on each vehicle. As the system was trammed through the test course, the markers were drawn across the kraft paper at each data station to permanently record the relative position of each vehicle at each station.

To conduct the tests, the entire MUCH system was trammed through the course of 10 data stations four times, twice in the inby direction and twice in the outby direction. After each traverse, the positions of both the lead and discharge vehicle, as indicated by the associated felt-tip mark on the kraft paper at each data station, were measured relative to both the left and right ribs. While tramming in the inby direction, the system was controlled by the steering of the lead vehicle. In the outby tram direction, the discharge vehicle steering controlled the system.

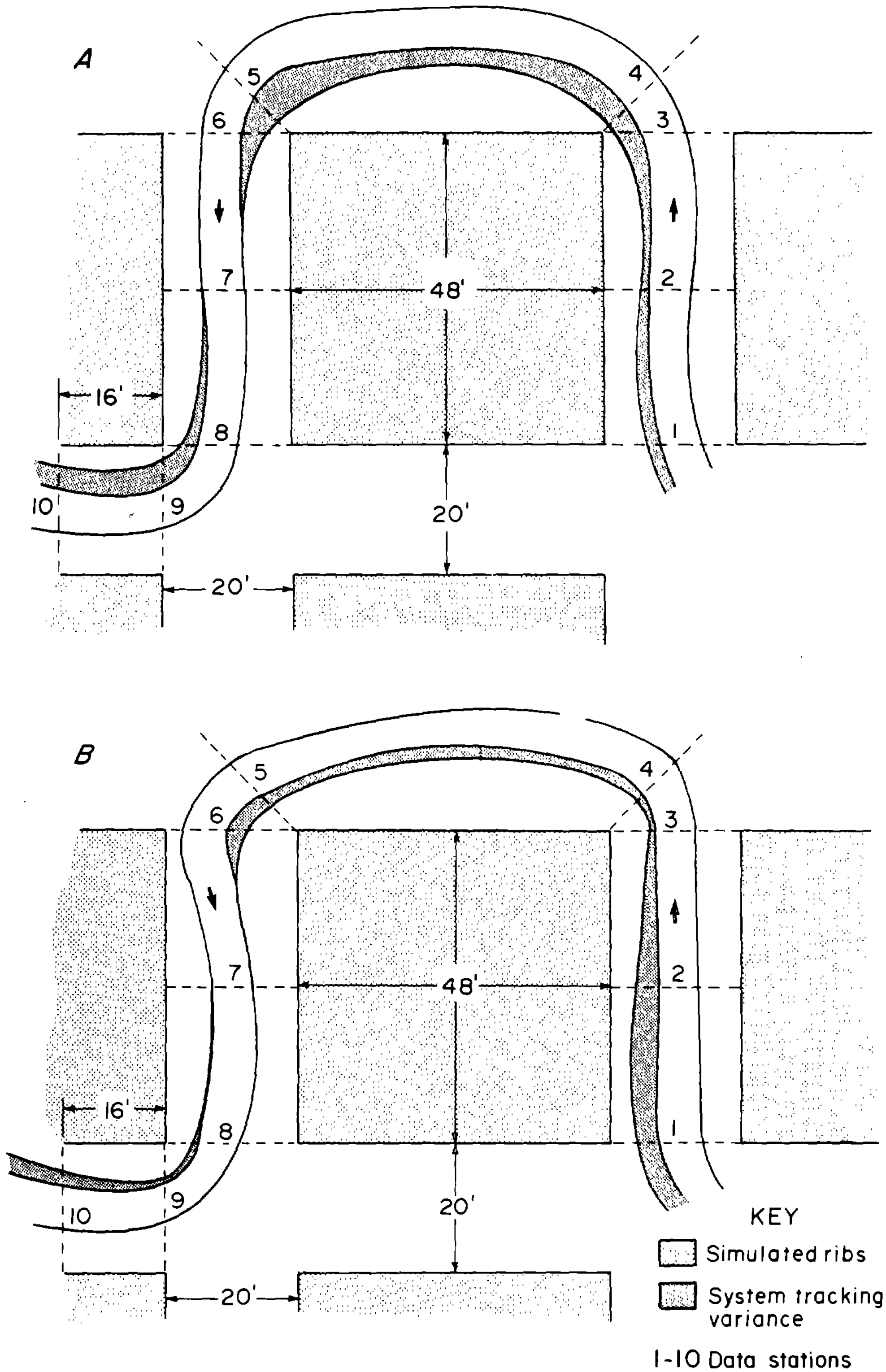

Figure 10.–Trial A (top) and trial B (bottom) inby.

Results and Discussion

The results of the tracking-retracking tests are given in table 1 and are shown in figures 10 and 11. Table 1 presents the analytical data acquired during the trials. Figures 10 and 11 present layout drawings of the test course that show the dimensions of the course, data station locations, direction of system travel, path of the operator-controlled vehicle through the course, and tracking error (variance) of the system through the course.

In table 1, the operating width column shows the maximum width required by the system at the associated data station. This width is equal to the width of the operator-controlled vehicle (6 ft 8 in) plus the maximum amount of tracking error (variance) shown by any other vehicle at the data station. The maximum variance column shows the maximum variance observed at each data station and whether the variance was to the left or the right of the position of the operator-controlled vehicle. The rib distance column shows the positioning of the required operating width of the system in reference to the left and right ribs at the data station. The left and right designations are relative to the operator position, which is always with the first vehicle facing the direction of travel.

During the tramming trials, the MUCH system tended to drift toward the inside radius of the turns. This tightening in the curves is probably due to the tramming resistance imposed by the automatic braking of the last two cars on the trailing end of the system. This braking keeps the system in tension, which keeps individual vehicles from jackknifing, but also tends to pull the vehicles toward the inside of a curve. The maximum variance that occurred while tramming in the inby direction was 68-in, which occurred at data station 5 during trial A (table 1, fig. 10).

Tramming in the outby direction was more difficult than tramming inby because of the more limited steering capacity of the discharge vehicle. During both trials, rib contact was made at the corner where data stations 5 and 6 intersect (fig. 11) when tramming in the outby direction. This corner was also the point of maximum system variance, 94- and 101-in, during trials A and B, respectively. During trial A, intermediate vehicle 4 contacted the corner. In trial B, as shown in figure 11, the system was trammed closer to the left rib prior to starting the turn to the right, and the turn was initiated sooner to create a smoother flowing curve. During this trial, only the last vehicle intersected the rib, although the overall variance was greater than in trial A.

Because of the lightweight fiberglass panels in the EMTA, the rib contact during tramming was a problem that prevented further tramming until the system was moved away from the rib. In a similar underground situation where rib contact would cause no damage, the system could have continued tramming while being guided by the rib. While rib contact is not desirable, it would not be as much of a problem underground as in the EMTA. In the author's opinion, the system tracked and retracked with enough consistency to operate satisfactorily in an underground mine of similar dimensions.

Simulated Production Cycle Trial

Objective

The objective of the simulated production cycle trial was to evaluate the tramming and tracking ability of the MUCH system during a simulated face production cycle.

Procedure

The EMTA was utilized to simulate an underground working area with 48- by 48-ft pillars, 20-ft-wide entries, and 20-ft-wide, 90° crosscuts. A National Mine Service shuttle car was utilized to simulate a continuous miner at a production face. The simulated production face was in the center entry of the three-entry simulated workings. The discharge vehicle of the MUCH system was in the left-hand entry; the 10 intermediate vehicles ranged from the discharge vehicle, through a 90° right crosscut to the center entry, through 90° left-hand turn to the lead vehicle, which was positioned directly behind the simulated miner at the face (fig. 12).

A continuous miner cut cycle was simulated by advancing the simulated miner and MUCH system 2 ft to simulate a sump and shear cycle, retreating 4 ft to prepare to cut the cusp and clean up, then advancing 6 ft to cut the cusp, clean up, and sump and shear. This cycle was repeated 10 times to simulate a 20-ft face advance. Upon the completion of this cut cycle, the simulated miner and MUCH system were backed up approximately 30 ft, the simulated miner was repositioned to the left-hand side of the face, and the cut cycle was repeated on the second lift. The overall functioning of the tram system, system alignment, and tire tracks were observed during the simulated cutting cycles.

Upon the completion of the simulated cutting cycles, the MUCH system was trammed to simulate a place change. The MUCH was trammed outby, down the middle entry, through the crosscut, and back into the left-hand entry. The system was then trammed straight ahead in the left-hand entry, past the crosscut and inby into the next 90° crosscut, and to the right to simulate a place change. The time required to complete with place change was measured.

Results and Discussion

The MUCH system trammed without problem and tracked fairly well during these trials. No rib contact, jackknifing, or tramming delays occurred during tests. The tracking of the system during the simulated cut cycles was good. As was evidenced in the previous tracking-retracking tests, the system tends to wander toward the inside radius of the 90° turns. This tendency was also observed during these trials but to a lesser degree.

The maximum tracking variance observed during these trials is shown in figure 13. This series of tire tracks shows the tracking variance at the 90° turn located at the intersection of the crosscut and middle entry. This "footprint" is the accumulation of tire tracks created while the system was trammed inby to the face, operated through a 20-ft cutting advance, a lift change, a second 20-ft cut advance, and then trammed outby during the place change. The total width of the track is 35 in. Therefore, the MUCH system required a total working width of 35 in plus the width of the lead vehicle to operate without rib contact. The total required operational width would be 9 ft 7 in at that location.

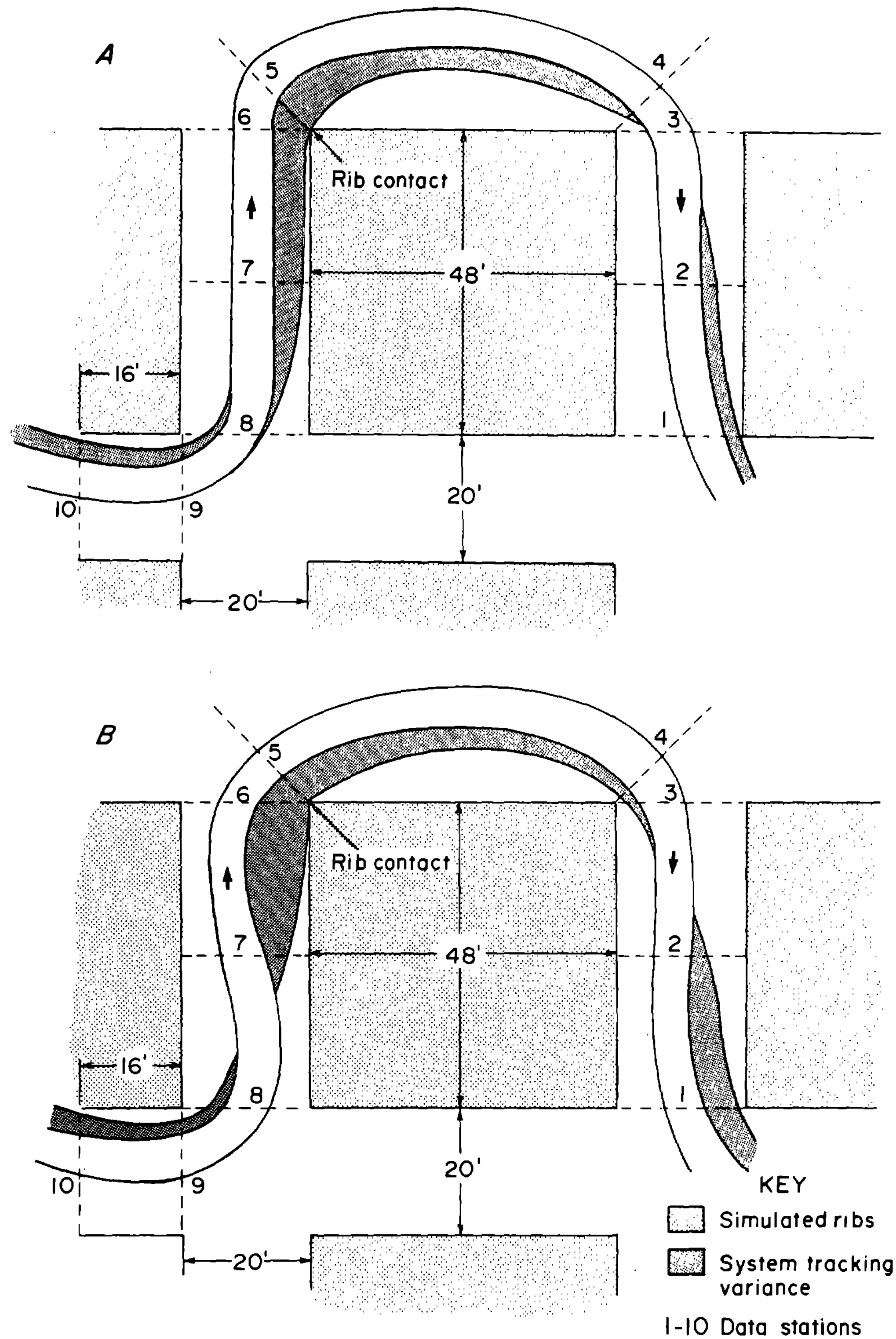

Figure 11.-Trial A (top) and trial B (bottom) outby.

Table 1.-MUCH tracking-retracking trials

Data station	Operating width		Max variance, in		Rib distance			
			Left	Right	Left		Right	
	ft	in			ft	in	ft	in
Trial A, inby:								
1	8	8	24	0	6	4	5	0
2	7	10	14	0	5	10	6	4
3	10	6	46	0	1	9	7	9
4	10	7	47	0	1	9	7	8
5	12	4	68	0	2	3	5	5
6	10	0	40	0	3	11	6	1
7	8	2	8	10	6	8	5	2
8	8	4	0	20	8	10	2	10
9	10	4	0	44	7	0	2	8
10	9	8	0	36	6	11	3	5
Trial B, inby:								
1	9	5	33	0	4	8	5	11
2	9	7	35	0	4	5	6	0
3	7	7	11	0	6	0	6	5
4	8	1	17	0	6	4	5	7
5	9	10	36	0	4	9	5	5
6	9	10	36	0	7	9	2	5
7	7	2	0	6	6	3	6	7
8	8	6	0	22	7	5	4	1
9	7	11	0	15	6	4	5	9
10	9	0	0	28	6	5	4	7
Trial A, outby:								
10	9	7	44	0	1	1	9	4
9	8	10	34	0	1	7	9	7
8	9	4	26	14	4	10	5	10
7	8	9	0	33	10	2	1	1
6	12	1	0	86	7	11	0	[1]0
5	13	10	0	94	6	2	0	[1]0
4	8	8	0	32	9	7	1	9
3	8	6	0	30	6	11	4	7
2	8	9	33	0	4	3	7	0
1	8	8	32	0	0	11	10	5
Trial B, outby:								
10	9	7	44	0	1	5	9	0
9	8	2	26	0	2	6	9	4
8	8	8	32	0	4	10	6	6
7	10	8	0	55	6	7	2	9
6	14	4	0	101	5	8	0	[1]0
5	11	3	0	70	8	9	0	[1]0
4	9	3	0	39	10	2	0	7
3	8	10	0	34	9	7	1	7
2	8	11	35	0	5	5	5	8
1	11	2	61	0	1	3	6	11

[1]Made contact with rib.

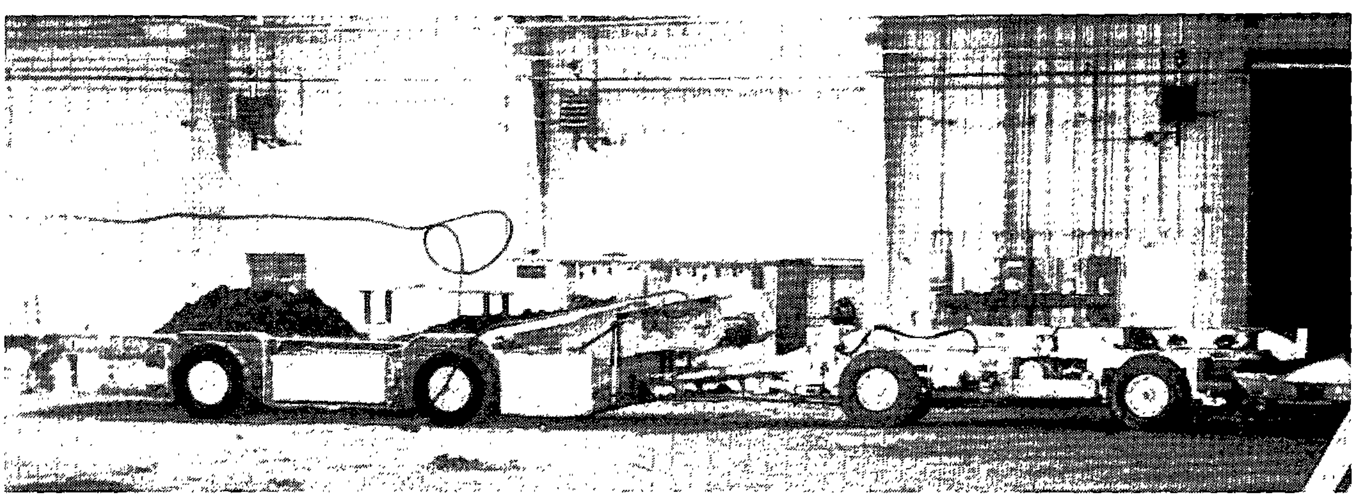

Figure 12.-Lead vehicle behind shuttle car.

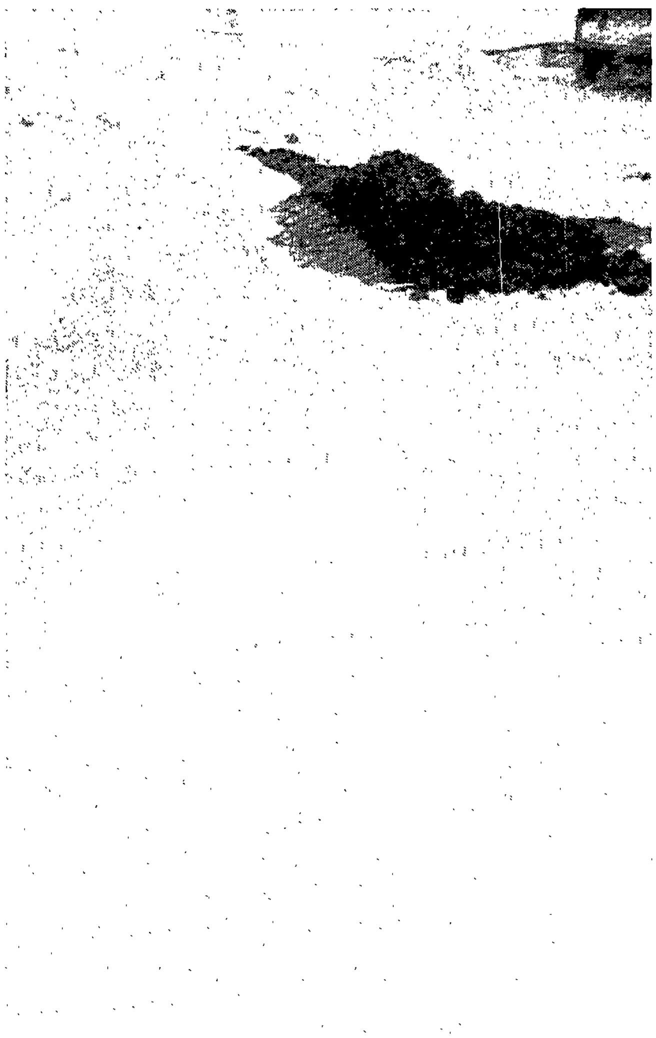

Figure 13.–Tracking variance.

The simulated place change that involved tramming the system outby from the middle entry, through a crosscut into the left-hand entry, then inby in the left-hand entry to the next crosscut and turning into the crosscut, covered a total tram distance of 245 ft. The outby tramming for the place change covered a distance of 155 ft and required a total time of 1 min 30 s, there was a pause of 10 s while the lead vehicle operator communicated with the discharge vehicle operator before starting to tram inby to complete the place change. The inby tramming covered a distance of 130 ft and required 1 min 40 s. The complete place change required a total of 3 min 20 s to complete and no problems occurred during the trial.

CONVEYOR SYSTEM TESTS

The main purpose of the MUCH system is to continuously convey coal cut by a continuous miner at the face to the section panel belt. Therefore, a large portion of the MUCH test program was devoted to testing, evaluation, repairs, and modification of the conveyor system.

Conveyor Noise-Level Survey

Objective

A conveyor noise level survey was conducted to determine noise exposure experienced by operators or miners working close to the MUCH system.

Procedure

The system was set up in a continuous loop behind the EMTA. The train was configured in a circle with the discharge vehicle dumping into the lead vehicle hopper while operating in the conveyor mode only. Tests were performed with the conveyors empty and with varying amounts of coal in the conveyors. Coal was a mixture of 2.5- by 2-in and 2- by 1.5-in sizes. All tests, except ambient noise measurement, were performed with the conveying system in operation.

Noise-level measurements were obtained in the center of the vehicle circle, around the outer perimeter of the vehicle train, and in the operator compartment. Attention was focused on the operator compartment of the lead vehicle.

A Bruel and Kjaer type 2205 handheld sound level meter (SLM) with a Bruel and Kjaer type 4117 piezoelectric microphone was used to perform the sound level measurements. Sound pressure level (SPL) was measured using the A-weighted network. This filter network patterns its response after the human ear. SPL meter response was in the slow mode.

All measurements were taken with the meter held away from the body to minimize the effects of noise reflection from the body. It should be noted, however, that performing a noise survey in an enclosed building, particularly with a large number of hard surfaces reverberating or reflecting noise, may cause SPL measurement errors. Larger objects with physical dimensions similar to the wavelength of the sound being measured are most likely to reflect noise and to be sources of error. Considering these limitations, the test area was adequate to perform such a general noise survey.

Results and Discussion

The noise level test configuration is shown in figure 14 and the survey results are given in table 2. The three tests are discussed in the following sections.

Test 1–Conveyor Off, Ambient Noise Level Measurement

The measured ambient noise level of 50 to 51 dBA in the test area introduced no error into the operating noise level measured. Ambient noise level greater than 10 dBA below the measured SPL will introduce no significant error in the data; therefore, no compensation for ambient noise level was made.

Table 2.–Conveyor noise test results (sound pressure levels)

Configuration	*dBA*
Conveyor off, ambient noise level	50 - 51
Conveyor operating, no coal in conveyor:	
Reading from center of conveyor loop (sound level meter rotated 360°)	98.3
Pass around (except at lead and discharge vehicle junction)	101 -102
Junction of lead and discharge vehicle	103 -104
Operator's ear level in lead vehicle	102
Conveyor operating with coal, level at lead vehicle operator's ear:	
Approximately 3.7 st/min	98
Approximately 7.4 st/min	97
Approximately 11.1 st/min	96.5 - 97.5

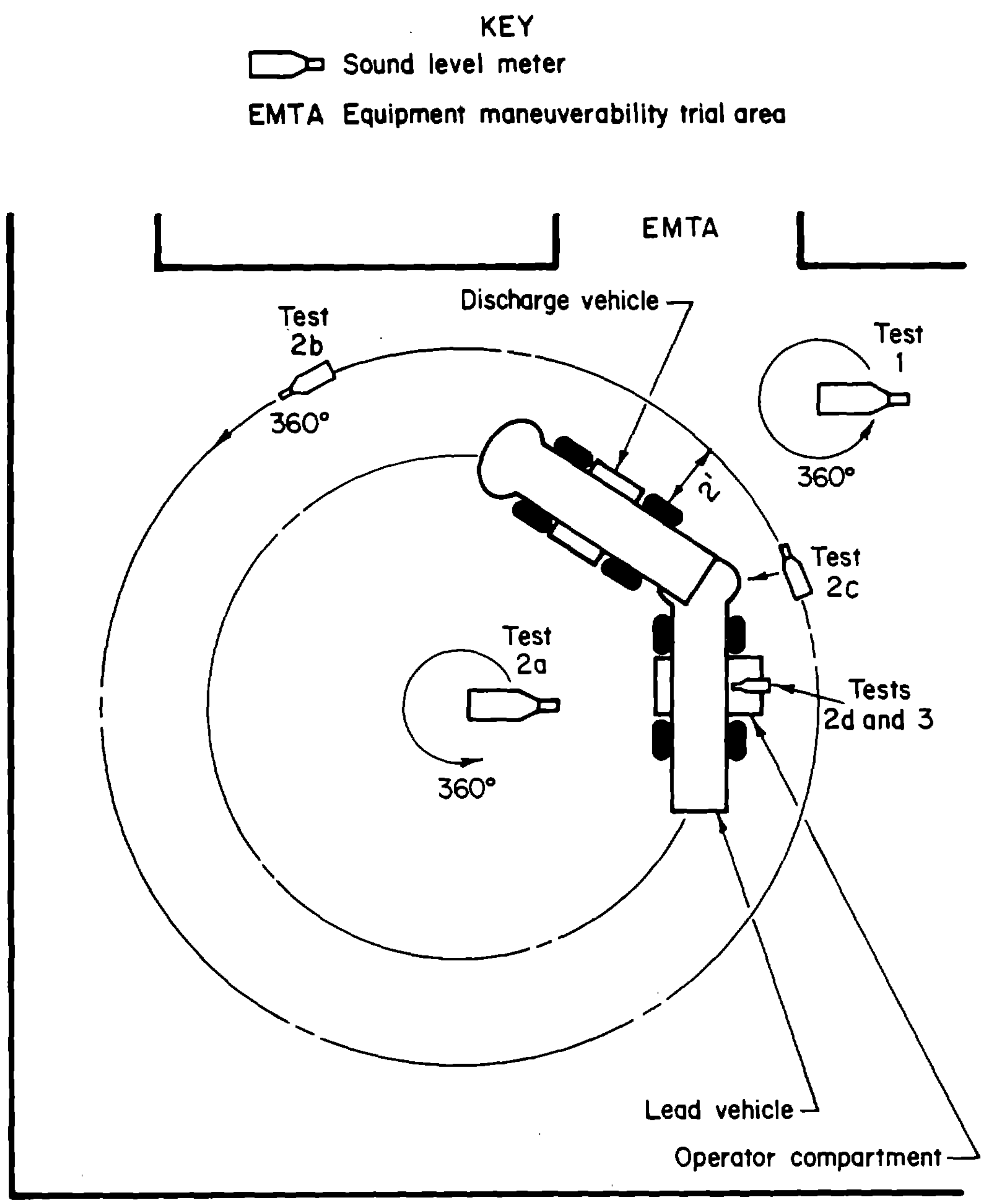

Figure 14.–Conveyor noise-level test configuration.

Test 2–Conveyor System On, No Coal

a. The SPL was measured in a 360° arc from a position in the center of the conveyor loop (fig. 14). The SLM was held away from the technician's body to minimize error, 4 to 5 ft above the ground and pointing toward the outside of the vehicle train circle. The measured SPL was 98.3 dBA.

b. A pass-by noise measurement was made by walking parallel to the train on the outside perimeter, approximately 2 ft from the vehicles. The SLM was held 4 to 5 ft above the floor and away from technician's body. The axis of the SLM was held parallel to the tangent of the MUCH circle, approximately 2 ft away from the vehicles. The SLM was held parallel to the vehicles to maintain geometric uniformity and to facilitate meter reading. The pass-by noise measurement is most closely associated with the noise level that a nearby miner would experience when in the same entry with the system. The measured SPL ranged from 101 to 102 dBA.

c. The SPL measured at the lead-discharge vehicle junction was 103 to 104 dBA in the pass-by mode as in test 2b. It was slightly more noisy than the junction of any other two vehicles.

d. The SPL in the operator's compartment at the operator's ear level was measured to determine noise level exposure of the lead vehicle operator. The operator compartment, at 102 dBA, was very noisy. Moving the microphone around the vicinity of the operator's ear showed minimal change in the readings, and thus increased the confidence in the SPL measurement. Accurate meter positioning was, therefore, found unnecessary.

In addition to customary hearing protection, a headset for communicating with the discharge vehicle operator would significantly attenuate the noise level to the operator.

Test 3–Conveyor System On, With Coal

A front-end loader was used to meter approximately 3 yd^3 of coal onto the MUCH system. The SPL at operator's ear level was measured while the system was conveying coal at approximately 3.7 st/min. The introduction of coal into the conveying system, even in small amounts, significantly reduced the noise level by damping the conveyor structure and absorbing acoustic energy. Noise level at the operator's ear was reduced by 4 dBA to 98 dBA.

The test was repeated, this time with 6 yd^3 of coal in the conveying system (7.4-st/min haulage rate). A 1-dBA noise level reduction was achieved by this doubling of the amount of coal on the conveyor (from 98 to 97 dBA).

The test was again repeated with 9 yd^3 of coal in the conveyors, an 11.1-st/min haulage rate. No further reduction in noise level over the 6-yd^3 test was achieved by the addition of 50 pct more coal (SPL, 96.5-97.5 dBA). It appeared that adding coal to the conveying system significantly reduced operator noise exposure, but further reduction in noise level was minimal beyond the 3.7-st/min haulage rate.

The noise levels measured indicate that some form of hearing protection will be required for the MUCH system operator to maintain an 8-h noise exposure of 90 dBA or below.

Conveyor Speed Test

Objective

A conveyor speed test was conducted to determine conveyor chain speed and motor rotational speed under both loaded and unloaded conditions and to determine the transport speed of discrete particles at various loading rates.

Procedure

A Micronta 63-5009 digital stopwatch was used to measure time required for the conveyor chain in each vehicle to make a complete cycle. When the system was loaded with coal, the conveyor chains were not visible; therefore, a General Radio 1531AB stroboscope was used to measure tailshaft rotational speed. After establishing a relationship between chain speed and tailshaft rotational speed, the chain speed was measured with a stroboscope. A stopwatch was also used to measure the transport speed of individual chunks of coal.

Results and Discussion

The conveyor surface speed was measured when empty and when loading 2 and 7 st/min (table 3). At 0 st/min (empty), the time required for the conveyor chain to make one complete pass through the vehicle was measured with a stopwatch. Three trials were conducted for each vehicle in the empty condition.

The conveyor chain lengths were 45 ft 5 in for the lead vehicle and 42 ft 5 in for all other vehicles. Dividing chain length by time yielded the average conveyor speed for each vehicle, which ranged from 270.7 to 273.7 ft/min, an average of 272.2 ft/min. The 272.2-ft/min average agrees within 2.7 pct of the published figure of 280 ft/min in the work cited in footnote 4.

Conveyor tailshaft speed was also measured to determine the speed relationship between the conveyor and tailshaft. Tailshaft rotational speed averaged 266.0 r/min under empty conditions. Thereafter, when coal was in the system, the conveyor speed was measured via tailshaft rotational speed. The ratio of conveyor speed to tailshaft rotational speed ($K_{r/min}$) was 1.025. Multiplying tailshaft rotational speed by $K_{r/min}$ gives the equivalent chain speed.

At a conveying rate of 2 st/min, conveyor chain speed averaged 268.3 ft/min. At 7 st/min, average chain speed was 270.9 ft/min. From 0 to 7 st/min, average conveyor speeds were within 1.6 pct of one another. At 2 st/min, average conveyor speed was 2.6 ft/min slower than at 7 st/min. This could be attributed to the small sample taken at 2 st/min (four vehicles) and the possibility of fines buildup on the conveyor deck during the trial.

Transport speed of individual particles was measured at various loading rates. Several chunks of coal and coalcrete (from 2 to 6 in) were timed as they passed through the system (table 4). With no coal in the system, the average transport speed was 253.5 ft/min. This figure is about 7 pct less than the average conveyor speed of 272.7 ft/min, reflecting the time required for the conveyor chain to pick up coal as it cascaded onto the next outby vehicle.

With the system conveying about 2 st/min, transport speed was about 3.2 ft/min less, or 250.3 ft/min. At 4 st/min, transport speed was 234.0 ft/min, but only one trial was performed.

Generally, transport speed will decrease somewhat at higher loading rates. From observation, it can be seen that some coal tends to ride over the chain flights as loading rate increases, thus tending to decrease the transport rate. It was also observed that as coal became finer from repeated handling, a greater percentage of it tended to ride up over chain flights, further reducing average transport speed. With relatively large pieces (2 to 6 in), however, no correlation could be seen between particle size and transport speed.

Another way of interpreting transport speed is to relate it to conveyor speed, providing a measure of conveying efficiency. At 0 st/min, transport speed is 253.5 ft/min and conveyor speed is 272.7 ft/min. Transport speed (or conveying efficiency) is 93 pct of the conveyor speed. Conversely, slippage would only be 7 pct. A 100-pct efficient system would transport coal at the same speed as the conveyor travels. At 2 st/min the effective conveyor transport efficiency is 93.3 pct, and at 4 st/min it is 87.3 pct.

Table 3.-Conveyor speeds

Vehicle	Time per conveyor cycle, s			Tailshaft speed, r/min	Av conveyor speed,[1] ft/min	Conveyor speed to tailshaft ratio ($K_{r/min}$), r/min
	Trial 1	Trial 2	Trial 3			
HAULAGE RATE: 0 st/min (SYSTEM EMPTY)-3/28/84						
Lead	10.04	9.88	10.02	266	273.1	1.027
1	9.49	9.31	9.39	265	270.7	1.022
2	9.34	9.54	9.30	266	271.0	1.019
3	9.30	9.33	9.29	266	273.4	1.028
4	9.27	9.34	9.35	266	273.1	1.027
5	9.39	9.31	9.26	266	273.1	1.027
6	9.27	9.30	9.32	267	273.7	1.025
7	9.43	9.37	9.28	266	271.9	1.022
8	9.31	9.28	9.30	266	273.7	1.029
9	9.30	9.34	9.25	265	273.7	1.033
10	9.31	9.35	9.35	266	272.5	1.024
Discharge	9.26	9.32	9.42	267	272.8	1.022
Average	NAp	NAp	NAp	266.0	272.7	1.025
HAULAGE RATE: 2 st/min-3/14/84						
8	NAp	NAp	NAp	264	270.6	NAp
9	NAp	NAp	NAp	257	263.4	NAp
10	NAp	NAp	NAp	263	269.6	NAp
Discharge	NAp	NAp	NAp	263	269.6	NAp
Average	NAp	NAp	NAp	261.8	268.3	NAp
HAULAGE RATE: 7 st/min-3/30/84						
Lead	NAp	NAp	NAp	262	268.6	NAp
1	NAp	NAp	NAp	264	270.6	NAp
2	NAp	NAp	NAp	264	270.6	NAp
3	NAp	NAp	NAp	264	270.6	NAp
4	NAp	NAp	NAp	264	270.6	NAp
5	NAp	NAp	NAp	263	269.6	NAp
6	NAp	NAp	NAp	263	269.6	NAp
7	NAp	NAp	NAp	268	274.7	NAp
8	NAp	NAp	NAp	266	272.7	NAp
9	NAp	NAp	NAp	263	269.6	NAp
10	NAp	NAp	NAp	266	272.2	NAp
Discharge	NAp	NAp	NAp	264	270.6	NAp
Av	NAp	NAp	NAp	264.3	270.9	NAp

NAp Not applicable.
[1]Calculated from total length of conveyor chain: lead vehicle-45 ft 5 in, all other vehicles-42 ft 5 in.

Table 4.–Particle transport speed

Particle diam, in	Time,[1] s Trial 1	Trial 2	Trial 3	Transport speed, ft/min Av	Av for all sizes at loading rate
0-st/min haulage:					
4	53.40	53.44	54.09	255.9	
4	54.69	NT	NT	250.0	253.5
6	53.87	54.55	54.89	252.1	
2-st/min haulage:					
2	57.10	NT	NT	240.4	
5	53.02	54.99	NT	254.1	250.3
6	54.35	NT	NT	252.5	
4-st/min haulage:					
4	58.66	NT	NT	234.0	234.0

NT No trial.
[1]Transport time from lead vehicle hopper to end of discharge vehicle.

Conveyor Time Sequencing

Objective

A conveyor time sequencing test was conducted to measure the startup sequence time required to start the conveyor system.

Procedure

Time required to power up all conveyors sequentially was recorded under loaded and unloaded conditions. During testing, with haulage rate established and coal evenly distributed throughout the MUCH system, the time required to restart the system was measured by the lead vehicle operator using a digital stopwatch. Time was measured from the time the conveyor start switch was pushed until the lead vehicle conveyor system started.

Results and Discussion

The time required to start up the entire train of conveyors was recorded at haulage rates of 0, 2, 3, and 4 st/min. From data in table 5, it can be seen that the average startup ranged from 1.36 s at no load (0 st/min) to 1.86 s at 4 st/min, generally increasing as the load on the conveyor increased.

On March 14, 1984, it took 2.73 s to start up the system at 2 st/min, an apparent anomaly. This was the first date of testing, and solidified coal had been in the idle conveyors for a period of several days before the testing started. The additional load imposed by this material caused a significantly longer startup time.

Coal Conveying Tests

Numerous coal conveying tests were conducted during the program and all but the final test were followed by system modifications and improvements in an effort to establish the coal-conveying capability of the system and to achieve a 95-pct system availability while conveying at a rate of 8 st/min over an 8-h shift.

March 14 to April 12, 1984

Test Configuration

A continuous haulage loop of the MUCH system was formed by having the discharge vehicle convey coal onto the 30-ft Long-Airdox belt structure and into the hopper-feeder-bolter (HFB). (Because of its ability to provide surge capacity, the HFB served as the coal-entry point for new coal being added to the system.) The HFB in turn dumped onto a 50-ft Long-Airdox belt structure, modified to accept the Ramsey Engineering belt weight scale, that loaded coal onto the lead vehicle of the MUCH system, thus completing the loop, as shown in figures 15 and 16.

Coal used during the first part of testing was a mixture of 2.5- by 2-in and 2- by 1.5-in coal acquired from the Bureau's Hydraulic Transport Research Facility (HTRF). The second portion of conveyor testing was performed with run-of-mine (ROM) coal from the Bureau's research mine.

Table 5.–Conveyor startup time

Trial	Date	Startup time, s Sequenced	Av
0-st/min haulage:			
1		1.51	
2	3/29/84	1.34	1.36
3		1.32	
4		1.28	
2-st/min haulage:			
1		[1]2.83	
2	3/14/84	[1]2.63	2.73
3		1.55	
4		1.59	
5	3/15/84	1.85	
6		1.79	1.72
7	3/30/84	1.86	
8		1.66	
3-st/min haulage:			
1		1.84	
2	4/03/84	1.81	1.87
3		1.95	
4-st/min haulage:			
1	3/30/84	1.89	1.86
2		1.82	

[1]Measured during initial startup of conveyor testing. Solidified coal was present in conveyors, which increased startup time by a significant margin.

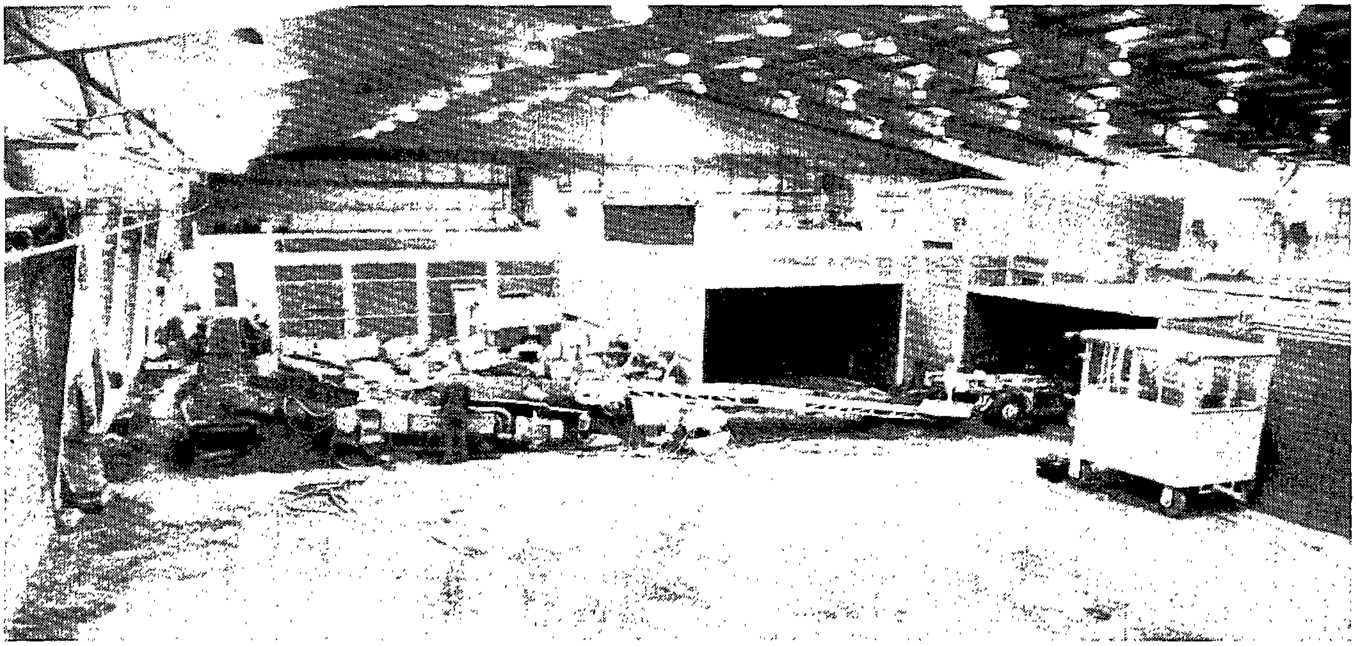

Figure 15.–Conveyor test configuration.

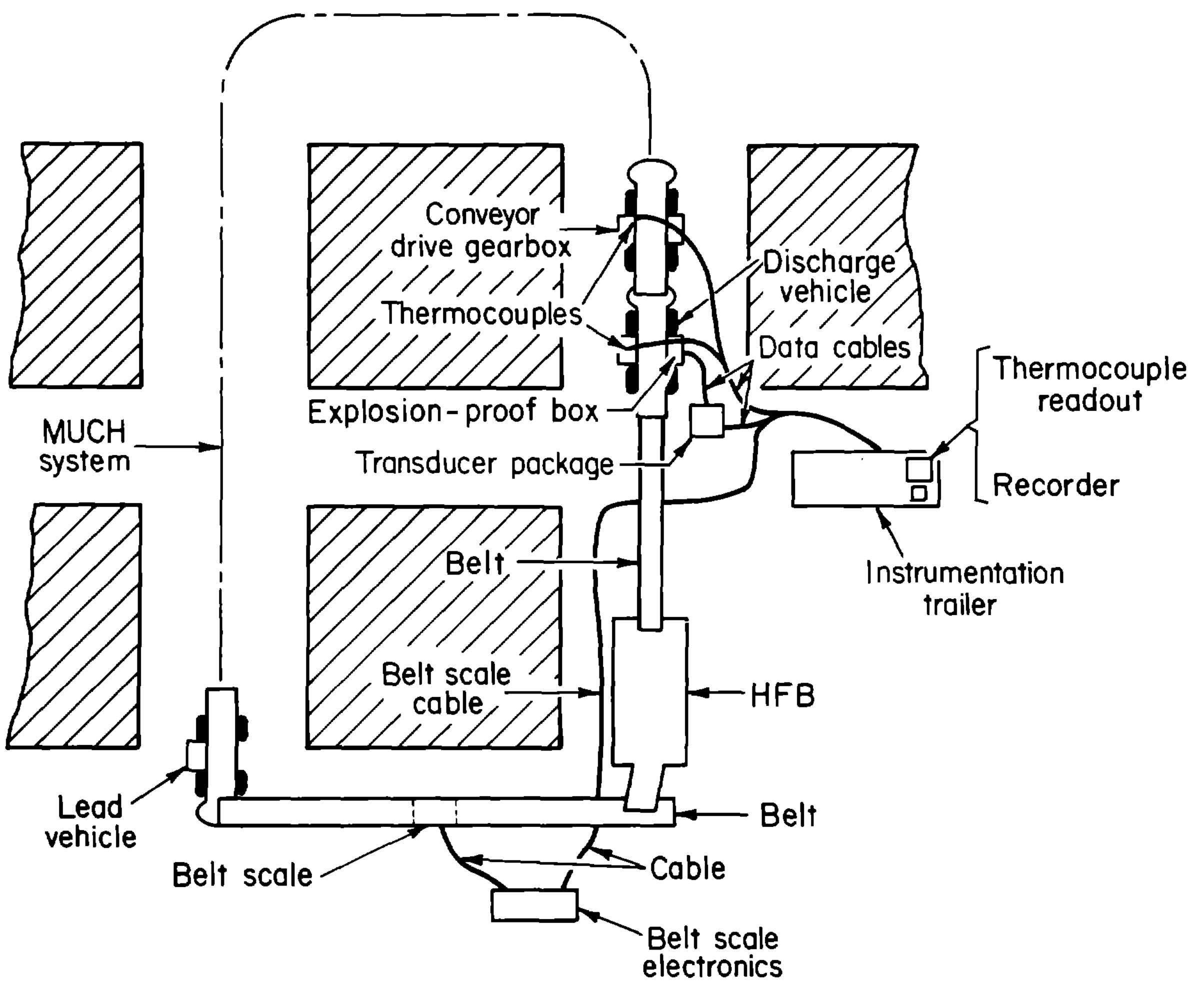

Figure 16.–Test configuration and instrumentation system.

Instrumentation

To more effectively evaluate the conveying system, pertinent electrical and mechanical parameters were measured and recorded during conveying tests. Shown in figure 16 are the layout and location of the instrument systems. The following sections describe parameters measured and instrumentation utilized.

Total Conveying System Power

Total system power was measured to provide an evaluation of power requirements of the system, particularly under various operating conditions.

True power, in kilowatts, of all 12 vehicles was measured with a Rochester Instrument Systems watt transducer. Current to the watt transducer was supplied by two 400:5 current transducers, one on each of two phases of the incoming three-phase power. Current transformers were placed in the discharge vehicle's explosion-proof box on the load side of the main breaker, thus coupling transducers to line power and reducing current by a factor of 80. Voltage leads from the watt transducers were connected to the load side of the main breaker.

Discharge Vehicle Power and Current

Individual vehicle energy requirements are not necessarily one-twelfth of the total system power because of differences in instantaneous loading rate from one vehicle to another and also for the conditions that exists on a specific vehicle such as plugging or coal fines buildup on the conveyor deck. Therefore, the electrical power and current requirements of the discharge vehicle were singularly measured. A Rochester Instrument Systems watt transducer, identical to that measuring total system power, was used to measure the discharge vehicle power and a Transdata model 10CS501 current transducer was used to measure the current required by the discharge vehicle conveyor motor. Two 50:5 current transformers were placed in two of three conductors supplying power to the conveyor motor contactor, thereby coupling the transducers to live current and reducing current by a factor of 10. Voltage leads were connected to the load side of the main breaker.

System Voltage

To monitor regulation of the power center and to ensure that the system operating voltage remained within acceptable limits, the system voltage was measured using a potential transformer and a potential transducer. The 550:120 voltage tap was used on the Trenco TR12182 potential transformer to supply an acceptable voltage level to a Rochester Instrument Systems 10PS101. The transformer primary was connected directly across the line on the load side of the main breaker. Output of this potential transducer was proportional to system voltage.

Haulage Rate

Perhaps the most important variable measured in the test program was haulage rate. It is the independent variable to which most other dependent variables are related.

A Ramsey Engineering, model No. 10-20/40-20, belt weigh scale was used to measure instantaneous loading rate. Belt-scale electronics included an integrator to total the number of tons of coal that was conveyed.

Conveyor-Reducer Drive Temperature

Lubricant temperature of two conveyor-reducer drive gearboxes was measured to evaluate suitability of the gearbox for loads imposed on the system. Two thermocouple junctions were made from Type K thermocouple wire. Each was entered through a sealed fitting into the gearbox sump in the discharge and the 10th intermediate vehicles. An Omega model 2168 digital thermocouple readout was used to measure gearbox temperature.

A Gould model 481 eight-channel strip-chart recorder was used to simultaneously record total system power, discharge vehicle power and current, system voltage, and haulage rate.

Procedure

Instrument Calibrations

All sensors being recorded on the Gould strip-chart recorder were physically calibrated prior to the test program. In addition, prior to each day's run, the Gould recorder was calibrated with approximate voltages for each channel to simulate proper sensor stimuli using a General Resistance DAS66AX Dial-a-Source.

Physical calibrations of the current, potential, and watt transducers were performed using a calibration device consisting of a bank of resistors (unity power factor) arranged to provide three-phase voltage and current to the transducers. Voltage devices were connected directly across the line. Current was measured using a Fluke 80J-10 current shunt and Fluke 8600 DMM (digital multimeter). Voltage was also measured with the Fluke 8600 DMM. Transducer output was adjusted to the proper voltage level appropriate for input stimuli. Gain of the Gould recorder was adjusted to give appropriate scale deflection. These transducer voltages were inserted into the Gould recorder, via the Dial-a-Source, on a daily basis during the test program to maintain recorder calibration.

The Ramsey belt scale was calibrated using weigh chains designed for that purpose. Calibration was performed with the belt running at normal operating speed with weigh chains placed on the moving belt. A three-point calibration was performed at 0-, 20-, and 40-pct full scale (0-, 4-, and 8-st/min loading rate). Gould recorder gain was also set to the appropriate level to indicate proper haulage rate. As with other transducers, the recorder was calibrated daily using a simulated signal from the Dial-a-Source.

Thermocouples were checked at ice point (32° F) using an ice bath and at boiling point (212° F) using a hotplate and water.

Haulage Test

A mixture of 2.5- by 2-in and 2- by 1.5-in coal from the HTRF was used in conveying tests from March 14 to April 5, 1984. For tests conducted April 5-12, 1984, ROM coal from the Bureau research mine was utilized.

On March 14, 1984, first trial of the haulage test was performed. A Clark front-end loader was used to load coal into the HFB. Steady-state loading rate was adjusted from 0 to 6 st/min over the test period from March 14 through April 12, 1984.

Total system power, discharge vehicle conveyor motor current and power, system voltage, and loading rate were recorded on the Gould strip-chart recorder for all loading trials.

On each day the testing was performed, the Gould recorder was allowed to warm up for 30 min and then it was calibrated with the Dial-a-Source. Belt-scale totalizer readings were recorded at the beginning and at the end of testing to determine the total amount of coal transported each day. A log was kept of all significant events, breakdowns, and repairs during the test period. The log was voice-recorded on a microcassette tape recorder and later transcribed.

Conveyor gearbox temperature was measured before the start of each test and every 10 min thereafter for the first 30 min. Subsequent readings were taken every 30 min or whenever the system was shut down.

After completion of testing, coal carryback loss lying beneath each vehicle's conveyor drive shaft was weighed with a 1,000-lb capacity balance beam platform scale.

Results and Discussion

One objective of the MUCH system test plan was to demonstrate reliability of the conveying system by operating for one full shift at 8 st/min, with no more than 24 min of downtime (95 pct availability). Because of a variety of recurring problems, no more than 136 min of running time was achieved in any one day (28 pct availability). Table 6 summarizes the amount of test time at the listed haulage rates.

Table 6.–Haulage test summary

Av haulage rate, st/min	*Test time, min*
0 (system empty)	197.0
1-2	244.2
3	65.0
4	23.9
5	29.0
6	1.5
Total	[1]560.6

[1]9 h 20.6 min.

The maximum average conveying rate reached was 6 st/min. Conveyor tests were performed in 13 days from March 14 to April 12, 1984. Total test time was 560.6 min. Total run time achieved above 4 st/min was 30.5 min. During the test period, 682.1 st of coal was conveyed. Average loading rate for the entire period was 1.22 st/min. Appendix B contains a conveyor test breakdown and repair log for this test period.

The system test was started on March 14, 1984, at 2 st/min, but after 2 days, only 65 min of run time had been achieved because of breakdowns. The system was then allowed to run empty for 2 days to determine if failures were load dependent. After the system had cleaned itself out, the breakdowns ceased, but then it took a period of time for the coal to be cleaned out from beneath conveyor decks. It appeared that after rehandling the coal many times and by adding water with the water sprays, the coal became a fine, granular, dense mixture, which required considerably more energy to convey than dry, coarse coal. This wet mixture plugged up around conveyor sprockets, under the conveyor deck, and even on top of the conveyor deck.

As coal was again added to the system, mechanical and electrical problems began to reoccur. Even as the loading rate was kept constant at 2 st/min, the rate of system malfunction increased with time. It is hypothesized that this was due to increased buildup of wet fines everywhere in the system.

Shown in table 7 is a summary of results for these conveyor tests. Discharge vehicle current, discharge vehicle power, and total system power are shown for loading rates from 0 to 5 st/min. Figure 17 shows typical Gould strip-chart traces of real-time data.

System power consumption ranged from 45 kW (60.3 hp) at no load (0 st/min) to 166 kW (222.5 hp) at 5 st/min. Average power consumption per vehicle 3.8 kW (5 hp) and 13.8 kW (18.4 hp) at 0 and 5 st/min, respectively. Measurement of discharge vehicle power indicated that its power consumption ranged from 7.5 kW (10.1 hp) at no load to 10.0 kW (13.4 hp) at 2.2 st/min. This was greater than the discharge vehicle power measured at 5 st/min (8.9 kW or 11.9 hp), because wetted coal fines had built up on the conveyor deck to a depth of 4.5 in. Figure 18 shows the extent of fines buildup on the conveyor deck. This buildup increased conveyor chain tension significantly and increased the coefficient of friction, requiring much greater power to convey the coal. Such buildup on the conveyor deck was also a problem with intermediate vehicle 5.

Once the coal fines buildup was removed from the deck, discharge vehicle power went down to 7.8 kW (10.5 hp), a reduction of 22 pct. Likewise, discharge motor current decreased from 21 to 14.8 A (25 pct decrease).

Under no-load conditions, the system consumed up to 36 pct of the power required at 5 st/min. This was due mainly to the wet fines plugging up the conveying system, particularly material carried back under the conveyors in the vehicle hoppers. On March 16, 1984, after the system became plugged with coal and was emptied, residual material caused the power consumption to remain artificially high at 68 kW (91.1 hp), even though no coal was being conveyed. On March 20, 1984, after allowing the system to run a period of time to purge itself, the power dropped 18 pct to 56 kW (75 hp). At that point, water sprays were turned on. After wetting the conveyor chains and deck for approximately 5 min, power consumption dropped another 18 pct to 46 kW (61.7 hp).

One measure of conveying efficiency is the amount of energy required to convey material a given distance, expressed as specific energy in kilowatt hours per short ton.

$$E_{sp} = \frac{P_t}{Q\ (60)},$$

where E_{sp} = specific energy, kW·h/st,

P_t = total system power, kW,

and Q = loading rate, st/min.

In terms of energy requirements, most efficient conveying is achieved at the higher loading rates. At 4 st/min, specific energy was 0.41 kW·h/st conveyed. At the 1.8-st/min loading rate on March 15, 1984, the specific energy was 1.36 kW·h/st. Poorer efficiency was due to two factors: (1) material had built up on conveyor decks and underneath conveyors, and (2) there is a certain tare power requirement to overcome system friction under no-load conditions.

Table 7.–MUCH haulage testing data summary–electrical parameters

Haulage rate, st/min	Discharge conveyor		System power, kW	Total system voltage, V	Coal conveying specific energy,[1] kW·h/(st·ft)	Comments
	Current, A	Power, kW				
3/14/84:						
2.4	13.5	NA	108	465	0.75	Material had been on conveyor decks for several weeks prior to system startup.
2.3	15.5	NA				
1.0	13.0	NA	117	465	.85	Values averaged over data window.
3.1	17.7	NA				
3/15/84:						
1.8	17.0	NA				
0.7	14.0	NA	147	460	1.36	Material buildup on conveyor decks, mechanical problems occurred because of high loads.
4.0	19.8	NA				
3/16/84: 1.6	15.2	8.4	118	465	1.23	15.5 st coal conveyed.
3/20/84:						
0	11.8	7.8	68	475	NAp	At 2:11 p.m.
0	12.2	8.0	60	480	NAp	At 3:11 p.m.
3/28/84:						
0	12.0	7.8	56	485	NAp	Before addition of water, 2:51 p.m.
0	10.2	7.5	46	485	NAp	After addition of water, 2:56 p.m.
0	12.2	7.8	45	485	NAp	At 3:14 p.m.
3/29/84: 2.0	13.0	7.6	87	475	.73	38.3 st coal conveyed.
3/30/84:						
2.0	11.5	7.4	81	475	.68	196.8 st coal conveyed.
4.0	13.0	7.4	99	470	.41	
4/05/84: 2.2	21.0	10.0	138	460	1.05	Discharge and intermediate vehicle 5 conveyor decks were plugged.
4/06/84: 3.0	14.8	10.5	138	460	.77	After-discharge and intermediate vehicle 5 conveyor decks cleaned. 59.8 st coal conveyed.
4/09/84: 5.0	18.0	8.9	166	455	.55	81.7 st coal conveyed.
4/12/84: 0	NAp	NAp	NAp	NAp	NAp	Mechanical failures and electrical problems effected the recorded data. 24.9 st coal conveyed.

NA Not available.
NAp Not applicable.
[1]Total mean–0.84 kW·h/(st·ft); standard deviation, 0.30 kW·h/(st·ft).

NOTE.–In tests conducted April 3 (44.9 st coal conveyed) and April 4 (46.6 st coal conveyed), electrical and mechanical problems were encountered. No operating data were obtained.

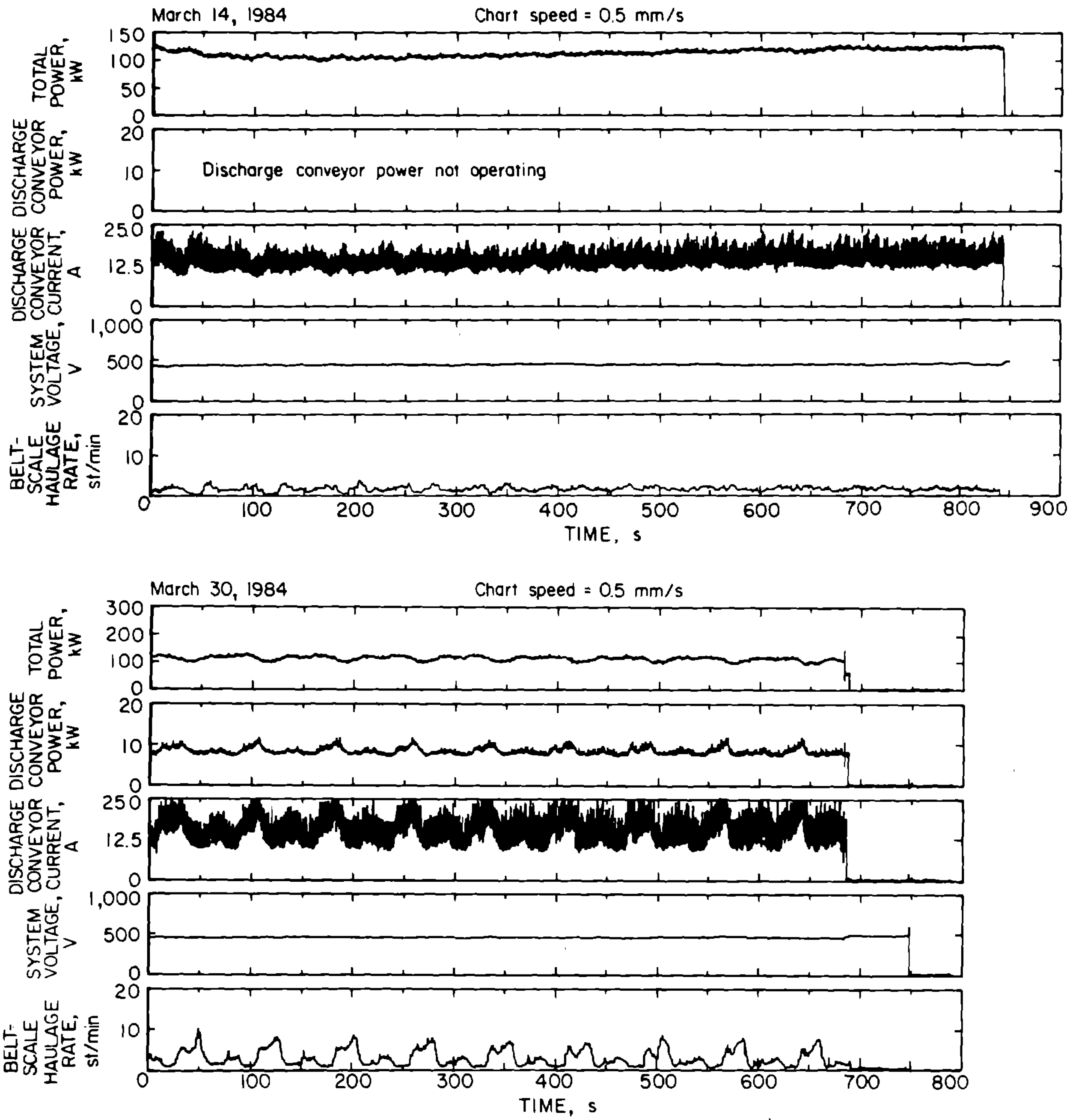

Figure 17.–Typical strip-chart data, 2.3 st/min (top) and 1 to 8 st/min (bottom).

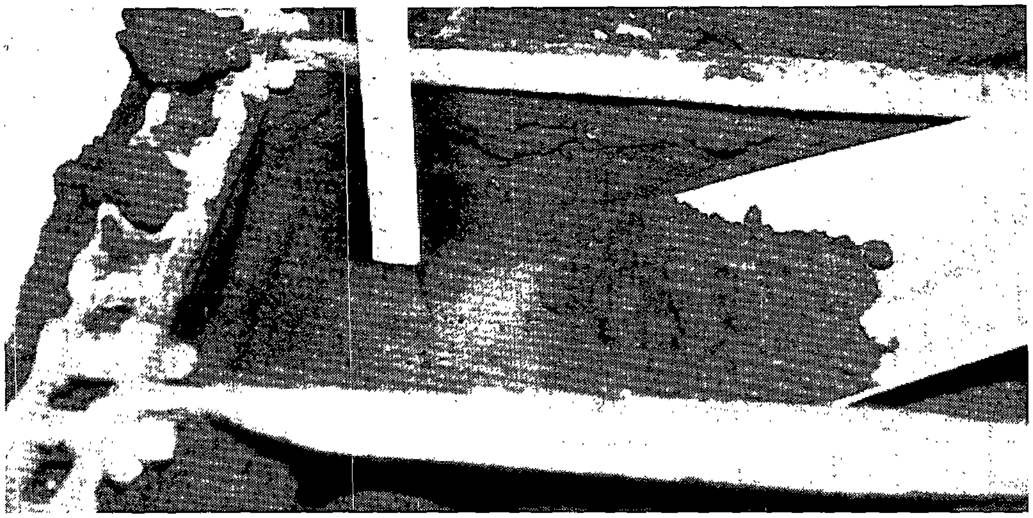

Figure 18.-Coal fines buildup on conveyor deck.

Throughout the test, the chain carried coal fines back underneath the conveyor decks. This material was ejected from the system at the point where the conveyor drive sprockets engaged the conveyor chain. It was only very fine material that piled up beneath the vehicles. It should be noted that no significant material loss was found around the hoppers at vehicle transfer points, even on those vehicles that were skewed at a steep angle.

After this testing was completed, the amount of coal lost from each vehicle was weighed. Typical coal carryback loss can be seen in figure 19. Table 8 shows the amount of coal lost under each vehicle and also quantifies that loss as a percentage of the total amount of coal conveyed. As a percent of the total 682.1 st conveyed, loss from individual vehicles ranged from 0.026 pct in intermediate vehicle 8 to 0.217 pct in intermediate vehicle 3. Average carryback loss was 0.063 pct per vehicle or 0.75 pct total. Vehicle 3 tended to lose more material than the others and was therefore manually cleaned out during the test. This in itself increased the amount of loss by allowing more room for material to drop from the conveyor. It should be noted that because material became finer as the test progressed, rate of carryback loss increased. In an underground mine, coal will only pass through the system once. The percentage of fines will be much lower; therefore, carryback losses will be less.

Table 8.-Conveyor carryback loss

Vehicle	Loss, lb	Portion of total conveyed,[1] pct
Lead	614	0.045
1	554	.041
2	747	.055
3	2,957	.217
4	755	.055
5	426	.031
6	475	.035
7	514	.038
8	355	.026
9	885	.065
10	1,241	.091
Discharge	740	.054
Total	10,263	.75

[1]Total conveyed, 682.1 st.

The conveyor gearbox temperatures of the discharge vehicle and intermediate vehicle 10 were monitored throughout the test program with Type K thermocouples. A temperature stabilization curve for gearbox lubricant is shown in figure 20. Data were obtained on March 28, 1984, at haulage rate of 0 st/min for a 135-min period. A stabilization curve at various loading rates could not be obtained because the system would not operate for a sufficient period of time to stabilize oil temperature.

The temperature at which the lubricant temperature stabilizes gives an indication of the load on the gearbox and its condition. Temperature in intermediate vehicle 10 stabilized at 131° F, with an ambient temperature of 56° F (or 77° F above ambient). This temperature is well within bounds of reasonable operating temperature.

The discharge vehicle gearbox temperature did not stabilize during this period of time, but the slope of the curve was decreasing rapidly toward the end of the run. Temperature at the end of the run was 102° F above ambient. Under these conditions, discharge conveyor motor current and power consumption were 12 A and 7.8 kW (10.5 hp), respectively, at 0-st/min haulage rate.

The results of the March 14 through April 12, 1984 conveyor tests follow:

- A total of 682.1 st of coal was conveyed during 560.6 min of operation.

- Coal fines, generated by recirculation of the coal, were getting wetter on each recirculation in the closed-loop system because of the water sprays. The wetted coal fines increased the operating loads significantly, which caused a high failure rate of conveyor drive components. Because of the large number of mechanical failures and electrical problems, no more than 136 min of operating time was achieved in any single day (28 pct availability).

o Conveying horsepower requirements ranged from 5 hp per vehicle at no load (0 st/min) to 18.4 hp per vehicle at 5 st/min (while conveying wet coal fines).

o Specific energy required to transport the coal through the entire 228.8-ft system was 0.41 kW·h/st.

o Almost no coal was lost through the system at vehicle transfer points. The only loss was through conveyor carryback, which represented about 0.75 pct of the total amount of coal conveyed.

o Conveyor chain speed was 270.9 ft/min at 4-st/min haulage rate. Transport speed of discrete particles was 234 ft/min at 4 st/min.

o Carryback on the conveyor return deck for each vehicle was between 0.5 and 1 in. Some material was consolidated and cemented to the deck, some was loose (fig. 21).

o Coal fines were tightly cemented into place between the chain guides and sideplates on the return deck of some vehicles. Material was so hard that the chain rode over the top of the coal.

o Very hard, consolidated material had built up in the pelican beak in front of the conveyor drive shaft (fig. 22). This area had been previously cleaned using a slate bar through the cleanout ports. This method of cleaning the pelican beak is ineffective, as the material becomes hard packed and will not flow out the cleanout ports. Even when loose, material will assume a natural angle of repose, building up to the edge of the cleanout ports before any coal exits the area. This material eventually is carried into the hopper.

o The amount of coal in the hoppers varied from vehicle to vehicle. Hard-packed coal fines were found throughout the hopper, especially between the chain guides and sides of the hopper, of intermediate vehicles 3 and 4. It was evident that this probably created a bind in the system, increasing the operating loads.

o There were not a great deal of consolidated coal fines in intermediate vehicle 8.

o It was found in intermediate vehicle 3 that the chain guides were spaced too close together and the chain links, rather than the chain rollers, were riding on the chain guides. This also increased operating loads.

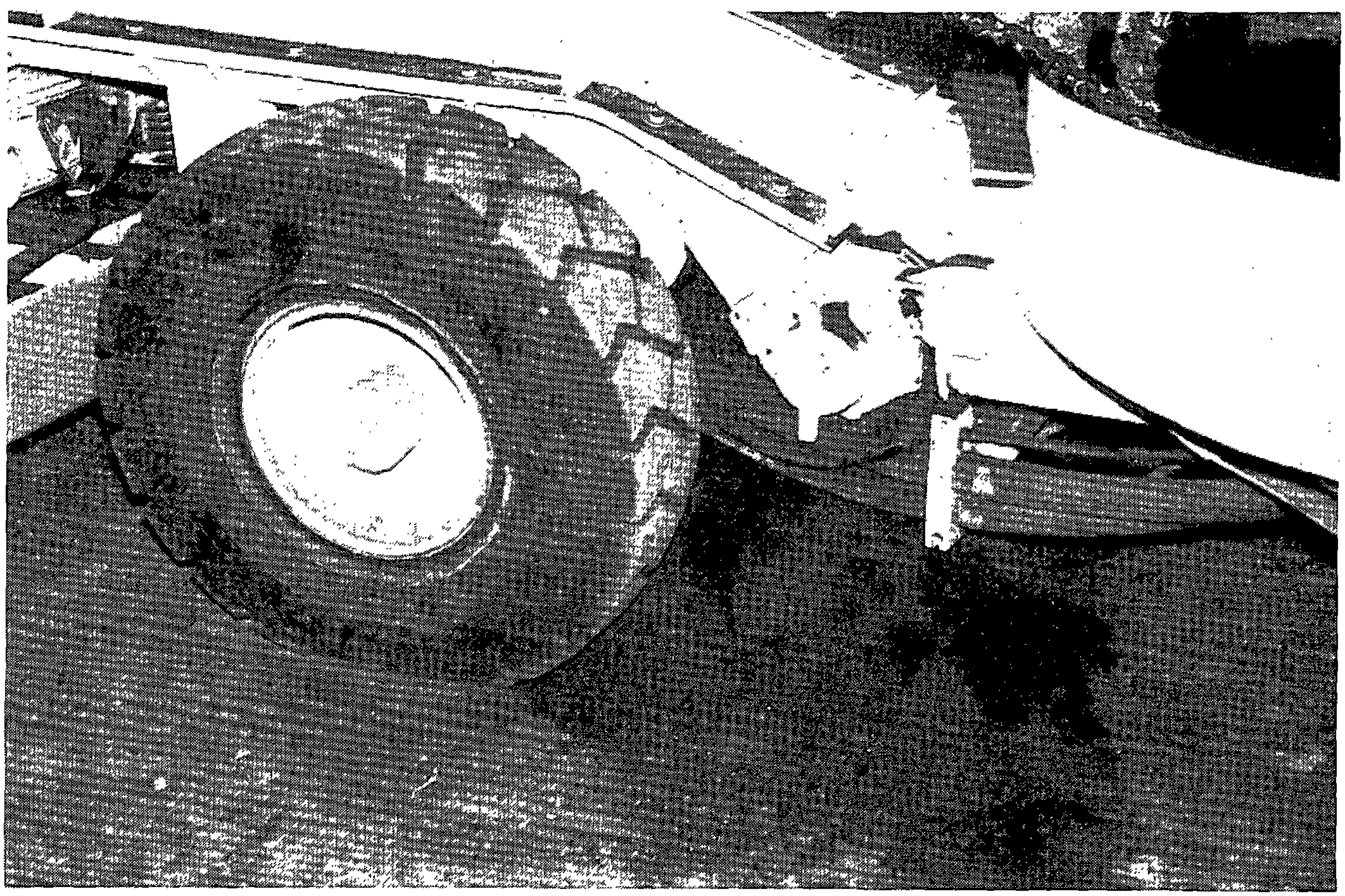

Figure 19.–Typical coal carryback.

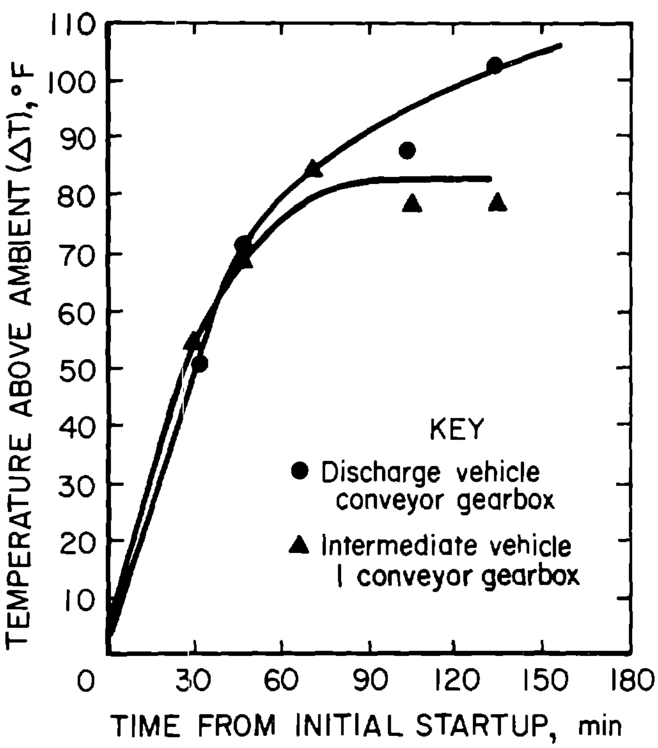

Figure 20.–Conveyor drive gearbox temperature rise.

July 31 to August 3, 1984

Objective

During previous haulage testing, it was found that one of the primary problems encountered was that of coal carryback. The coal carried back into the return deck increased power consumption and caused the system to periodically plug and shear drive pins, or cause other component failures.

In an attempt to diminish the problems caused by carryback, it was decided to cut cleanout holes in the return deck to allow the coal carryback to exit, preventing its buildup. Therefore, cleanout ports were cut in the following areas of intermediate vehicles 3, 4, and 8 and the discharge vehicle:

1. *Hopper.*–Coal that is carried back into this area can cause the conveyor to stall, therefore two 4- by 8-in holes were flame cut just behind the foot shaft.

2. *Pelican beak.*–Heavy buildup always occurs in this area. Even when the cleanout ports are removed, material will not clean itself from this area. Two 7.5- by 8-in cleanout ports were cut in the bottom of the pelican beak (fig. 23).

3. *Return deck.*–Material builds up on this deck, increasing chain flight friction, therefore, the built-in outby cleanout port covers and middle cleanout ports were removed. The expectation was that the material that would otherwise be carried back would drop out onto the next outby vehicle hopper.

4. *Area between chain guides and sideboard.*–Slots, approximately 0.75 by 6 in, were burned in the return deck between the chain guides and sideboards to allow material carried back by the chain links to exit. This area has a tendency to become plugged to a greater extent than the return deck.

Coal buildup on the top decks was another problem that was addressed. Chain holddowns were fabricated from 2- by 2- by 0.25-in, 10-ft-long angle iron and bolted to the hopper sideboards of intermediate vehicles 3, 4, and 8, and the discharge vehicle.

The chain guides on the return deck of intermediate vehicle 3 were removed, allowing the chain flights to contact the return deck. It was expected that this would help purge the return deck of coal.

Procedure

The MUCH system was trammed into a circle with the discharge vehicle dumping onto the lead vehicle, the bridge conveyor was not utilized. The belt scale was unavailable, as it was assigned to another project. Otherwise, instrumentation that was used in April haulage test was again utilized. Total conveyor system power, discharge vehicle conveyor power, discharge vehicle conveyor current, and system voltage were measured on the Gould model 481 strip-chart recorder.

The conveyor system was loaded with a fine mix of coal of an unknown size consistency from the Bureau's Hydraulic Transport Research Facility (HTRF) stockpile. Haulage rate was maintained at an estimated 3 to 6 st/min.

Results and Discussion

The system was operated for several hours during July 31 to August 3, 1984, at haulage rates between 0 and 6 st/min (estimated). Because of the variability of the data and lack of haulage rate data, it was difficult to draw any definitive conclusions, but the system suffered no mechanical component failures (except shearpins) during this test period.

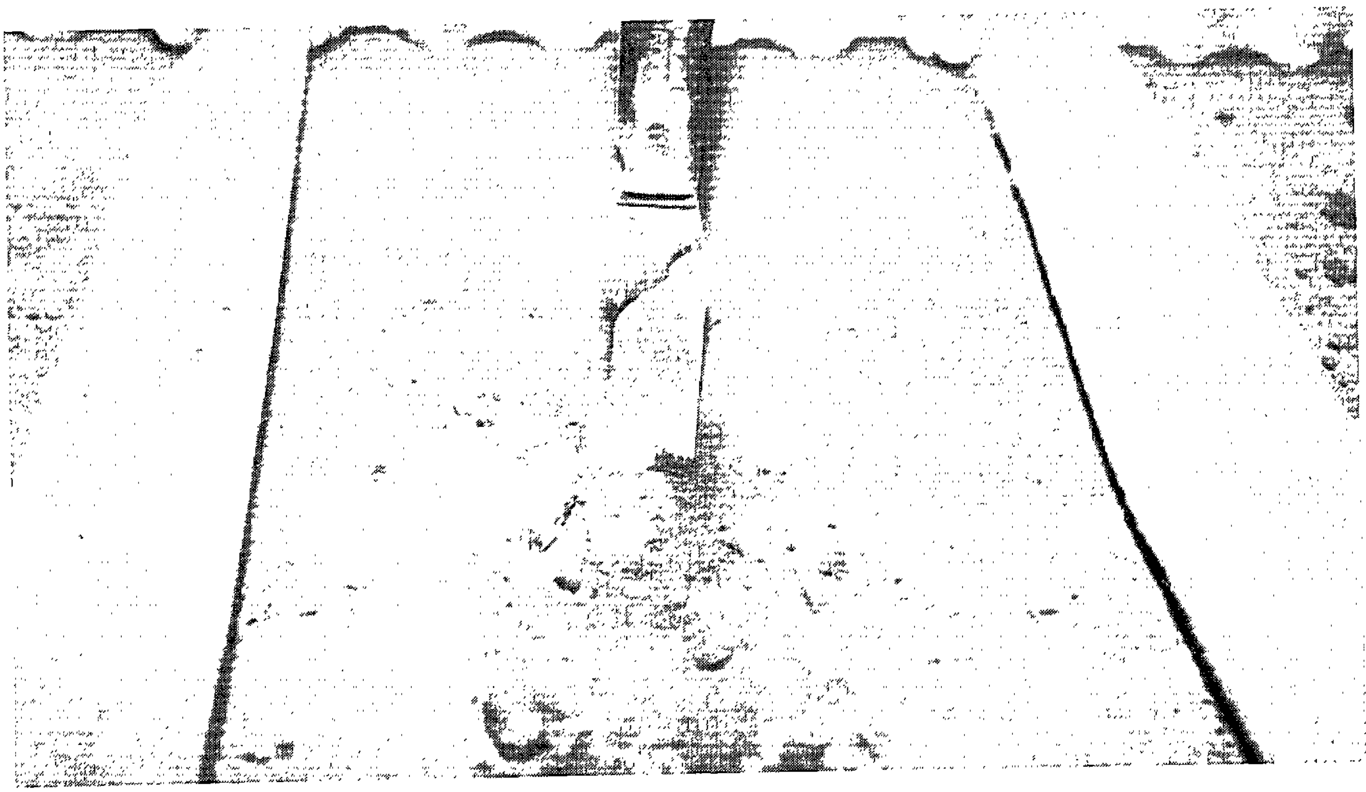

Figure 21.-Conveyor return coal buildup.

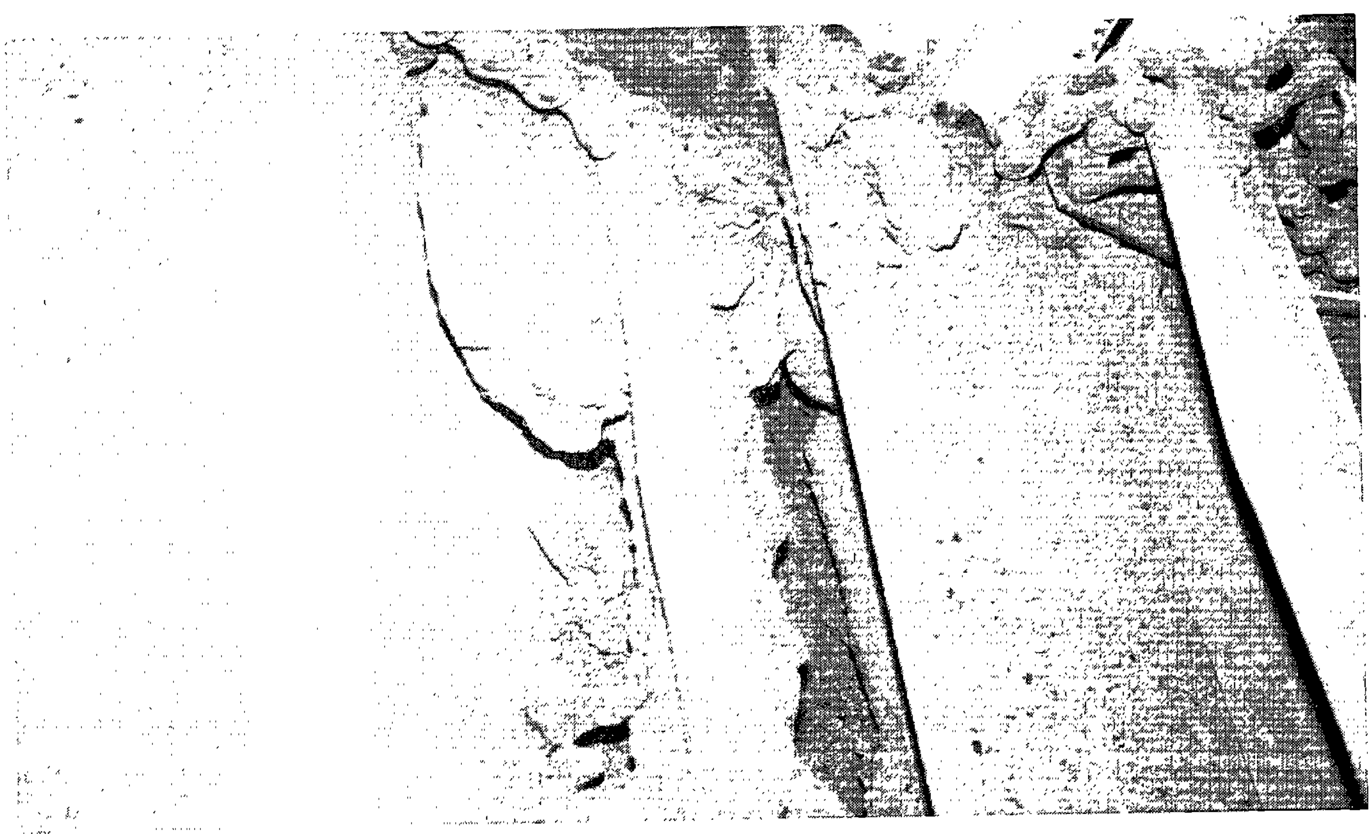

Figure 22.-Coal fines buildup in pelican beak.

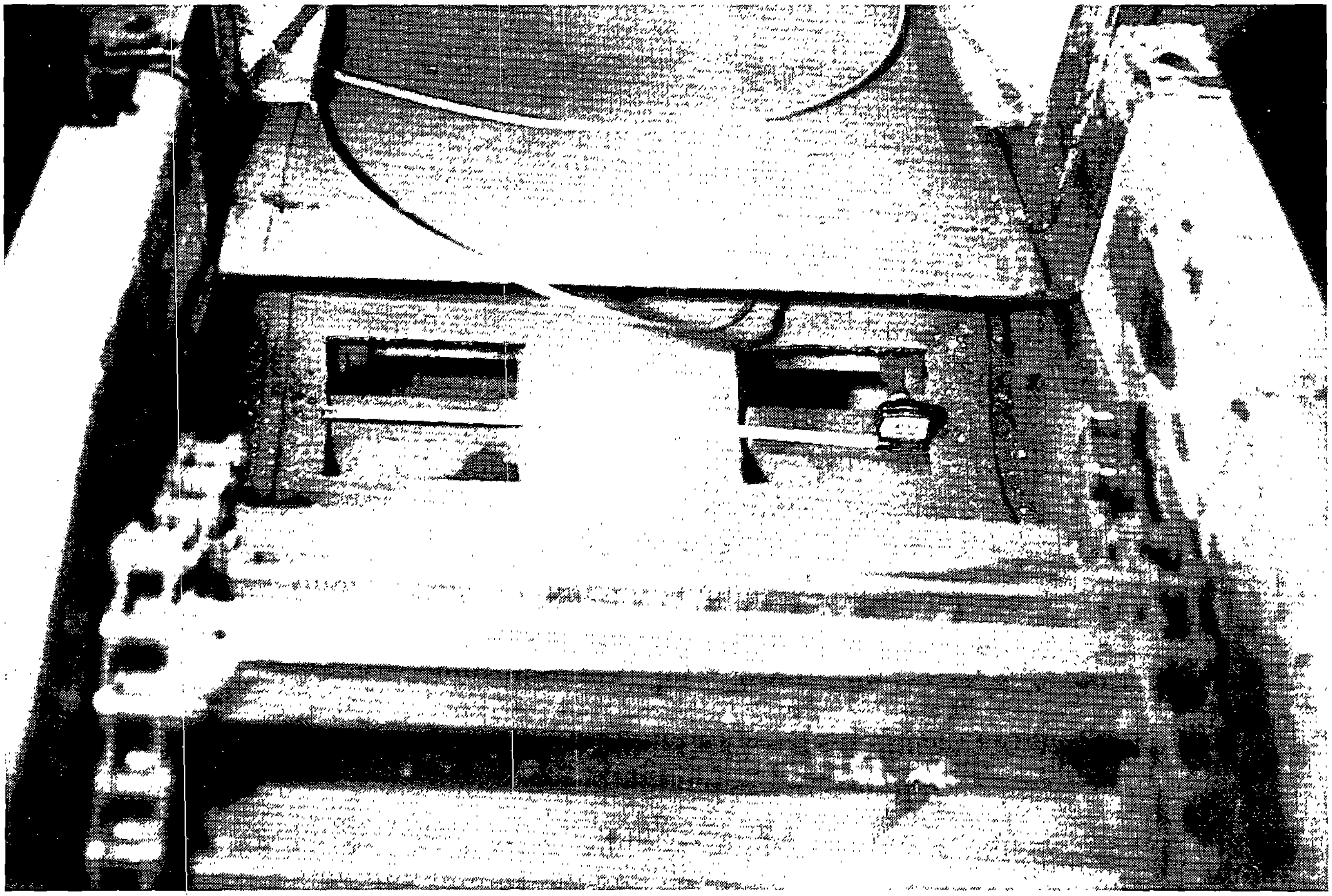

Figure 23.–Cleanout ports in pelican beak.

Generally, the steady-state power consumption was 85 to 95 kW, discharge vehicle current was 12 to 13 A, and power was 8 to 8.5 kW at an estimated (rough) 3 st/min. Comparing these figures to the regression equations developed for discharge vehicle current (I_d, = 16 A) and total system power (P_t, 123 kW) during the April 1984 haulage testing, it appears that the cleanout ports were effective to some degree judging by the reduced motor current and power consumption. Upon teardown of the upper decks, it was observed that there was much less coal left on the return decks and in the hopper return area than in the previous haulage testing in March. The regression used were

$$\text{Discharge vehicle current } (I_d) = 12.17 + 1.26\,\dot{Q}, \text{ and}$$

$$\text{Total system power } (P_t) = 62.8 + 20.2\,\dot{Q},$$

where $\dot{Q}$ = haulage rate, st/min.

The problem with these cleanouts was that an excessive amount of material was lost through the cleanout holes (fig. 24). Appendix C shows the carryback loss for each vehicle. Carryback loss averaged 264 lb for the unmodified (no cleanout ports) vehicles. The modified (cleanout ports added) vehicles averaged a 2,255-lb loss, or an unacceptable 8.5 times greater carryback loss; therefore, the cleanout ports were covered.

Chain holddowns were effective in preventing coal buildup on the top decks. None of the modified vehicles had any appreciable buildup on the top decks. The configuration of the leading edge of the holddowns on the discharge vehicle had to be modified. The ramped shape of the leading edge caused coal to wedge between the holddown and chain, pinching the chain, increasing sliding friction, and stalling the conveyor. Cutting off this ramped transition area resolved the problem, returning the motor current to its normal level. Removal of the chain guides on the return deck of intermediate vehicle 3 was very effective in cleaning the deck.

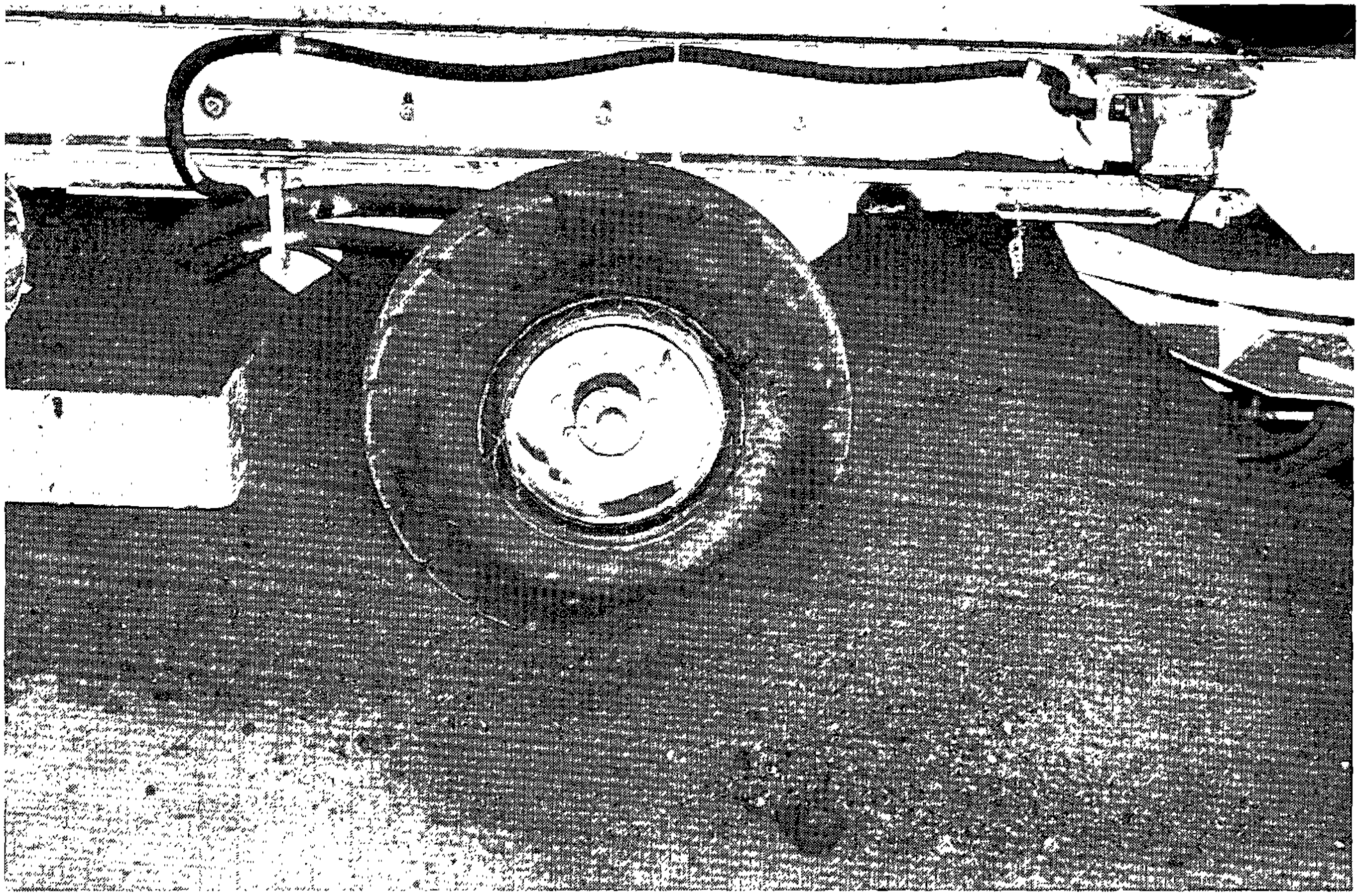

Figure 24.–Coal carryback loss after haulage testing, July 31 to August 3, 1984.

September 10-11, 1984

Objective

Water sprays were fabricated to test the concept of spraying water in the return deck to minimize the problem of coal carryback plugging the system. Four spray bars or manifolds were fabricated (fig. 25) using eight Whirl jet 3/8 BD 8 size 1 nozzles manufactured by Spraying Systems Co. Two nozzles were used per spray bar, 16 in apart. Two spray bars were installed in intermediate vehicle 8 and two in the discharge vehicle. One spray manifold in each vehicle was inserted in the pelican beak through the cleanout ports. Nozzles were faced up and outby approximately a 45° angle (fig. 26). The middle cleanout port in the return deck of both vehicles was removed and the manifolds were fastened into place with the nozzles facing up and outby. Cleanout ports used during the July 31 through August 3 haulage test were covered.

Procedure

The MUCH system was placed in a circle and instrumented as before in July and August conveying tests. Testing was conducted as before, adding coal as needed to achieve an estimated haulage rate of between 3 and 6 st/min.

Results and Discussion

Water pressure at the machine inlet was 135 psi when the eight water sprays were functioning. Total flow rate measured was 2.69 gal/min or 0.336 gal/min per nozzle.

The test did not run long before the coal was extremely wet, to the point of being a slurry. The material leaked out of the conveyors because of its liquid nature and the floor became extremely wet. This amount of water lost would present a serious problem underground, therefore the water spray manifolds were removed.

Umbrella Miner Cutting Trials Support

Objective

To gain more information on the haulage characteristics of the MUCH system conveyors, it was decided to utilize the system to remove coalcrete cuttings that were produced during umbrella miner cutting trials in the Bureau's Miner-Bolter Test Structure (MBTS).

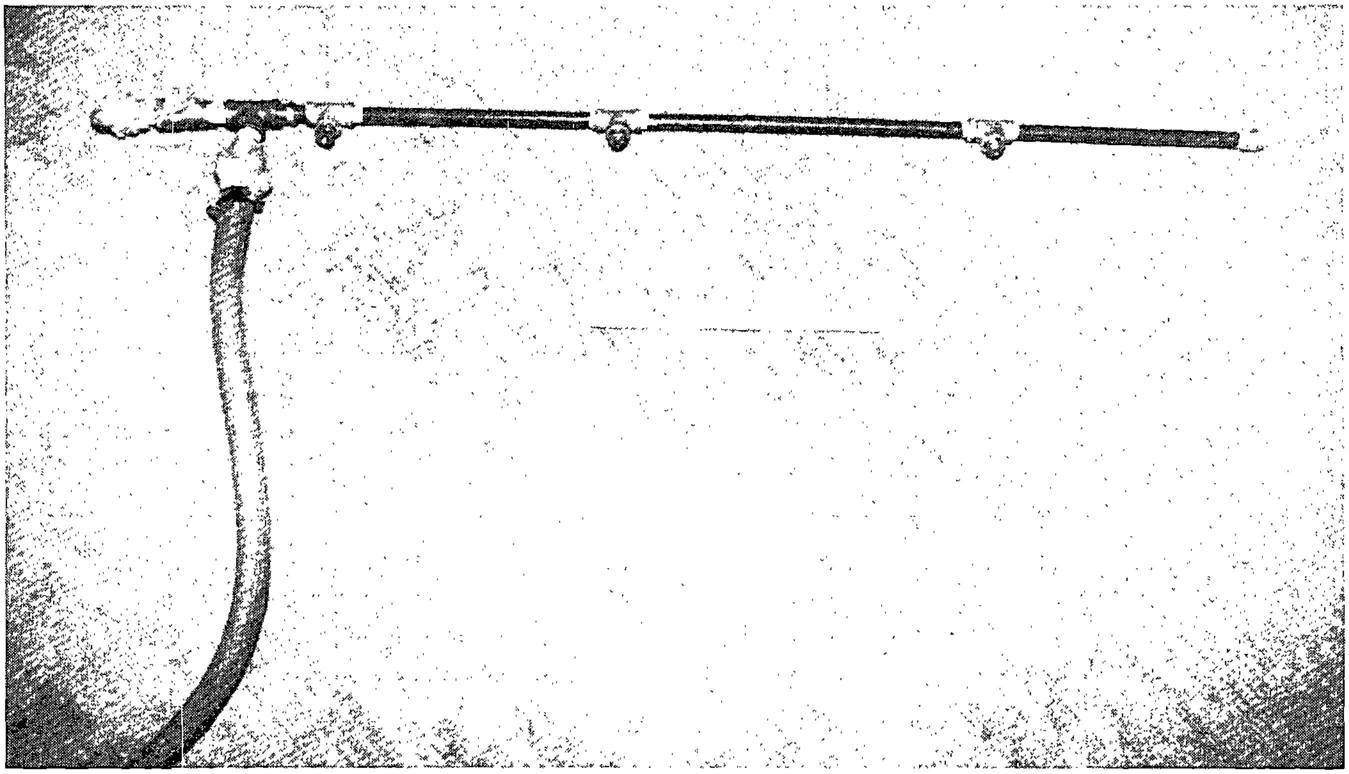

Figure 25.-Water spray nozzles.

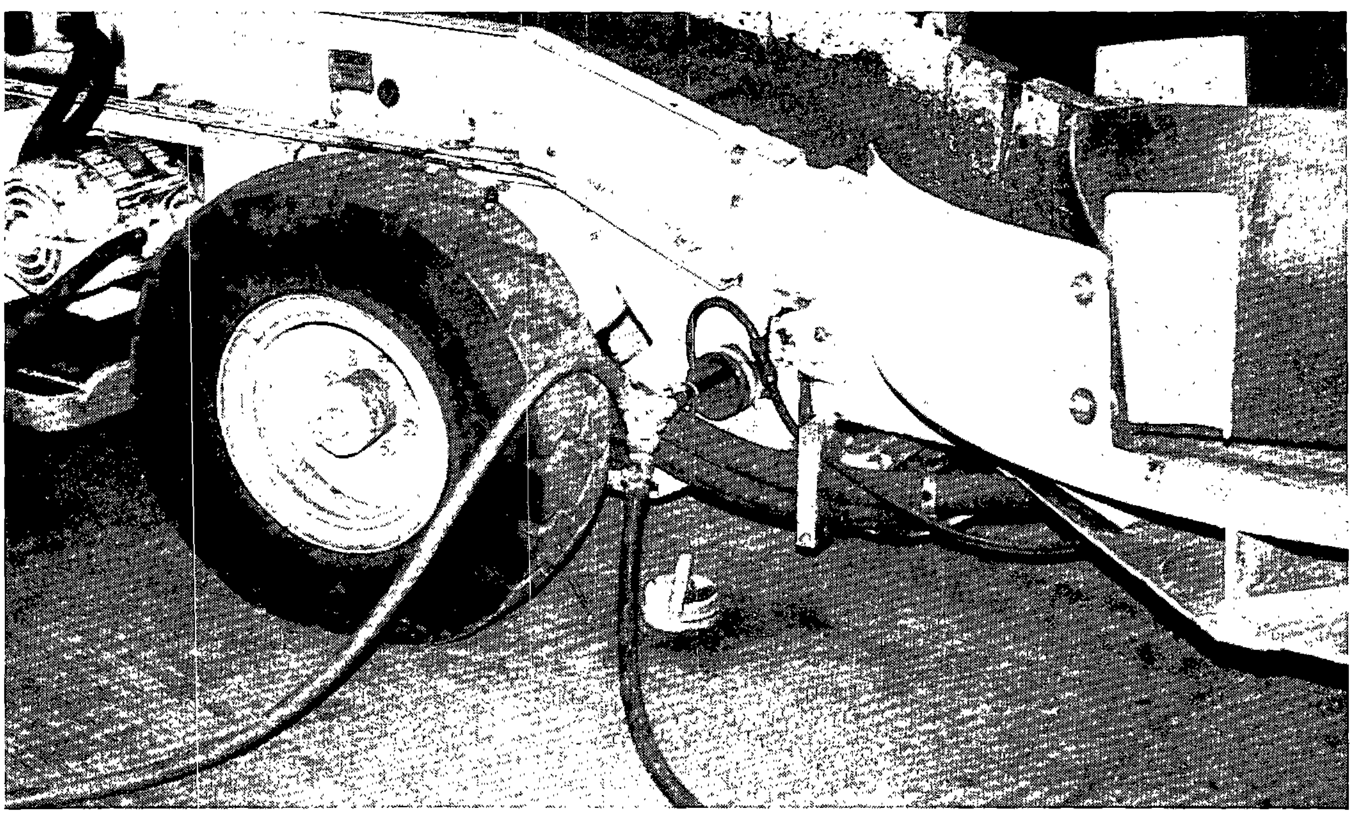

Figure 26.-Water spray manifold in pelican beak.

Procedure

The MUCH system was positioned to receive cuttings from the umbrella miner in the MBTS, to convey the material through the simulated workings, and then to discharge the cuttings from the bridge conveyor into the bucket of a front-end loader at the opposite end of the building. The system was instrumented to monitor the electrical system voltage, the total system electrical power in kilowatts, the discharge vehicle conveyor power in kilowatts, and the total system current requirements in amperes. Before testing, the conveyor system on each vehicle was run to verify proper operation and to remove any old coal fines from the conveyors. Each conveyor chain was oiled to free up any "frozen" links, and a new conveyor drive clutch was installed in the discharge vehicle. During this conveyor trial, coalcrete cuttings would be conveyed only once over the system, as opposed to the previous closed-loop type of haulage tests, which tended to overload the conveyors with a large quantity of fines from material degradation over a period of time. This trial would closely simulate actual coal haulage in an underground production situation.

Results and Discussion

Although the MUCH conveyor system performed almost flawlessly during the trial and the only downtime was 10 min because of one conveyor drive shearpin failure, the trial did not produce much useful information. Mechanical difficulties with the umbrella miner while trying to cut the coalcrete were the main problems. An insufficient amount of coalcrete material was loaded to tax the MUCH conveyor system much above its power requirements while running empty.

The maximum power requirement observed during the trial, except for startup power, was 25 kW (33.5 hp) for the total system power. When the system was running empty with no load, 18.5 kW (25 hp) was required. The maximum observed loading was only a 35-pct increase over the no-load conditions. The maximum load required to start the system was 40 kW (54 hp).

During the previous closed-loop conveyor trials conducted in 1984, no-load (0 st/min) power consumption data were taken for the system; these did not include the bridge conveyor. Because of previous coal haulage trials, a large amount of coal fines was in the return deck and pelican beak areas of the vehicles, as well as wet fines that had accumulated on the conveyor decks. No-load power consumption data during those trials ranged from 45 kW to 68 kW (60 to 90 hp), which is approximately three times higher than what was observed in this test. Obviously, the accumulation of coal fines and conveyor chain lubrication have a significant influence on the amount of power required to drive the conveyor system and the subsequent loads.

Conveyor System Problem Summary

Upon reviewing the results of the four tests, it became obvious that there was a major problem with the conveyor system ability to handle fines. The coal fines, built up on the conveyor top deck and on the return deck, ultimately overloaded the system to the point of failure. The addition of water only made the problem worse. Cleanout ports in the return deck helped reduce fines buildup but lead to excessive loss of material from the system by dumping fines on the bottom. The closed-loop test setup was also contributing to the fines handling problems. Conveying coal in a closed loop continuously degrades the coal and yields all fines, which is not reflective of a normal ROM product that the system would be handling at an underground production face. Based on these observations a number of changes were made to the system as follows.

- A conveyor chain slack adjustment mechanism was added to each vehicle to provide a simple means of keeping the conveyor chains tight so as to limit fines buildup on the decks. This modification is described in the "System Modifications" section.

- Autoguard torque clutches were removed from the conveyor drive systems and replaced by Lovejoy couplings.

- The conveyor drive shearpins were removed and the shearpin couplings were welded solid. Conveyor drive motor thermal overloads were resized to adequately protect the drive components.

- The conveyor chain holddowns were removed, redesigned, and reinstalled.

Approximately 12 st of 3- by 1-in gravel was purchased for further conveyor system evaluation after the completion of these modifications. Approximately 5 st of the 3- by 1-in gravel was dumped onto the system. The system was started and ran a very short period of time before it shut down on electrical overload trippings because a conveyor jammed.

The jam was caused when the conveyor chain rode over a rock (fig. 27) and the rock was caught between the tail sprocket shaft and the chain flight (fig. 28). The primary cause was the large gap between the end of the conveyor deck and the tail sprocket shaft as shown in figure 29. This gap permitted material to drop between the end of the conveyor deck and the tail sprocket shaft. Material that was too large to fall through would either be broken by the next flight and exit the gap, stay in the gap momentarily and lift the conveyor chain as it passed over (which allowed material to get caught under other chain flights), or it would become jammed between the chain flight and the tail sprocket shaft. Fines that were small enough to fall through the gap would mostly fall into the hopper of the next vehicle but a fraction would be carried by the return chain flights into the return deck to cause further jamming problems.

The solution to these jamming problems was to (1) design and install a plate on the conveyor deck to bridge the gap, and (2) redesign the conveyor chain holddowns on each vehicle. These modifications are fully described in the "System Modifications" section of this report and are shown in figures 30 and 31.

Coal Conveyor Acceptance Test–October 29, 1986

Upon the completion of the conveyor modifications on all vehicles, except the bridge conveyor, the system was run empty for a couple of hours to check operation and to break the system in before attempting to load the conveyors. After the initial break-in of the conveyors, the system was moved outside to a level open area and was trammed into a closed loop for testing. The bridge conveyor was included in the loop. Approximately 8 st of 1- by 3-in gravel was loaded onto the conveyors and the conveyors were run in a closed loop for 30 min with no problems occurring. The rock was then conveyed off the system and 6 st of ROM coal was loaded onto the system for the next day's acceptance test.

Objective

The goal of the conveyor acceptance test was to continuously convey coal at a rate of 8 st/min over an 8-h period while maintaining a system availability of 95 pct. For an 8-h test (480 min) this allowed only 24 min of downtime.

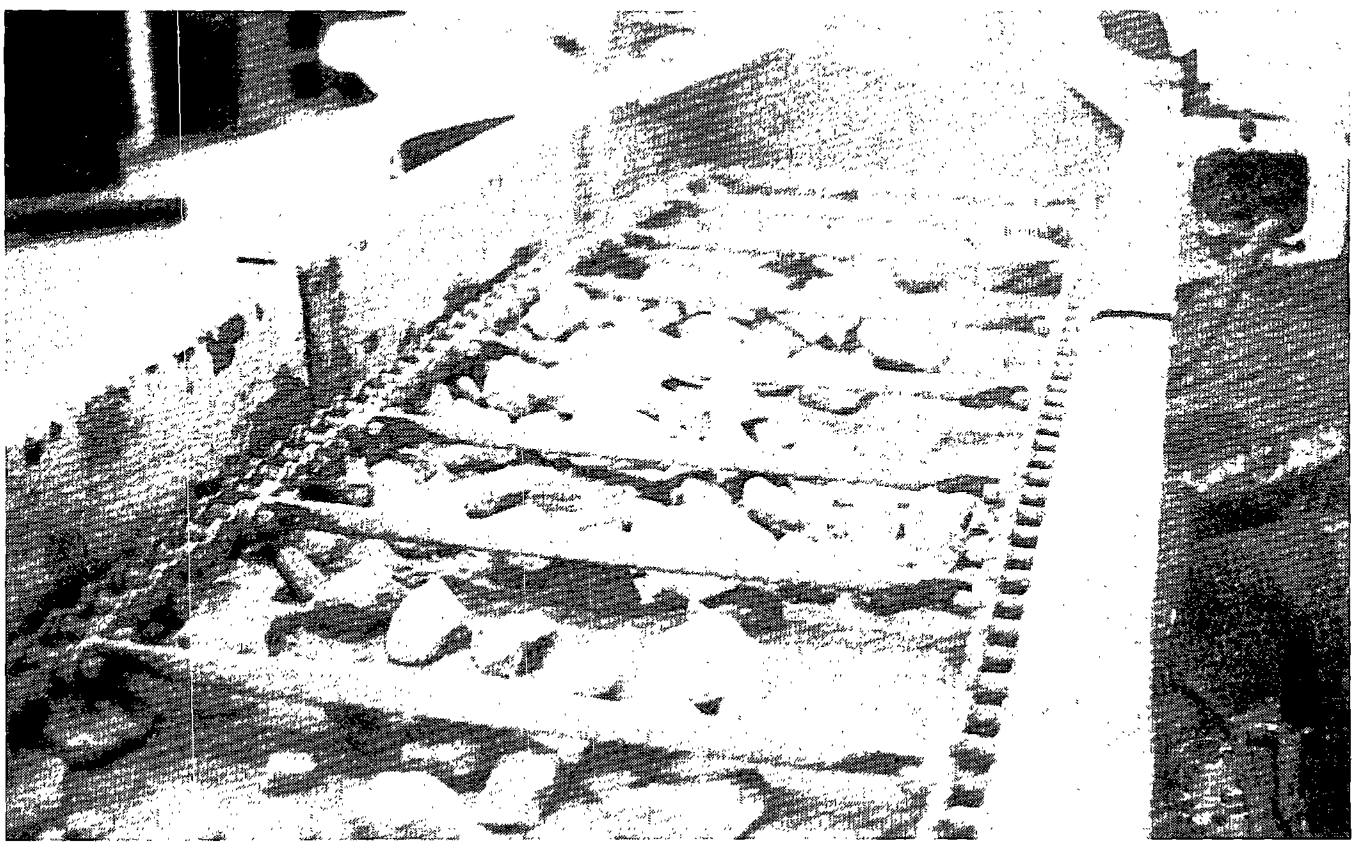

Figure 27.–Conveyor chain riding over rock.

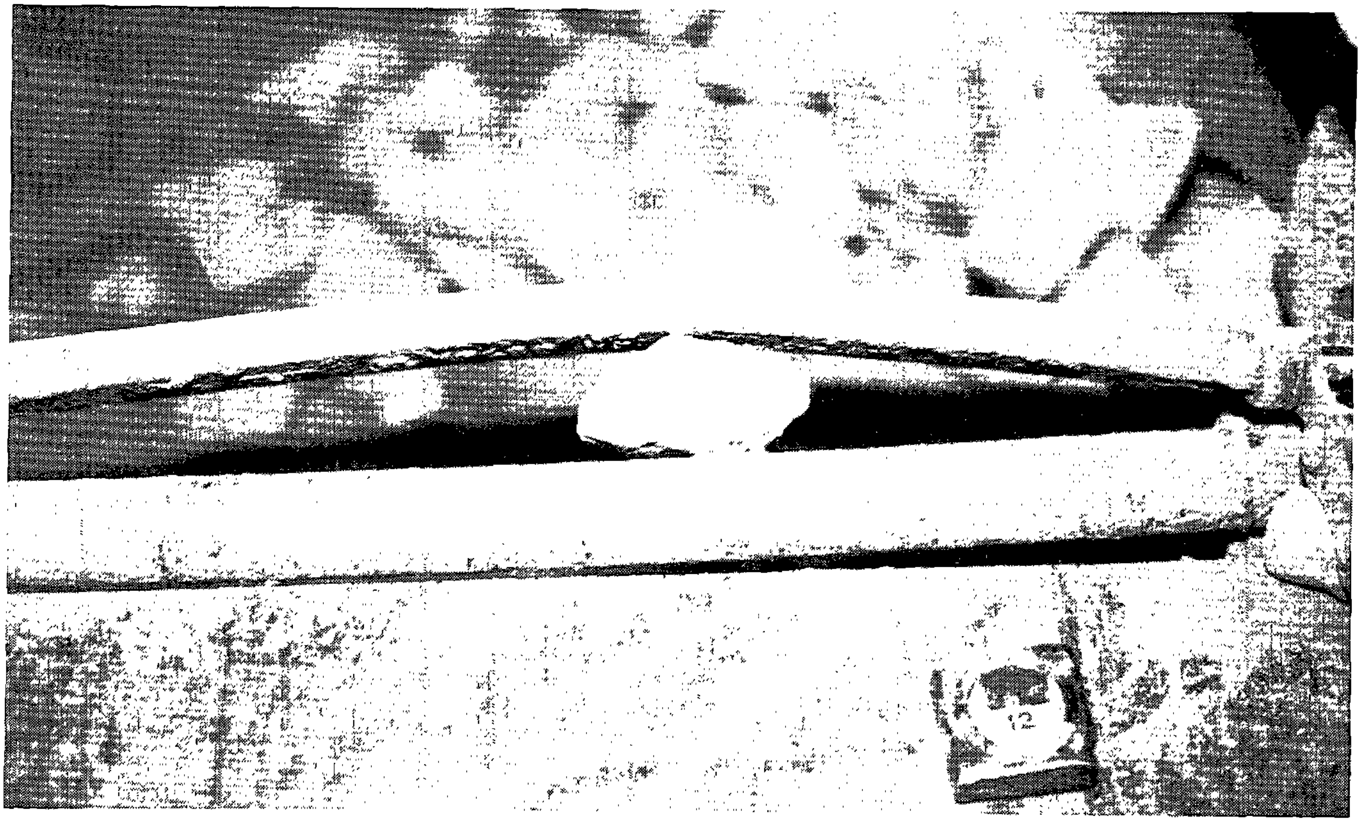

Figure 28.-Rock caught between shaft and flight.

Figure 29.-Gap between deck and shaft.

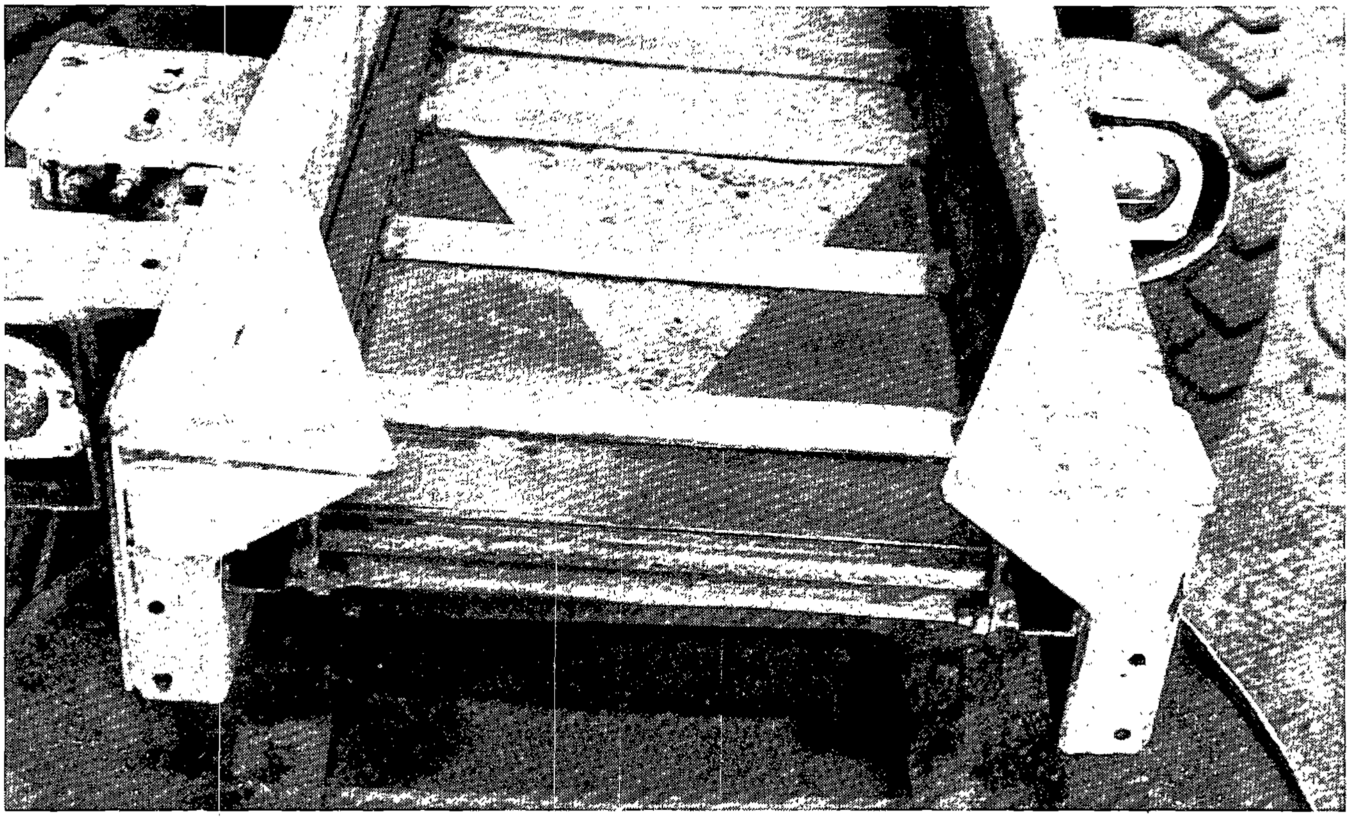

Figure 30.–Conveyor deck plate.

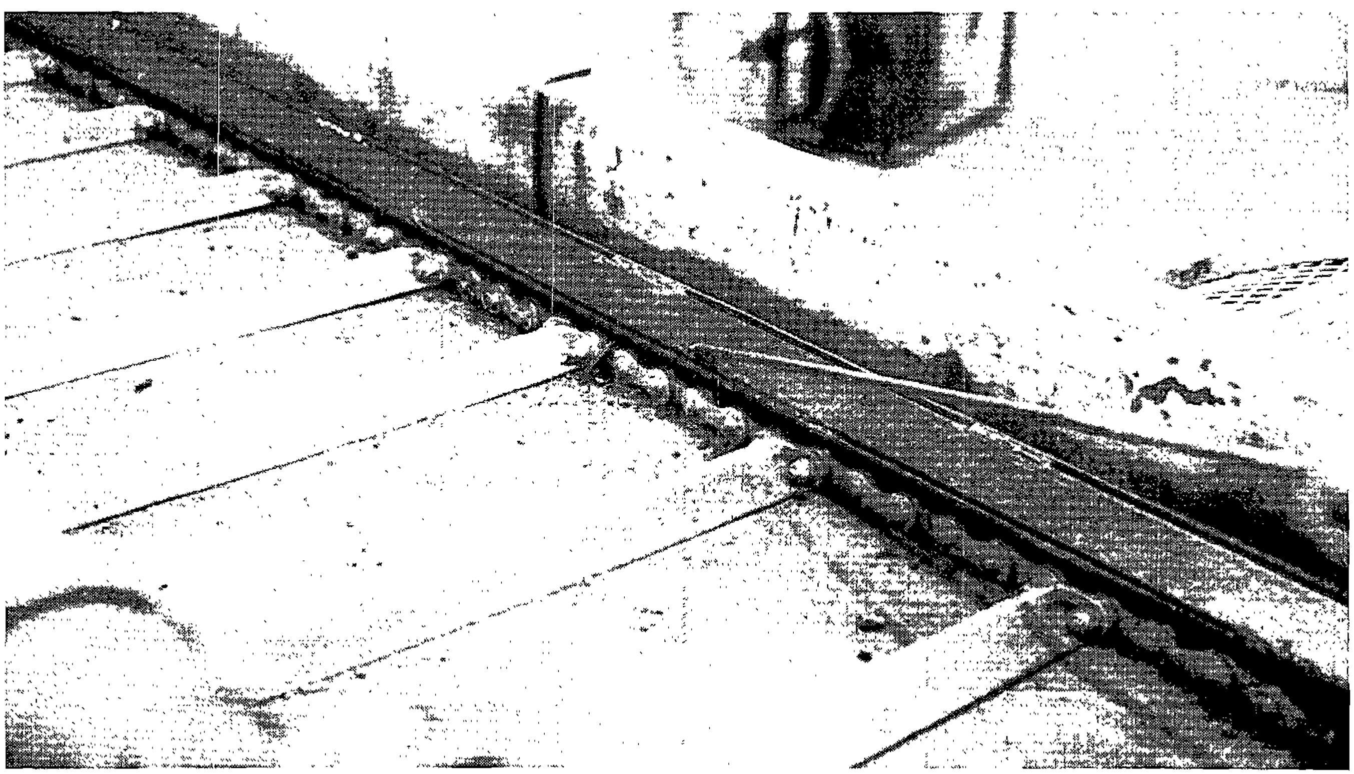

Figure 31.–Conveyor chain holddowns.

Procedure

The test was conducted on October 12, 1986. The log of the test event is given in appendix D. The test was initiated with 6 st of ROM coal on the system. During the first 30 min of the test an additional 6 st of coal was added to make up for the loss of fines as the coal degraded and to maintain 8 to 10 st of material on the system. A belt scale was not utilized during the test to avoid unnecessary downtime the additional belt might cause.

Approximately 1 h into the test an additional 2 st of coal was added to the system. At this point in the test a total of 14 st of ROM coal had been utilized, approximately 12 st was on the system and about 2 st had been lost as fines. As the test continued the coal kept degrading and fines were being lost, so after about 2 h of running time, 3 st of the 1- by 3-in gravel was added and mixed with the coal.

Because the test was being conducted outside, and to avoid problems previously encountered because of excess water, very little water was added to the material during the test. The water sprays were used only when the dust became excessive. As the test continued, the conveyor drive motors thermal overloads kicked out because of excessive loads, especially on the lead vehicle. The haulage rate was estimated between 12 to 13 st/min of mixed coal and rock. Approximately 2 st of material was removed from the system and the test continued. The test was terminated at 2:24 p.m. with over 8 st of material remaining on the system.

Results and Discussion

The test time was from 8:00 a.m. to 2:45 p.m., a total of 405 min. The total haulage time was 337 min and the total downtime was 68 min. The system availability for the test was 83.2 pct. The majority of the downtime, 58 min or 85 pct, was due to thermal overload trips on the conveyor drive motors on the lead, intermediate vehicle 2, and intermediate vehicle 5. After the test the conveyor drive motor thermal overload setting on each vehicle was checked. On the three problem vehicles the settings were found to be lower than on the other vehicles, which contributed to the excessive tripping problem.

The total amount of coal-rock mixture conveyed is estimated by 337 min of haulage time at an approximate rate of 10 st/min or 3,370 st. A total of 17 st of material was loaded on the system of which 2 st was removed and 8 st remained on the system at the end of the test. Therefore, 7 st of material was lost during testing, mostly as fines, with some of the loss due to spillage at dumping points between vehicles. The total amount of lost material, 7 st, is equivalent to 0.2 pct of the total 3,370 st conveyed.

DRAWBAR PULL TEST

Objective

A drawbar pull test was conducted to measure the tractive effort produced by the MUCH system. The amount of tractive effort available gives an indication of the capability of the system to tram up grades and, if needed, to tow or pull a disabled or buried piece of equipment. It should be noted that the MUCH system is not intended or designed to be a towing vehicle and the use as such could lead to system damage.

Procedure

The drawbar pull of the MUCH system (without bridge conveyor) was measured by pulling against a 50-st-capacity Dillon dynamometer, which was anchored to a 35-st mobile crane (fig. 32). The dynamometer was attached to the discharge vehicle at the conveyor bridge mounting holes in the frame using a heavy chain. The dynamometer has a resolution of ±500 lb and a factory-stated accuracy of 0.5 pct of full scale. The brakes of the crane were applied during the pull test to keep the crane from moving. The test was conducted outdoors on a heavily compacted, level, dry, dirt surface. The system was trammed and positioned into as straight a line as possible prior to being connected by chains to the dynamometer and crane.

Results and Discussion

The first pull test was conducted with no material in the conveyors. The empty weight of the MUCH system is approximately 117,800 lb. The tractive effort measured by the dynamometer was between 27,000 and 30,000 lb of steady pull. It was observed during this trial that some of the tires were spinning against the ground and on some of the intermediate vehicles the wheel rims were spinning inside the tires. This may be one disadvantage of having foam-filled tires as opposed to air-filled tires, the adhesion of the foam-filled tire to the rim is less in some cases than the friction between the tire and the ground, which can reduce tractive effort.

The second pull test (fig. 33) was conducted after approximately 10 st of material was loaded onto the MUCH system, which increased the total weight to approximately 137,800 lb. The steady drawbar pull increased to about 35,000 lb. After a few moments of tractive effort at this level three of the tram motor overloads tripped, shutting tram motors down.

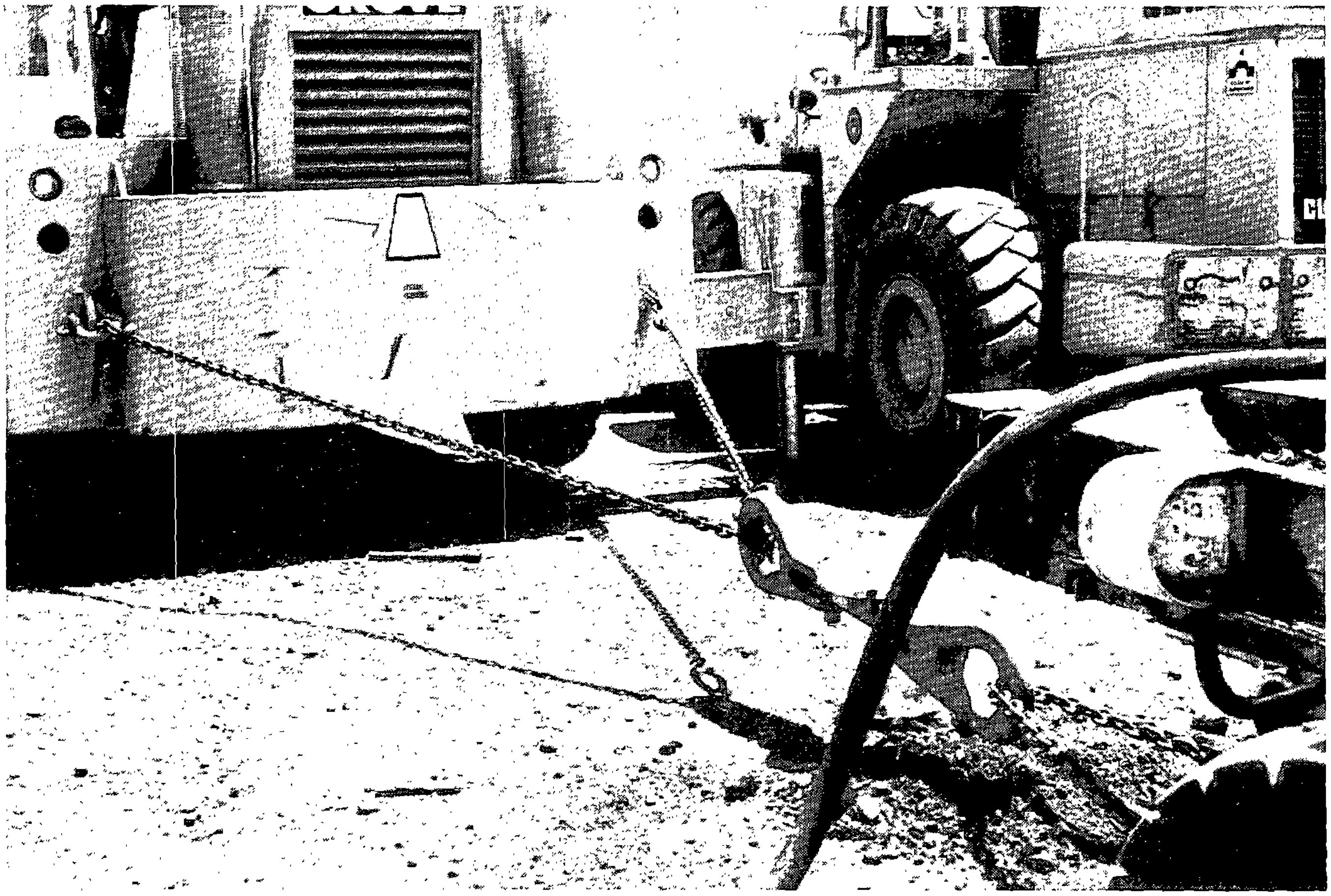

Figure 32.–Crane and dynamometer used during drawbar pull test.

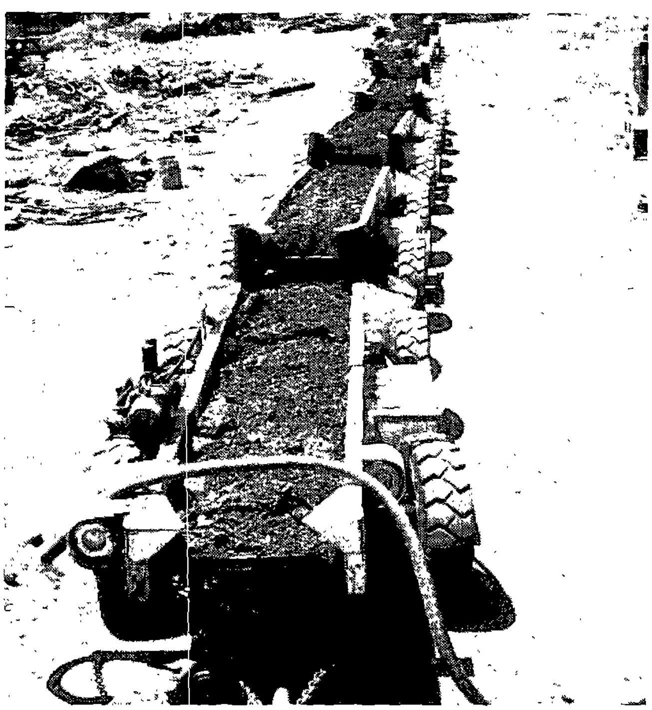

Figure 33.–Drawbar pull test.

ROUGH TERRAIN TEST

Objective

A rough terrain test was conducted to evaluate the performance of the system while tramming over areas with difficult bottom conditions.

Procedure

The test was conducted outdoors in an area with a surface of mixed mud and coal that had been saturated with water. A 2-ft-high by 13-ft-wide by 13-ft-long pile of mixed dirt, coal, rock, and 4-in by 4-in by 4-ft wood crib blocks was constructed as an obstacle. A roughly 10-ft-diam, 6-in deep water-mud hole was also part of the test course. The MUCH system was trammed repeatedly over the test course.

Results and Discussion

The system trammed over and through the pile of material and water-mud hole with no problems and no vehicle-to-vehicle binding or interference was observed. The front hopper on the lead vehicle acted as a plow to help remove high obstacles prior to driving over them with the system. The wet, muddy, slippery bottom conditions had no adverse effect on the tracking ability of the system. As tramming over the test course continued, 6- to 8-in-deep ruts developed in one area that were of such length that two vehicles had their drawbars and undercarriages dragging through the mud. There was no noticeable loss of tram speed or tracking ability in this area.

CANOPY LOAD TEST

The operator canopy, which protects the lead vehicle operator, was redesigned and load tested. The canopy redesign and installation is described in the "System Modifications" section of this report.

As stated in the Code of Federal Regulations (30 CFR 75.1710-1), a cab or canopy must have a "minimum structural capacity to support elastically:

(1) A dead weight load of 18,000 pounds, or

(2) 15 p.s.i. distributed uniformly over the plan view area of the structure, whichever is lesser."

The elastic load criterion required by Federal law is based upon a combined statistical, analytical, and experimental investigation where it was found that a cab or canopy designed to meet this elastic load criterion has at least enough potential energy (in the form of available strain energy) to withstand the majority of roof falls as determined by the statistical analysis of all fatal roof falls from 1966 to 1972.

The new MUCH system canopy was tested following MSHA guidelines.[7] The following items were used during the canopy load test:

1. A Sensotec, Inc., 20,000-lb-capacity compression force transducer.

2. A Fluke digital multimeter.

3. A Honeywell Accudata 218 bridge amplifier.

4. A 20-st-capacity hydraulic jack.

5. A displacement dial indicator accurate to 0.001 in.

6. A 12-in I-beam, 24 in long.

7. The roof cart of the miner-bolter test structure (MBTS).

8. Wood cribbing.

[7]Sawyer, S. G., and D. K. Brogam. A Testing Procedure for the Certification of Underground Protective Cabs and Canopies. MSHA IR 1002, 1974, 15 pp.

Procedure

In general, the test procedures, as outlined in MSHA IR 1002, were followed. Specifically, the lead vehicle on which the canopy is mounted was positioned directly under the roof cart of the MBTS and wood cribbing was used to support the canopy base and raise the lead vehicle's wheels off the floor. The top of the canopy was marked to show the middle ninth area of the total plan view area and the centroid of the canopy top was marked on the bottom surface of the canopy. A 12-in I-beam was cut and machined to cover and distribute the load over the middle ninth area of the canopy. The Sensotec 20,000-lb-capacity compression force transducer, which was previously calibrated, was placed on the I-beam directly above the centroid of the canopy and connected to the Fluke digital multimeter and the bridge amplifier. The 20-st-capacity hydraulic jack was placed on the force transducer and also made to contact the bottom of the MBTS roof cart. The displacement dial indicator was mounted on the floor of the operators station and in contact with the centroid location on the underside of the canopy.

The testing was conducted by utilizing the hydraulic jack to load the canopy in 1,000-lb increments, by reacting against the MBTS roof cart and by using the dial indicator to measure the canopy deflection. Once the canopy was loaded to 18,000 lb the load was removed and the residual deformation of the canopy was measured.

Results and Discussion

The results of the load test are shown graphically in figure 34. The maximum canopy deflection at a load of 18,000 lb was 0.152 in at the centroid and the residual deformation at the centroid, once the load was removed, was 0.009 in. This residual deformation is 5.9 pct of the total deflection and falls well within the maximum allowable residual deformation of 10 pct as stated in MSHA IR 1002. Therefore, the canopy is classified as substantial and is certifiable by a site-registered engineer.

GROUND PRESSURE EVALUATION

Objective

A ground pressure evaluation was conducted to determine the amount of pressure (pounds per square inch) that will be applied to the mine floor by the MUCH system tires during normal underground operations. This value is important in that the higher the pressure is at the tire-floor interface, the more likely the floor will be to deteriorate and ruts will be generated.

Results

For the evaluation, an average tire "footprint" was determined to be 10 in long by 8.5 in wide or 85 in^2. Each vehicle has four tires for a total ground contact area of

340 in^2. The unloaded weight of the lead vehicle was 11,600 lb, the discharge vehicle weight was 13,200 lb, the intermediate vehicles weighed 9,300 lb each, and the bridge conveyor weight was about 3,000 lb. The ground pressure per tire for each vehicle empty and loaded with 1 st of coal per vehicle is given in table 9. The ground pressure is the highest at the discharge vehicle with the bridge conveyor attached.

Table 9.–Tire ground pressure for loaded and unloaded vehicles

Vehicle	Weight, lb	Rear tire ground pressure, psi
Lead:		
Unloaded	11,600	34
Loaded	13,600	40
Intermediate:		
Unloaded	9,300	27
Loaded	11,300	33
Discharge:		
Unloaded	13,200	39
Loaded	15,200	45
Discharge and bridge:[1]		
Unloaded	14,700	48
Loaded	17,700	54

[1]Front tire ground pressures were 39 psi (unloaded) and 45 psi (loaded).

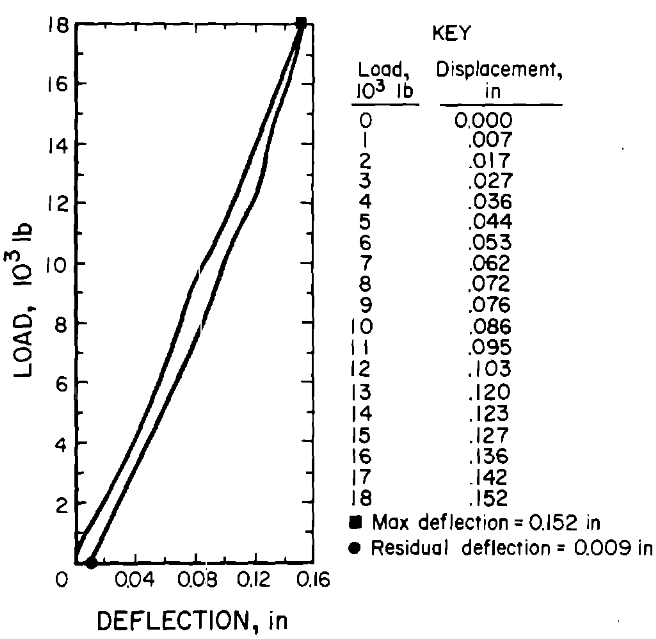

Figure 34.–Lead vehicle operator canopy load test, load versus deflection.

SURFACE TEST SUMMARY

The MUCH system was evaluated and modified during the surface test program to prepare the system for an in-mine production trial. The following are highlights of the surface test program.

- The 12-unit MUCH system successfully trammed inby and outby through a simulated underground work area with 20-ft-wide entries and crosscuts at 90° to each other and 48-ft-square pillars. During these tracking and retracking trials, the system tended to drift toward the inside radius of the 90° turns. When tramming in the inby direction the maximum observed variance of any vehicle from the path of the lead vehicle was 68 in. When tramming in the outby direction the maximum observed variance of any vehicle from the path of the discharge vehicle was 101 in. The system tracked and retracked with enough consistency to operate in an underground mine of similar dimensions.

- A maximum tracking variance of 35 in was observed during the simulated production cycle trial when the system was operating behind a simulated continuous miner making a 20-ft cut, a lift change, and second 20-ft cut.

- The time to complete a simulated place change was 3 min 20 s while tramming over a distance of 245 ft.

- A noise level survey was conducted. At the lead vehicle operator compartment, a noise level of 102 dBA was measured when the conveyor system was operating with no coal. At a haulage rate of 3.7 st/min of coal, 98 dBA was measured; at a haulage rate of 7.4 st/min, 97 dBA was measured at the operator station. The noise levels measured indicate that some form of hearing protection will be required for the MUCH system operators to maintain their 8-h noise exposure to 90 dBA or below.

- The conveyor chain speed was measured on a number of vehicles. The average conveyor chain speed with no load was 273 ft/min. The chain speed at a haulage rate of 7 st/min was 271 ft/min. The average startup time for the conveyor system was 1.36 s at no load and 1.86 s at a 4-st/min load rate.

- After a number of conveyor tests and numerous modifications on conveyor system, an acceptance test was conducted. During the test, the conveyor system operated for 337 min out of a total test time of 405 min (83.2 pct availability), the haulage rate averaged 10 st/min of mixed coal and rock, and the total amount of material conveyed was approximately 3,370 st.

o A drawbar pull test was conducted in which the empty MUCH system exerted a steady pull of 30,000 lb tractive effort; when the system was loaded with 10 st of material, the drawbar pull increased to 35,000 lb.

o During a rough terrain test the system trammed through a 2-ft-high mixture of mud, coal, rock, and crib blocks, a 6-in-deep water-mud hole, and numerous 6- to 8-in-deep ruts with no loss in tram speed or tracking ability.

o The redesigned operator canopy was load tested with a static load of 18,000 lb applied to the center ninth area of the canopy. The maximum deflection was 0.152 in and the residual deformation once the load was removed was 0.009 in. The canopy is classified as substantial and is certifiable by a State registered engineer.

o A ground pressure evaluation was conducted. The highest ground pressure was at the rear axle of the discharge vehicle with the bridge conveyor attached and a full load of coal on the discharge vehicle and bridge conveyor. The maximum rear tire ground pressure was 54 psi.

SYSTEM MODIFICATIONS

Numerous modifications were made to the MUCH system to prepare it for in-mine trials and to correct observed deficiencies found during surface testing. A summary of the modifications is given in appendix F. The system modifications are discussed in the following sections.

MSHA EXPERIMENTAL PERMIT APPROVAL

In order to allow an in-mine trial of the MUCH system, an experimental permit had to be acquired from the Mine Safety and Health Administration (MSHA) Approval and Certification Center. The effort to acquire an experimental permit was initiated by a meeting of Bureau, Boeing Services, and MSHA personnel on January 24, 1985, at the MSHA Approval and Certification Center, Triadelphia, WV. The purpose of the meeting was to discuss the procedures of permit acquisition, to find out what materials MSHA currently had concerning the MUCH system (JMMD had at one time initiated an approval effort), and to acquire information on any new changes in permit requirements that might effect the system. The formal application for the experimental permit was sent to MSHA under Company Application Code No. 112525 on July 29, 1985. After numerous drawing changes, many MUCH system revisions, and much correspondence with MSHA, experimental permit approval was received for the system on October 1, 1986. Experimental permit No. EP-541-0 as issued for the MUCH system had a duration of 6 months and with a request to MSHA, was renewed on April 7, 1987, for an additional 6-month period.

During the approval process, the following electrical deficiencies were defined and resolved to meet MSHA's requirements.

The electrical control circuits of the MUCH system, as designed by JMMD, operated at a voltage of 460 V ac line-to-line. As stated in a MSHA memorandum, dated October 25, 1982, "The voltage of alternating current control circuits shall not exceed nominal 120 V line-to-line." The 460-V ac MUCH control circuits were in violation of MSHA regulations and had to be dropped to 120 V to be approvable. The electrical control circuit was modified by installing a Mining Control, Inc., 24610 460-V primary to 120-V secondary, 2-kV•A, stepdown control transformer contained in an approved (X/P 2504-1) enclosure mounted on the discharge vehicle adjacent to the system's main electrical control box. The low-voltage secondary was equipped with 15-A fuses to protect the control circuitry wiring. The control voltage reduction necessitated replacement of the main circuit breaker (CB-1), shunt trip coil, and all magnetic coils for the motor control contactors and time delay relays.

The emergency stop circuit was not-fail safe because normally open contacts at the emergency stop switches had to be closed to activate the shunt trip coil and open the contacts of the main circuit breaker. If the emergency switch contacts were not functional because of corrosion or mechanical problems, the circuit could not be completed to activate the shunt coil and open the breaker contacts. The emergency stop circuit and shunt trip coil were changed so that a closed circuit was required to activate the shunt trip coil and hold the main circuit breaker contacts closed. The contacts of emergency stop switches were changed from normally open to normally closed so that when the switches were activated, the contacts would open and break the circuit to the shunt trip coil and open the main breaker contacts.

The trailing cable of the MUCH system was originally a 1/0, three-conductor, 90° C cable rated at an approximate ampacity of 219 A for underground use. The total full load motor current of the system is approximately 374 A and a normal load that would be expected during underground operations over a given period of time would be approximately 250 A. Obviously, the 1/0 trailing cable was of insufficient ampacity to meet MSHA requirements. The 1/0 cable was replaced with a three-conductor, 3/0 G-GC, 2,000-V, 90° C cable with a maximum ampacity of 294 A. The 1/0 size power cable between the discharge vehicle and intermediate vehicle 10 was replaced with 3/0 cable; and 1/0 power cables between intermediate vehicles 10 and 9 and between 9 and 8 were replaced with 2/0 cable.

The electrical fault ground-check circuit was originally connected to grounding studs in the explosion-proof (XP) box of each vehicle. This was not an MSHA approved circuit because an open ground fault circuit to an individual vehicle would not be detected if the ground-check circuit to any other vehicle was continuous. The circuit was revised such that the ground check could only be connected to the ground stud in the lead vehicle, thereby providing one continuous path instead of a parallel path at each vehicle.

The headlight circuit on the discharge vehicle was originally energized through a connection box located on the right side of the vehicle, which did not meet MSHA specifications. The circuit was changed to include a two-pole lever action pushbutton switch mounted in an XP enclosure in lieu of the connection box.

Inspection of the MUCH electrical system components revealed that all electrical motor thermal overload relay heater elements were improperly sized to adequately protect the motors during normal operation. All heater elements were replaced with properly specified elements.

The following refurbishment, replacement, and/or repair actions were required prior to the MSHA on-site inspection.

1. Electrical cable conduit and clamps were installed or relocated to provide sufficient protection to electrical cables.

2. The routing and suspension of electrical cables between vehicles was modified to prevent mechanical damage.

3. The trailing cable ground-check circuit was checked for proper operation.

4. Certification tags on electrical components were replaced or relocated to permit easier access during field inspection.

5. The cable for the pager phone system was inspected and repaired as necessary.

6. All hose conduit damaged during the test program was replaced or repaired.

7. All cable packing glands were checked for proper clearances.

8. All control box covers and edges were polished and checked for proper clearances.

9. All electrical components in control boxes were tagged for easy identification.

10. All electrical connections and wire terminals were checked and tightened as necessary.

11. Electrical layout and wiring diagrams were updated to include all electrical modifications and corrections.

OPERATOR CANOPY REDESIGN

The operator canopy, which protects the lead vehicle operator, was redesigned and load tested. During tramming operation of the system in the EMTA, the top of the canopy came into contact with the simulated ribs of the EMTA thereby bending the canopy's vertical supports (fig. 35). Although this original JMMD canopy design met MSHA requirements for vertical load capacity, it was very weak under horizontal loading conditions. After the three vertical supports were deformed by rib contact, the ability of the canopy to support the MSHA-required vertical load was reduced to an unacceptable level. This required redesign of the canopy to increase the strength and number of canopy supports.

In the new design (figs. 36-37), the three vertical supports were replaced by four vertical supports. The new supports are constructed of rectangular steel tubing instead of threaded rod as was used in the original design. Each of the four vertical supports consists of a length of 4-in-long by 2-in-wide by 1/4-in-thick rectangular steel tubing welded to the canopy and a length of 5-in-long by 3-in-wide by 3/8-in-thick rectangular steel tubing welded to the base of the operator compartment. The 4- by 2-in tubing fits inside the 5- by 3-in tubing and is secured in place by locking pins inserted through holes drilled in both lengths of tubing. The modified canopy has an operating height range of 42 in to 56 in, and is adjustable in 3.5-in increments over the height range. The canopy top strength was increased by using 1/2-in-thick steel plate instead of the original 3/8-in plate and by increasing the size of the steel tubing frame. Additional structural modifications were made to increase the strength of the floor area. Upon the completion of the installation of the modified canopy the structure was load tested and met MSHA requirements.

ELEVATION OF CONVEYOR DECK

In the original JMMD design there was approximately 1.5 in of clearance between the forward edge of a vehicle's coal receiving hopper and the bottom surface of the conveyor deck of the next forward vehicle when the system was on level terrain, as shown in figures 38 (top) and 39 (top). During tramming and maneuvering of the system it became obvious that additional clearance was needed in this area.

When tramming over uneven terrain, especially when one vehicle is twisting along the longitudinal center line relative to the next as when one wheel runs over a crib block, the vehicles bind together in this throat area (fig. 40, top) and the tracking ability of the vehicle is lost. To alleviate this problem, the frame of each vehicle was modified to raise the discharge end of the conveyor deck by approximately 5 in. This modification increased the height of the throat area from 13 in (fig. 40, top) to 18 in (fig. 40, bottom) and increased the hopper clearance from 1.5 in to 8 in (figs. 38-39, bottom).

CABLE HANDLING TRAY

Cable handling trays were fabricated and installed through the drawbars between vehicles. The trays (fig. 41), fabricated from 8-in C-channel, pivot in the middle to help reduce cable and hose damage. The trays support the power cable, control cable, telephone cable, and water hose between vehicles.

DISCHARGE VEHICLE STEERING SYSTEM

The hydraulic steering unit on the discharge vehicle was inadequate to allow the system to tram through the EMTA in the outby direction. A greater steering angle was needed; therefore, the linkage system was redesigned and modified to increase the steering angle and still utilize the existing cylinder. A bell crank arrangement was manufactured, and the steering cylinder was relocated. Figure 42 shows the arrangement.

EMERGENCY SHUTDOWN SYSTEM

During the surface tests it became obvious that an improved method of emergency shutdown of the MUCH electrical system was required. The system as received had only three locations of emergency shutoff, one at the lead vehicle operator station and one on each side of the discharge vehicle; these two locations are approximately 225 ft apart. In an underground situation neither the lead nor discharge vehicle operator can see the middle third of the system. If a problem occurred at a location not visible to either operator there was no way to shut down the entire system. To remedy this situation during surface testing, an emergency shutdown pull cord was installed along the center of the system for the entire length, which allowed complete system shutdown from any location (fig. 43). It must be noted that although this shutdown system worked well during the surface testing of the MUCH system, it was not designed for underground use.

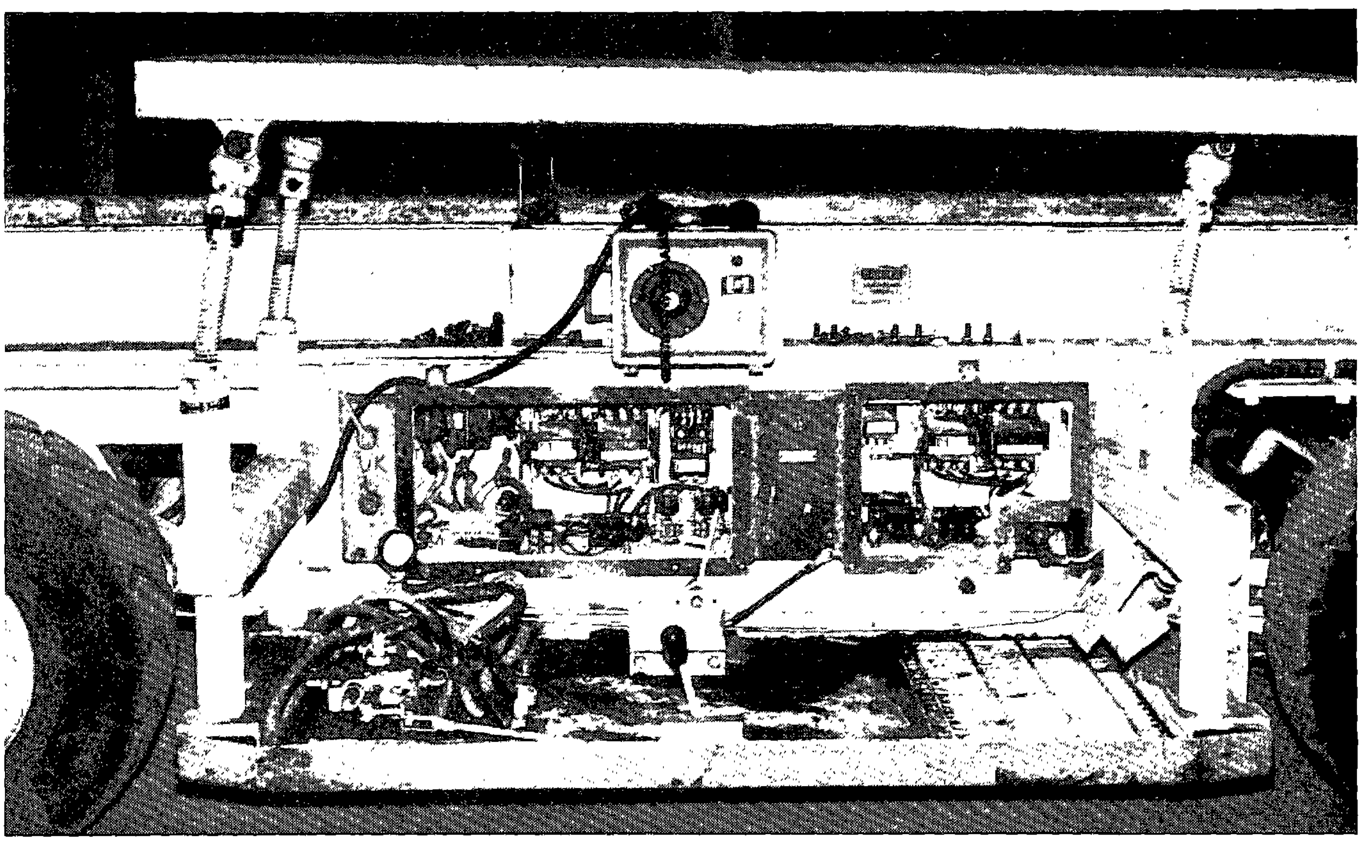

Figure 35.-Original operator canopy.

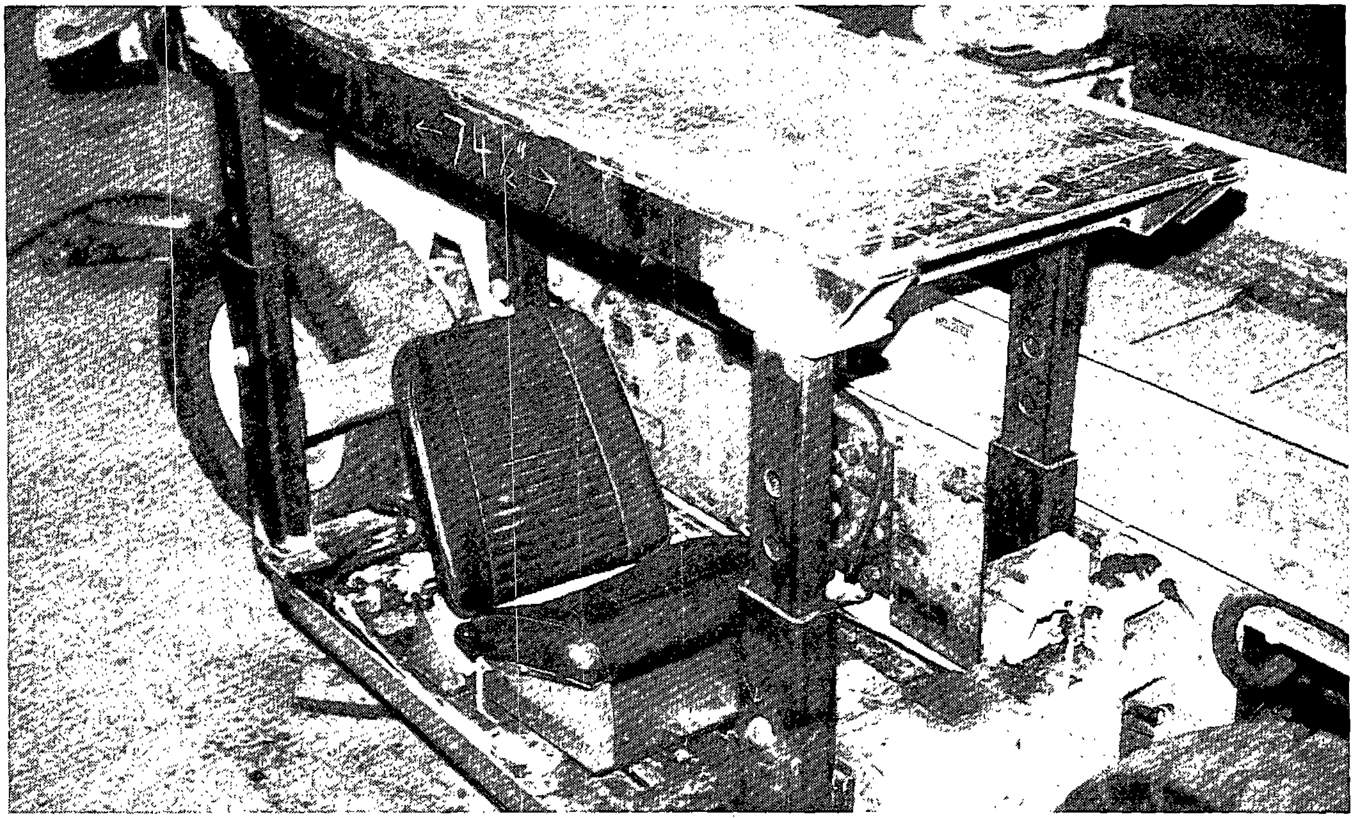

Figure 36.–Lead vehicle operator canopy.

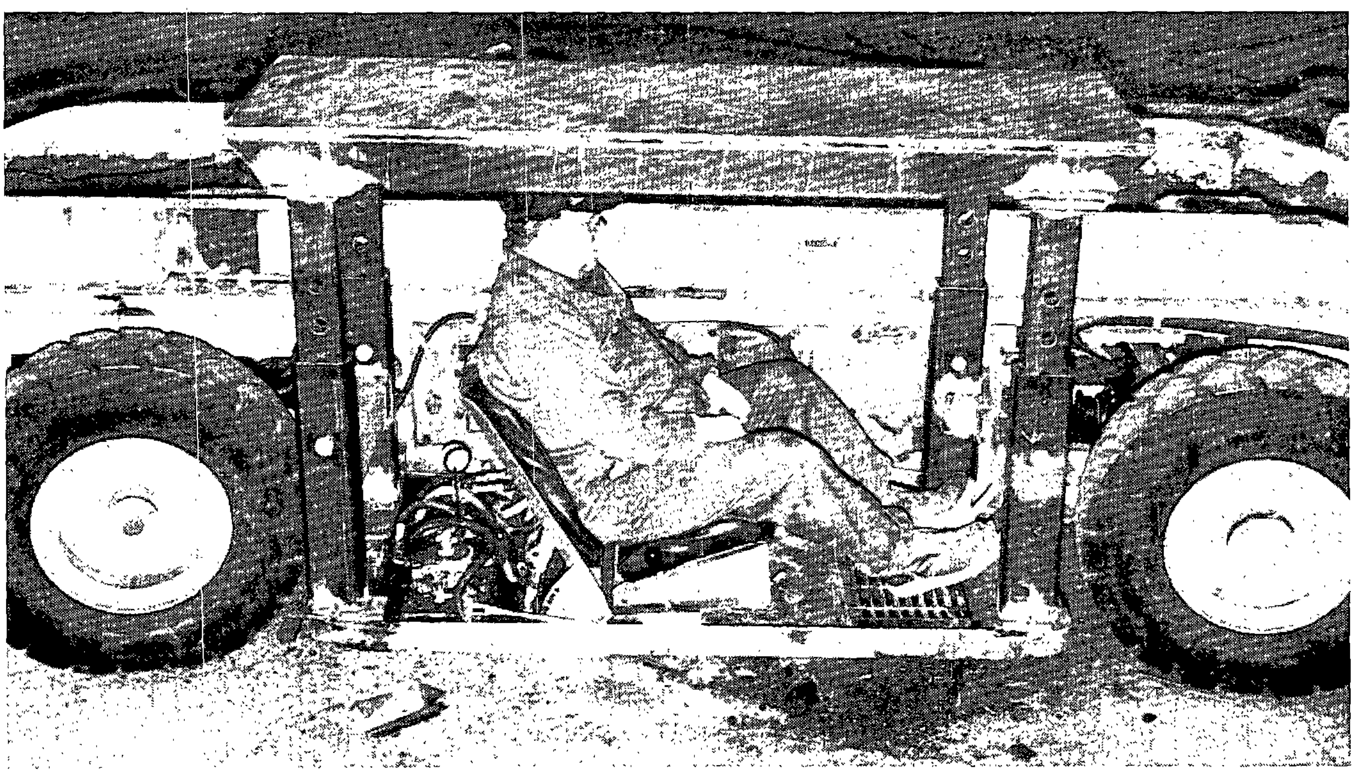

Figure 37.–Operator in lead vehicle.

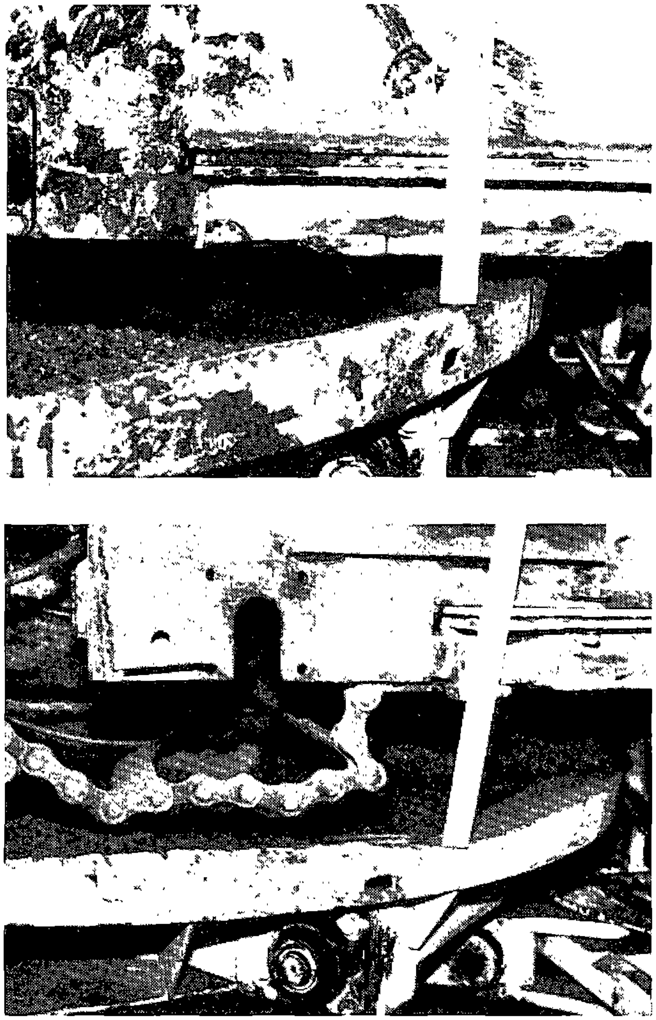

Figure 38.–Clearance between hopper and deck–original design (top) and modified design (bottom).

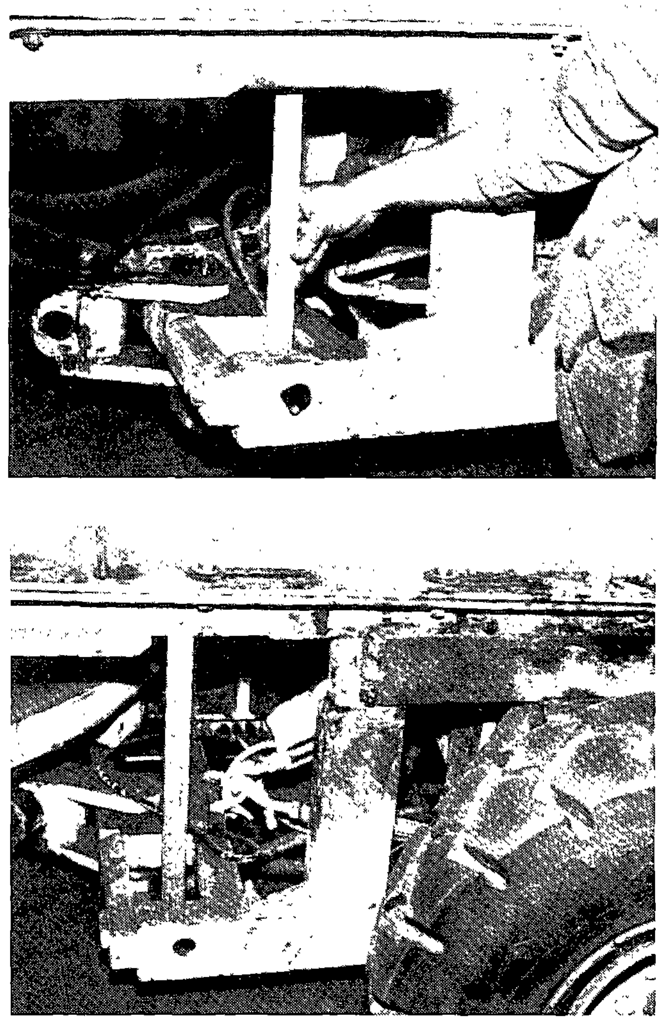

Figure 39.–Conveyor deck. Before modification, 1.5-in throat clearance (top); after modification, 8-in throat clearance (bottom).

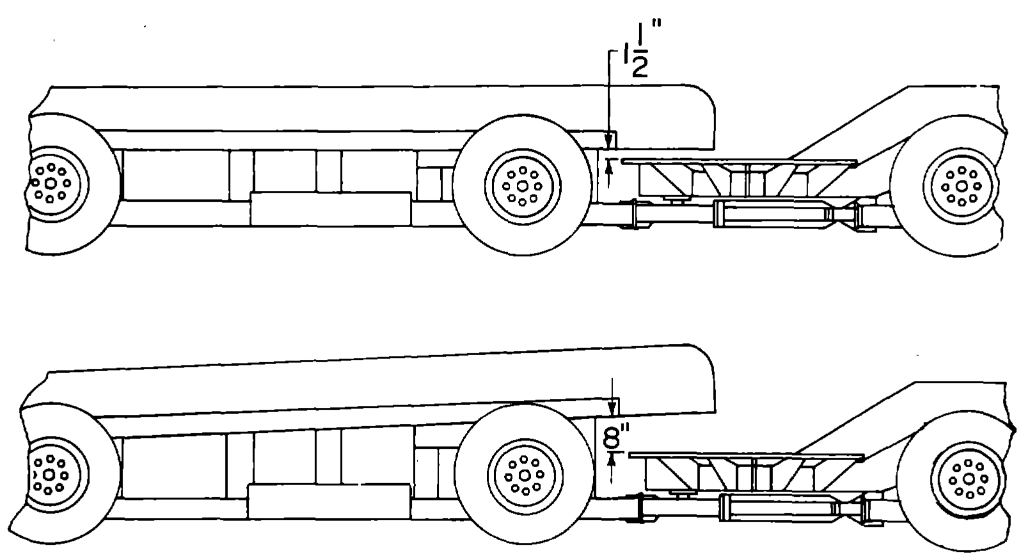

Figure 40.–Height of throat area–original design (top) and modified design (bottom).

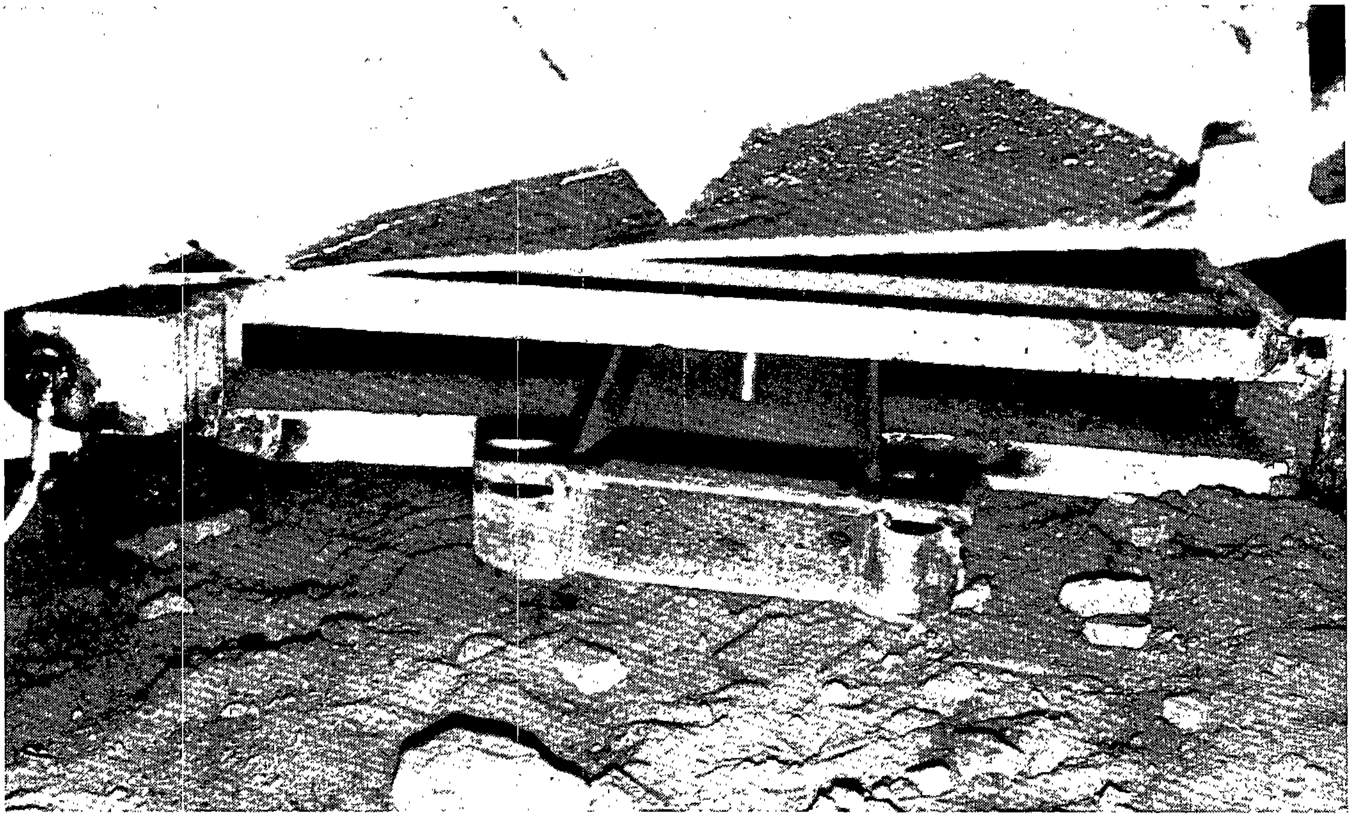

Figure 41.-Cable handling tray.

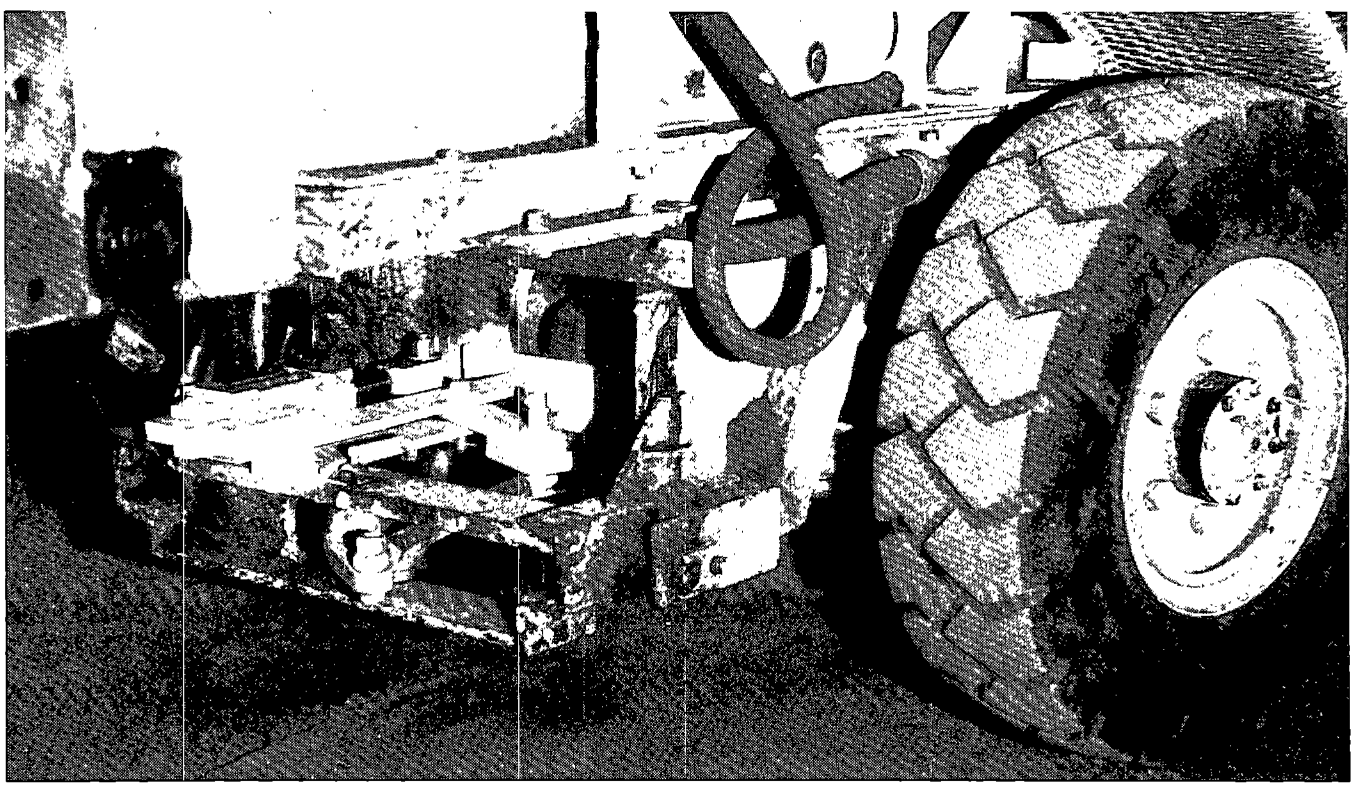

Figure 42.-Discharge vehicle hydraulic steering system arrangement.

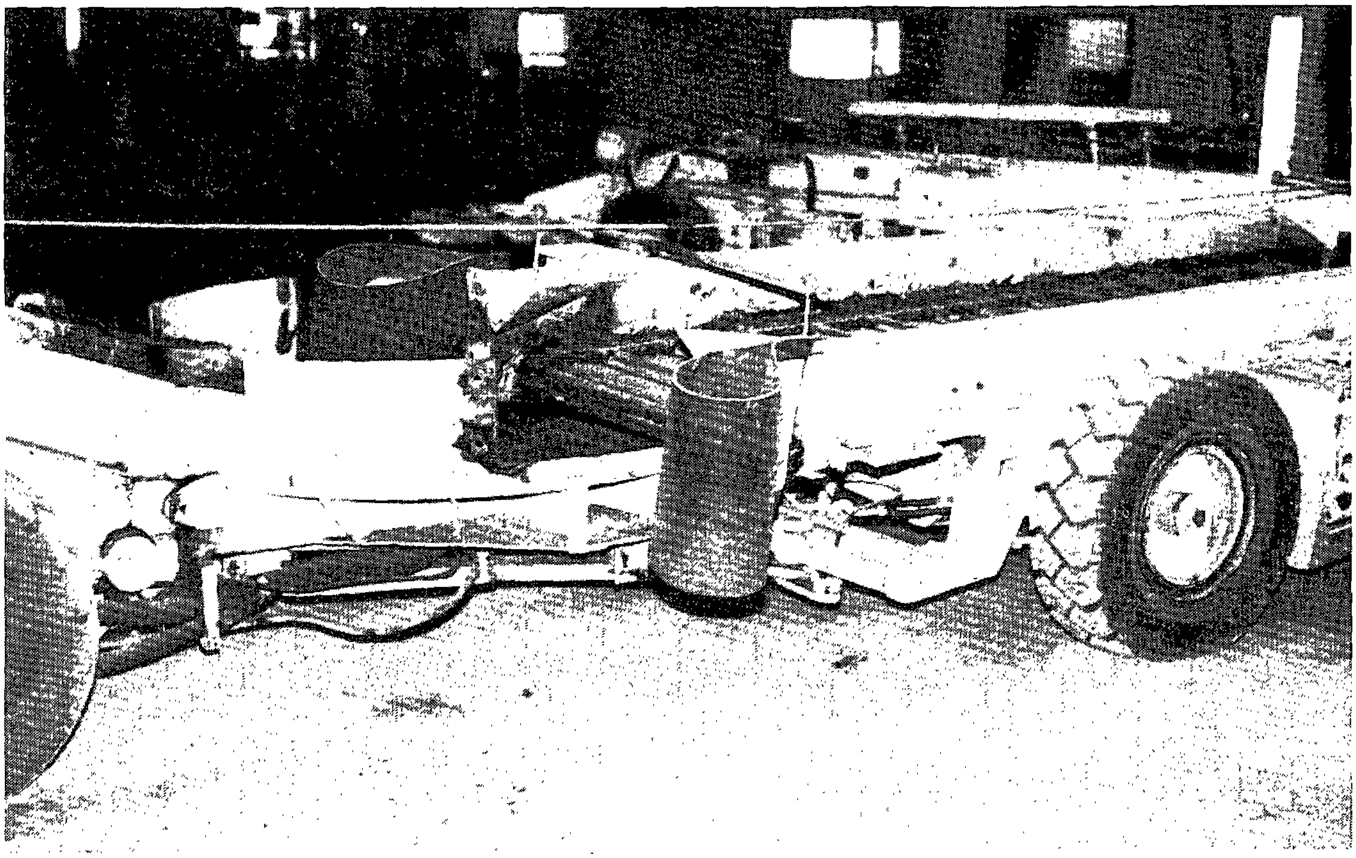

Figure 43.–Emergency shutdown pull cord.

CONVEYOR CHAIN SLACK ADJUSTMENT MECHANISM

During coal haulage trials, problems occurred from fines buildup on the conveyor decks, which significantly increased haulage power requirements. One major cause of these problems was the slack in conveyor chain. As the MUCH system was originally designed, the only way of removing slack from the conveyor chain was by physically removing one or more links from the chain; but this could only be accomplished when 2 in or more of slack was available. Under extreme circumstances 2 in of chain slack was sufficient to allow a buildup of approximately 5 in of fines on the conveyor deck. To correct this problem, a conveyor-chain slack adjustment mechanism was designed and installed on the vehicles.

The mechanism, as shown in figures 44 and 45, consists of two adjustment screws–one on each of the conveyor sideboards. The sideboards, which support the tailshaft of the conveyor chain, were cut approximately 65 in from the tailshaft, and the sideboard mounting holes were slotted to allow 5 in of movement. The adjusting screws are mounted on the forward half of the sideboard, and used to position the rear half of the sideboards and tailshaft to adjust conveyor-chain tension. The sideboard mounting bolts hold the sideboards and tailshaft in position when the chain adjustment is completed.

CONVEYOR CHAIN HOLDDOWNS

In the course of evaluating the conveyor system it became obvious that an effective conveyor chain holddown system was needed to help prevent fines buildup on the conveyor decks. Prior to the final conveyor acceptance test, a chain holddown system was installed on all vehicles (fig. 31). The holddowns consisted of 2- by 2- by 0.25-in steel angles bolted to the conveyor sideboards with 0.25-in clearance between the top of the chain and the bottom of the angle. The holddowns extend the entire length of the conveyor deck from the end of the loading hopper to about 4 in past the tail sprocket, which minimizes the amount of material that could be pulled in between the chain and the holddowns. A 2.5- by 0.25-in steel strip was welded between the legs of the angle to eliminate another potential area of material buildup. The chain holddown mounting holes in the rear portion of the conveyor sideboard were slotted to allow chain slack adjustment without removing the holddowns (fig. 46).

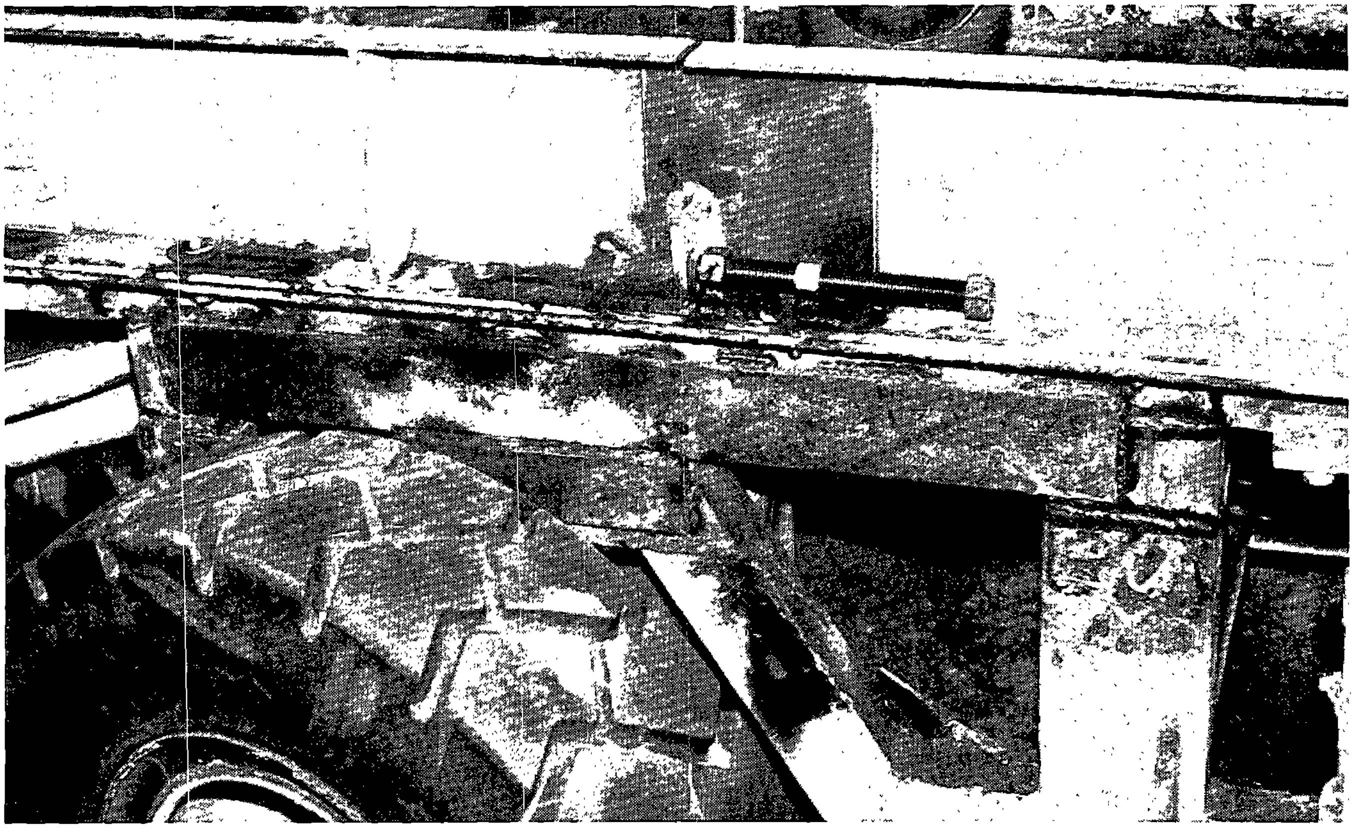

Figure 44.-Conveyor chain slack adjustment screw.

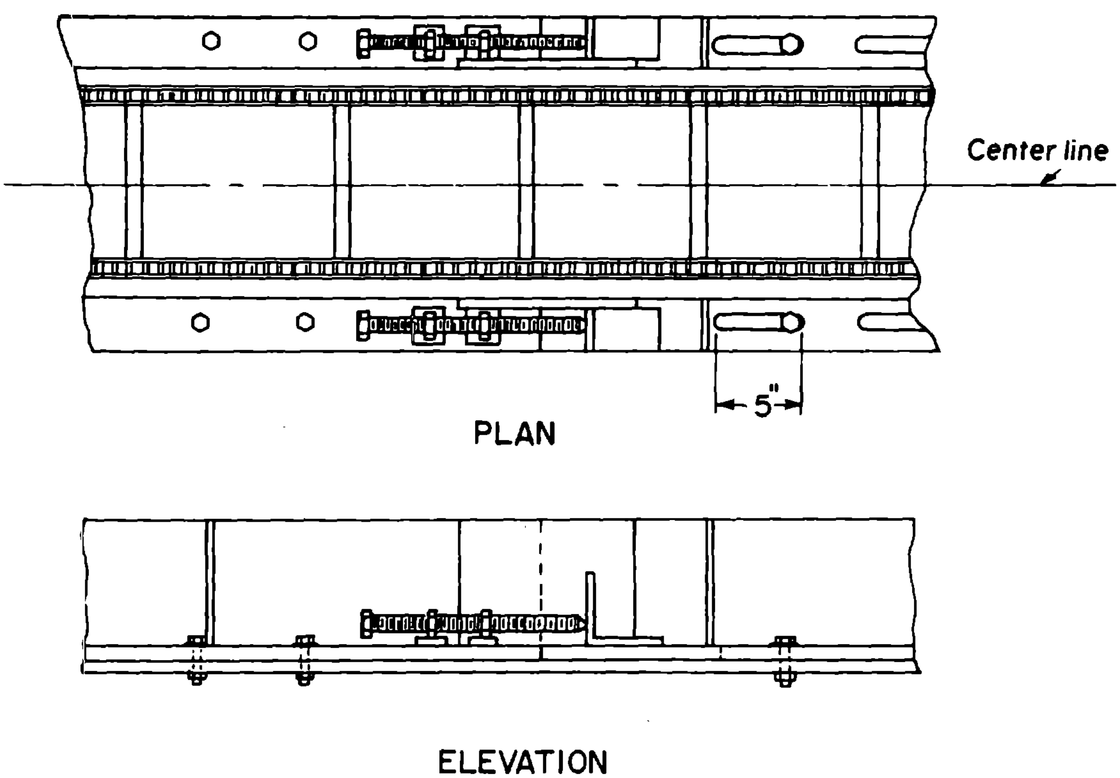

Figure 45.-Conveyor chain slack adjustment mechanism.

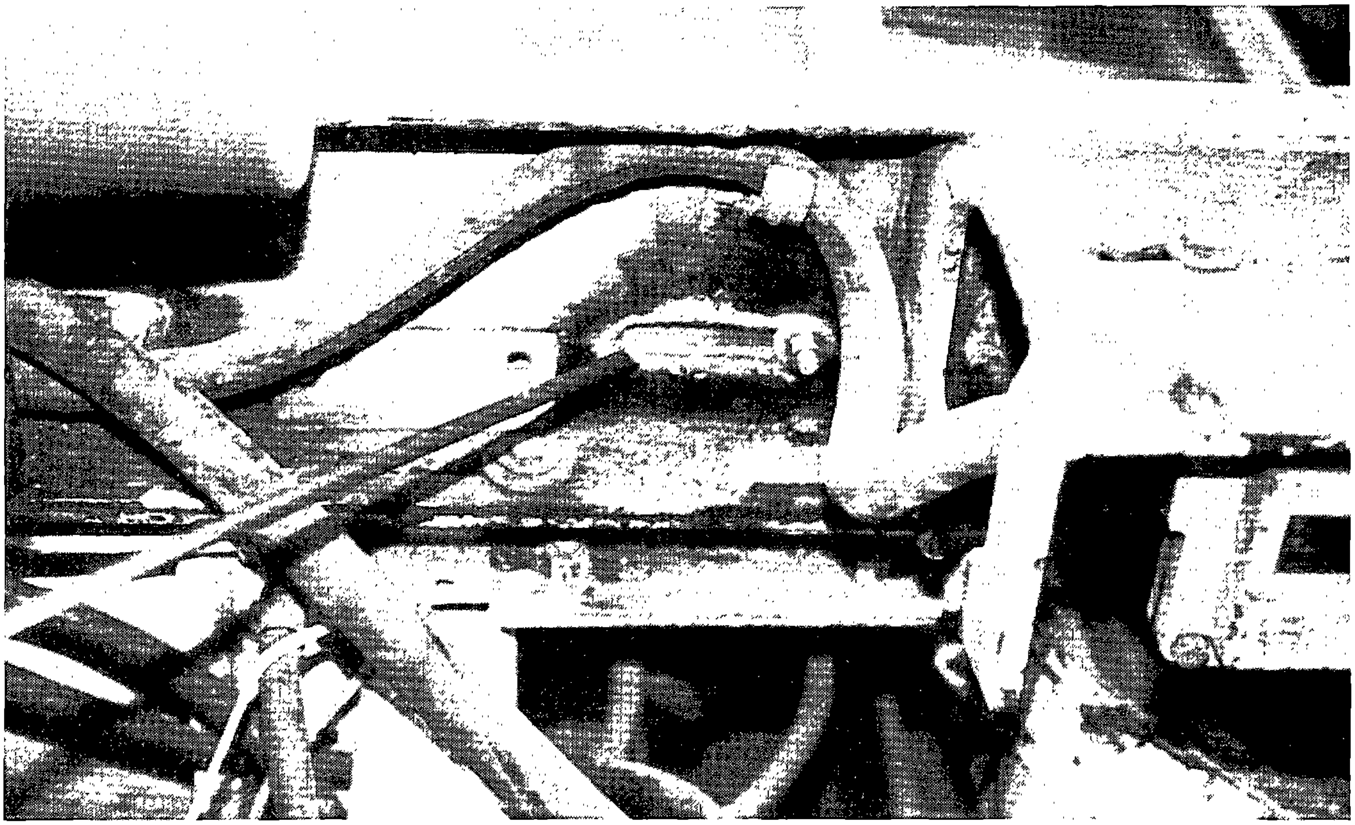

Figure 46.–Chain holddown mounting hole.

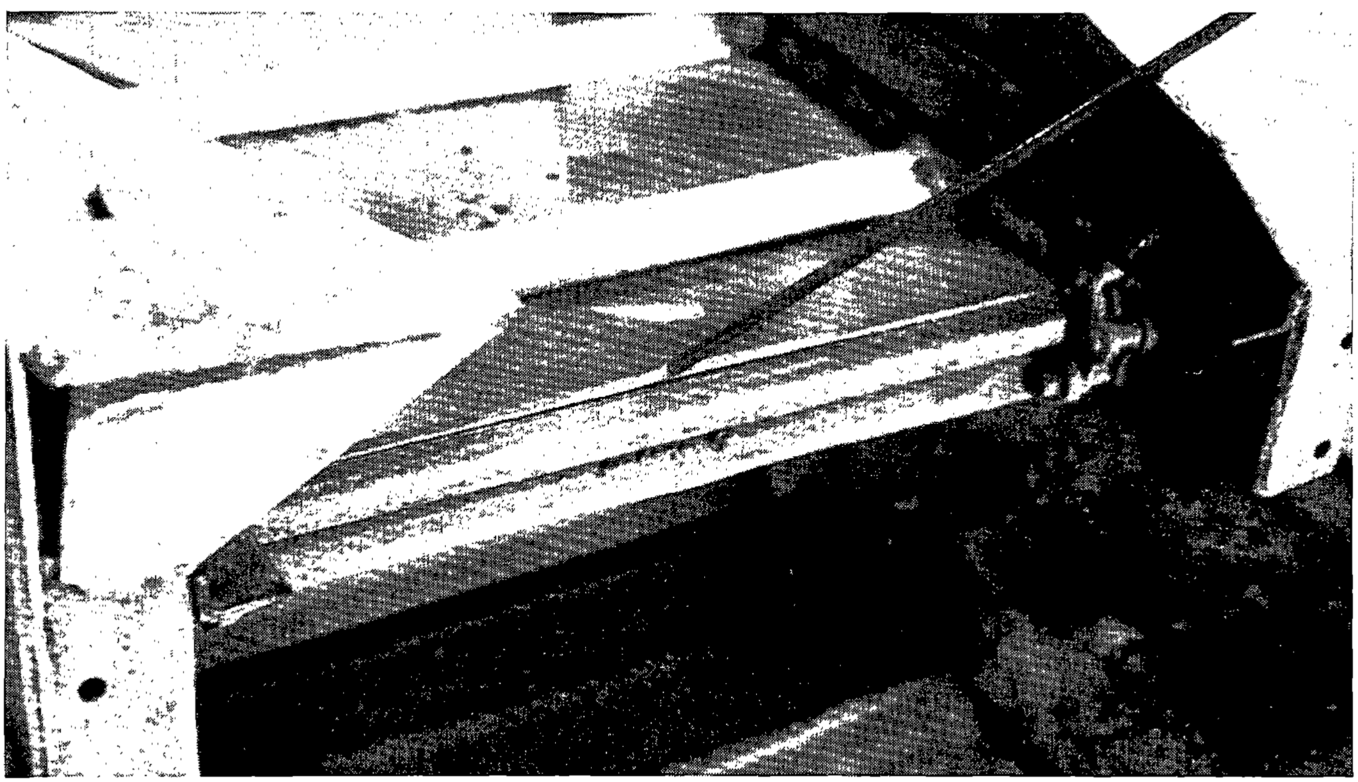

Figure 47.–Conveyor deck extension plate.

CONVEYOR DECK EXTENSION

As discussed in the "Conveyor System Tests" section, there was a material conveying problem because of a gap between the end of the conveyor deck and the tail sprocket shafts (fig. 29). This gap allowed chunks of material to get caught and either jam the conveyor or elevate the chain. This problem was amplified by the addition of the chain tensioning system by which the chain tension was increased by moving the sideboards. Increasing the chain tension also moved the tail sprocket shaft away from the conveyor deck, increasing the size of the gap. This problem was corrected by the addition of a 0.25-in-thick steel plate to bridge the gap between the conveyor deck and the tail sprocket shaft (fig. 47). The plate lies on the conveyor deck with the rear edge as close as possible to the tail sprocket shaft. The two outside edges of the plate are welded to the two conveyor sideboards so that the positioning between the plate and the sprocket shaft remains constant when the conveyor chain tension is adjusted. The leading edges of the plate are chamfered so as not to bind the conveyor chain.

CONCLUSIONS

Surface testing showed that the MUCH system has the potential to substantially increase the productivity of a room-and-pillar mining system. The following items are recommended to improve its general functioning in an underground operation.

- An improved emergency shutdown system should be designed and installed on the system to allow electrical deactivation from any point along either side of the conveyor train.

- Voice-activated headsets should be used by the lead and discharge vehicles operators to permit hands-off communications.

- State-of-the-art noise reduction technologies should be applied to reduce the conveyor system noise levels.

- MSHA-approved in-line cable connectors should be installed in the power and control cables between vehicles. These connectors would reduce vehicle change out time.

- All conveyor drive components should be strengthened by at least 50 pct.

- A positive means of locking the foam-filled tires to the rims is needed to prevent the tires from spinning on the rims.

- A cable reel mounted along the panel belt for the MUCH system trailing cable would help to limit the possibility of the discharge vehicle tramming over the cable.

- An adequate supply of spare parts should be available at the mine site to minimize downtime. Appendix E is a recommended spare parts list.

- A thorough operator's training program should be conducted for the face production personnel prior to the start of an in-mine trial.

- Additional lighting along the length of the system would be an advantage for both tramming and safety.

- Audio signal on the system to warn the persons when it moves forward or rearward.

- Rearview mirror in the lead vehicle operator cab to permit the operator to see behind the lead vehicle without turning around.

APPENDIX A.–MUCH SYSTEM SPECIFICATIONS

The MUCH system is designed to be used in an underground room-and-pillar mining system. Because it is a versatile system, it can be used in a highwall operation also.

Total system length (12 units)	ft	250
Estimated total weight (12 units including bridge conveyor)	lb	120,800
Power, 3-phase, 60 Hz	V ac	460
Minimum turning radius	ft	24
Mine configuration:		
Entry-crosscut width	ft	18-20
Entry-crosscut angle	deg	60-90
Minimum working height, in:		
With canopy on lead car		60
Without canopy on lead car		48

Vehicle units	Lead vehicle	Intermediate vehicle	Discharge vehicle
Frame:			
Length . . . ft	23.25	21.75	[1]21.75
Active length . . . ft	19.75	19	[2]19
Height . . . in	41	44	41
Width . . . ft	6.5	6	6
Canopy (adjustable) . . . in	42-56	NAp	NAp
Conveyor:			
Chain speed . . . ft/min	280	280	280
Chain width . . . in	30	30	30
Trough height . . . in	9	9	9
Motor . . . hp	15	15	15
Capacity . . . st/min	12	12	12
Tram:			
Speed . . . ft/min	80	80	80
Motor . . . hp	7.5	5	7.5
Wheel size	8.25 by 15	8.25 by 15	8.25 by 15
Tread width . . . ft	5	5	5
Brakes	Disk, hydraulic	Spring activated, electrically released	Disk, hydraulic
Drive	Front wheel	Front wheel	Rear wheel
Steering:			
Forward	Manual	Automatic	NAp
Reverse	Automatic	NAp	Manual
Communications	Pager phone	Optional on 1 unit	Pager phone
Hydraulics:			
Pump	1 unit	NAp	1 unit
Motor . . . hp	1	NAp	1
Headlights:			
Number	2	None	2
Voltage . . . V ac	11	NAp	11

NAp Not applicable.
[1]28.25 ft with bridge conveyor.
[2]22 ft with bridge conveyor.

APPENDIX B.–CONVEYOR TEST BREAKDOWN AND REPAIR LOG

Cumulative run time, min	Av loading rate, st/min	Machine	Description
3/14/84:			
10	2	MUCH	Breaker on intermediate vehicle 8 tripped out, reason unknown. Breaker was reset.
18	2	HFB	Main breaker on HFB tripped out. Unable to reset breaker. Problem was diagnosed as a poor connection on source side of the breaker. Connections were tightened, resolving the problem.
50	2	HFB	Main breaker on HFB tripped out, reason unknown. Breaker was reset.
3/15/84:			
59	2	MUCH	Clutch slipped on intermediate vehicle 10. Vehicle was jogged back and forth to reset the clutch and unjam the conveyor.
59.1	2	MUCH	Intermediate vehicle 9 breaker tripped, reason unknown. Breaker was reset.
63.1	2	MUCH/HFB	Power center tripped out, reason unknown. Breaker was reset.
64.1	1	MUCH	Intermediate vehicles 2 through 10 and the discharge vehicle breakers tripped out, reason unknown. Intermediate vehicle 9 sheared a pin. Shearpin was replaced and the breakers were reset.
65.1	1	MUCH	Key sheared on conveyor drive shaft on intermediate vehicle 9. Keyseat was damaged and driveplate bore was galled. To repair the conveyor driveshaft, the driveplate was welded solidly to the driveshaft.
3/16/84:			
68.6	1.5	MUCH	Clutch slipped on intermediate vehicle 10 and was reset.
69	1.5	MUCH	Clutch slipped on intermediate vehicle 10 and was reset.
76	1.6	MUCH	Breakers on intermediate vehicles 7 and 8 tripped out, reason unknown. Intermediate vehicle 9 sheared a pin. Breakers were reset and shearpin was replaced.
3/20/84: 79	0	MUCH	Discharge vehicle speed sensor shaft seized and failed the stub shaft on end of tailshaft (speed sensor sprocket drive). Speed sensor was electrically bypassed. Replacement speed sensor and tailshaft were later replaced.
3/28/84: 215	0		No breakdowns.
3/29/84:			
283	2	MUCH	Breaker on intermediate vehicle 4 tripped out, reason unknown. Breaker was reset.
284	2	MUCH	Breaker on intermediate vehicle 8 tripped out, reason unknown. Breaker was reset.

See explanatory notes at end of tabulation.

Cumulative run time, min	Av loading rate, st/min	Machine	Description
3/29/84:			
298	2	HFB	Main breaker on HFB tripped out. Problem was a poor HFB connection on load side of the breaker. Breaker was overheated in that area. Connection was tightened, resolving the tripping problem.
3/30/84:			
299	2	MUCH	Breakers on intermediate vehicles 6 and 7 tripped out, reason unknown. Breakers were reset.
341	4	HFB	HFB conveyor hydraulic motor hose failed. The crimp failed, allowing hose to blow out end of the fitting. Crimp-type staple-lock fitting was replaced.
341.5	4	HFB	HFB was jammed. Wet coal overloaded conveyor system. Hopper was emptied manually. Also, 1 hydraulic fitting on conveyor motor was leaking, its O-ring was replaced.
346.5	4	MUCH	Breaker on intermediate vehicle 1 tripped out, reason unknown. Discharge vehicle sheared a pin during system troubleshooting. Breaker was reset and shearpin was replaced.
360.5	5	MUCH	Breaker on intermediate vehicle 4 tripped out, reason unknown. Breaker was reset.
362.5	5	MUCH	Intermediate vehicle 3 conveyor drive chain broke and taper lock bushing bolt (on conveyor drive sprocket) sheared. Motor fan was also hitting against guard. Chain was repaired; bolts and fan were replaced.
4/03/84:			
364	3	MUCH	System shut down for unknown reason. System started up without a problem.
367	3	MUCH	Intermediate vehicles 7 and 8 breakers tripped out, reason unknown. Breakers were reset.
370	3	MUCH	Intermediate vehicles 4 and 6 breakers tripped out, reason unknown. Breakers were reset.
374	3	MUCH	Intermediate vehicles 4 and 6 breakers tripped out, reason unknown. Breakers were reset.
375	3	MUCH	Intermediate vehicles 4, 6, and 7 breakers tripped out, reason unknown. Breakers were reset.
4/04/84:			
375.2	2	MUCH	Intermediate vehicles 1, 2, 3, and lead vehicle did not start. Problem was traced to a loose wire in electrical XP box on intermediate vehicle 4 (control power circuit). Instantaneous overload in breaker was set to its highest level, eliminating the nuisance tripping problem in intermediate vehicle 4.
382.2	2	MUCH	Breaker on intermediate vehicle 3 tripped out, reason unknown. Breaker was reset.

See explanatory notes at end of tabulation.

Cumulative run time, min	Av loading rate, st/min	Machine	Description
4/04/84:			
402.2	1.5	MUCH	Intermediate vehicle 3 sheared a pin. Pin was replaced.
422.2	1.5	MUCH	Clutch on intermediate vehicle 1 slipped. It was reset.
436.2	1.5	MUCH	Clutch on intermediate vehicle 1 slipped; it could not be reset. Clutch was taken out and replaced with a solid coupling (Lovejoy).
4/05/84:			
439.2	1	MUCH	Intermediate vehicle 6 sheared a pin. Shearpin was replaced.
443.2	.8	MUCH	Clutch on intermediate vehicle 10 slipped. It was reset.
446.2	.8	MUCH	Breaker on intermediate vheicle 1 tripped out, reason unknown. Breaker was reset.
447.2	.8	MUCH	Breaker on intermediate vehicle 1 tripped out, reason unknown. Breaker was reset.
517.2	3	MUCH	Breakers on intermediate vehicle 5 and the discharge vehicle tripped out (thermal overload). After discharge vehicle motor cooled, breakers were successfully reset.
04/06/84:			
518.2	3	HFB	HFB was jammed. Ran conveyor back and forth to clear hopper.
525.2	3	MUCH	Intermediate vehicle 1 stopped. Reason unknown.
525.7	3	HFB	HFB was jammed. Manually shoveled the system and ran conveyor back and forth to clear system.
531.7	4	HFB/MUCH	HFB was jammed. Manually shoveled the system and ran conveyor back and forth to clear system. Intermediate vehicle 1 also stopped, reason unknown.
536.7	4	MUCH	Clutch on intermediate vehicle 10 slipped; it would not reset. Clutch was removed and replaced with a solid coupling (Lovejoy).
4/09/84:			
537.2	5	MUCH	Intermediate vehicle 3 stalled. It appeared that the overload trip was caused by fines buildup.
548.2	5	MUCH	Breaker on intermediate vehicle 8 tripped out, reason unknown. Breaker was reset. Intermediate vehicle 10 conveyor drive gearbox overheated. Temperature after 15 min of operation was 239° F. Haulage rate was 5 st/min average. It was found that the fines buildup around the conveyor drive chain had caused it to seize, significantly increasing the system load, and thereby increasing the gearbox temperature. After end of run on April 9, chain was removed, cleaned, and reinstalled. Gearbox no longer overheated.
550.2	5	MUCH	Breaker on intermediate vehicle 8 tripped out, reason unknown. Breaker was reset.

See explanatory notes at end of tabulation.

Cumulative run time, min	Av loading rate, st/min	Machine	Description
4/09/84:			
550.7	5	MUCH	Breaker on intermediate vehicle 8 tripped out, reason unknown. Breaker was reset.
551.7	6	MUCH	Breaker on intermediate vehicle 8 tripped out, reason unknown. Breaker was reset.
552.2	6	HFB	HFB conveyor hydraulic motor hose blew out of crimp fitting. (This hose was on the opposite port to hose that failed on 3/30/84.) Hose was replaced.
4/12/84:			
553.2	5	MUCH	Lead vehicle sheared pin. Shearpin bushing was also damaged. Pin and bushing were replaced.
554.2	4	HFB	HFB was jammed. Ran conveyor back and forth to clear system.
556.2	4	MUCH	Breakers on intermediate vehicles 7 and 8 tripped out, reason unknown. Breakers were reset.
556.3	4	MUCH	Breakers on intermediate vehicles 7 and 8 tripped out, reason unknown. Breakers were reset.
556.4	4	MUCH	Breaker on intermediate vehicle 7 tripped out, reason unknown. Breaker was reset.
556.5	4	MUCH	Breaker on intermediate vehicle 8 tripped out, reason unknown. Breaker was reset.
556.6	4	MUCH	Breakers on intermediate vehicles 7 and 8 tripped out, reason unknown. Breakers were reset.
560.6	4	MUCH	Breakers on discharge vehicle and intermediate vehicle 10 tripped out. Wire on load side of main breaker was loose. Connections were tightened and breakers were reset.

HFB Hopper-feeder-bolter.
MUCH Multiple-unit continuous haulage system.
XP Explosion-proof.

APPENDIX C.–MUCH SYSTEM COAL CARRYBACK LOSS-CONVEYOR TEST–JULY 31 TO AUGUST 3, 1984

	Weight of coal, lb
Lead vehicle: Outby end	369
Intermediate vehicle 1: Conveyor drive shaft area	215
Intermediate vehicle 2: Conveyor drive shaft area	318
Intermediate vehicle 3 (modified), 2,504 lb lost:	
Hopper cleanout port	225
Conveyor drive shaft area and pelican beak	681
Midship cleanout port	780
Outby cleanout port and intermediate vehicle 4 hopper cleanout port	818
Intermediate vehicle 4 (modified), 1,853 lb lost:	
Conveyor drive shaft area and pelican beak	353
Midship cleanout port and chain slots	668
Rear chain slots and outby cleanout port	832
Intermediate vehicle 5: Conveyor drive shaft area	155
Intermediate vehicle 6: Conveyor drive shaft area (caps open)	147
Intermediate vehicle 7: Conveyor drive shaft area	617
Intermediate vehicle 8 (modified), 3,240 lb lost:	
Hopper cleanout port	253
Conveyor drive shaft area	535
Midship cleanout port, chain slot, and outby	2,632
Intermediate vehicle 9 (vehicle misaligned)	109
Average loss of unmodified vehicles	264
Average loss of modified vehicles	2,258
Increase in coal spillage	854

APPENDIX D.–CONVEYOR ACCEPTANCE TEST LOG–OCTOBER 29, 1986

Clock time	Run time, min		Delay time, min		Event
	Run	Cumulative	Delay	Cumulative	
a.m.:					
8:00	0	0	0	0	Start test, 6 st coal on system.
8:20	20	20	0	0	3 st coal to system.
8:23	3	23	0	0	System shutdown, lead vehicle conveyor motor.
8:25	0	23	2	2	Restart system.
8:27	2	25	0	2	3 st coal added to system.
9:07	40	65	0	2	System shutdown.
9:08	0	65	1	3	Restart system.
9:10	2	67	0	3	2 st coal to system.
9:37	27	94	0	3	System shutdown.
9:38	0	94	1	4	Restart system.
9:45	7	101	0	4	1 st rock to system.
9:47	2	103	0	4	1 st rock to system.
9:50	3	106	0	4	System shutdown, lead vehicle conveyor motor.
9:52	0	106	2	6	Restart system, add 1 st rock.
9:55	3	109	0	6	System shutdown, intermediate vehicle 2, conveyor motor.
9:57	0	109	2	8	Restart system.
10:00	3	112	0	8	System shutdown, intermediate vehicle 5 conveyor motor.
10:02	0	112	2	10	Restart system.
10:05	3	115	0	10	System shutdown, lead vehicle conveyor motor.
10:06	0	115	1	11	Restart system.
10:16	10	125	0	11	System shutdown, lead vehicle conveyor motor.
10:17	0	125	1	12	Restart system.
10:26	9	134	0	12	System shutdown, lead vehicle conveyor motor.
10:28	0	134	2	14	Removed 2 st material from system.
10:30	0	134	2	16	Restart system.
10:42	12	146	0	16	System shutdown, lead vehicle conveyor motor.
10:45	0	146	3	19	Adjusted thermal trip setting on lead vehicle.
11:05	0	146	20	39	Restart system.
p.m.:					
12:20	75	221	0	39	Stop system, repair coupling on conveyor drive motor discharge vehicle.
12:30	0	221	10	49	Restart system.
2:00	90	311	0	49	System shutdown, lead vehicle conveyor motor.
2:15	0	311	15	64	Restart system.
2:30	15	326	0	64	System shutdown, intermediate vehicle 5 conveyor motor.
2:34	0	326	4	68	Restart system.
2:45	11	337	0	68	Stop test.

NOTE.–Total test time–405 min, total run time–337 min, total down time–68 min.

APPENDIX E.–RECOMMENDED SPARE PARTS FOR IN-MINE TRIAL

Item	*Quantity*
7.5-hp tram motor, frame 215 TC	1
5.0-hp tram motor, frame 215 TC	2
Cone drive tram speed reducer	1
1.0-hp hydraulic pump motor, frame 182 T	1
Lovejoy couplings, 1-1/2-in × 5/8 in	6
Speed switch chain, Morse 40 5B 80 pitch	4
Speed switch sprocket	4
Conveyor drive sprocket, 14-tooth	4
15-hp conveyor motor, frame 254 TDZ	2
Conveyor drive shaft with housing and bearings	2
Conveyor toe shaft assembly	2
Conveyor tail shaft assembly	2
Tram brake assembly	1
Headlight assembly	2
Conveyor speed reducer, lead vehicle	1
Conveyor speed reducer, intermediate and discharge vehicles	2
Tires on rims	2
Drive axle assembly (1 intermediate and 1 lead)	2
Nondrive axle assembly	2
Conveyor chain, including flights	2

APPENDIX F.—MUCH SYSTEM MODIFICATIONS SUMMARY

Modification	*Reason*
Voltage of electrical control circuits were reduced from 460 V ac to 120 V ac line-to-line.	MSHA memorandum stated "The voltage of alternating current control circuits shall not exceed nominal 120 V line-to-line."
460-V primary to 120-V secondary, 2-kV•A stepdown control transformer installed in the electrical control circuit.	Transformer was required to step down control circuit voltage from 460 V ac to 120 V ac.
15-A fuses added to control circuits.	To protect the control circuit components from overcurrents.
Main circuit breaker (CB-1) for system, shunt trip coil, and all magnetic coils for motor control contactors and time-delay relays were replaced.	Changes required because of reduction of control circuit voltage from 460 V to 120 V.
Emergency stop electrical circuit modified to provide fail-safe-operation.	Safety.
Size of system trailing cable increased from a 1/0, 3-conductor, 90° C rated cable to 3/0, 3-conductor, 90° C rated cable.	Amperage of 1/0 cable was insufficient to handle normally expected amperage requirements of system.
1/0 size cable between the discharge and intermediate vehicle 10 replaced with 3/0 cable, and 1/0 cable between intermediate vehicles 10 and 9 and beteen 9 and 8 replaced with 2/0 size power cable.	Do.
Electrical fault ground-check circuit was originally connected to grounding studs in the explosion-proof (XP) box of each vehicle. This was modified so that circuit was only connected to ground in the lead vehicle XP box.	Original grounding was not MSHA approvable because an open ground fault in an individual vehicle would not be detected.
Electrical connection box for the discharge vehicle headlight circuit replaced with 2-pole lever action pushbutton switch mounted in XP enclosure.	Connection box did not meet MSHA specifications.
All electrical motor thermal overload relay heater elements were replaced.	Original overload elements were improperly sized to adequately protect the motor.
Lead vehicle operator canopy was redesigned and load tested.	Original canopy was damaged during testing.
Frame structure of each vehicle was modified to elevate discharge end of conveyor deck by 5 in.	To reduce vehicle-to-vehicle interference during tramming over uneven terrain.
Cable support trays were fabricated and installed through the drawbars between vehicles to support cables and water hose between vehicles.	To help reduce cable and hose damage between vehicles.
Discharge vehicle steering system was modified by fabricating a new bell crank arrangement and relocating steering cylinder.	More steering angle was needed to improve steering ability of discharge vehicle.

Modification	*Reason*
Pull-cord type emergency shutdown system was installed for surface testing of the MUCH.	To provide complete shutdown of entire system from either side along the length of the system.
Conveyor chain slack adjustment system was designed and installed on vehicles.	To reduce amount of fines buildup on conveyor decks.
Conveyor chain holddowns were installed on vehicles.	To prevent fines buildup on conveyor decks.
Conveyor deck extension plate was added to each vehicle to bridge gap between end of the conveyor deck and conveyor chain tail shaft.	To prevent material from getting jammed between end of conveyor deck and tail shaft.
Shearpins located in conveyor drive mechanism were removed and shearpin couplings were welded solid.	During conveyor testing, the shearpins proved to be unreliable.
Autoguard torque clutches were removed from conveyor drive system and replaced with Lovejoy couplings.	Clutches failed and could not be reset.

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INT.BU.OF MINES,PGH.,PA 28977