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REMOTELY PLACED CONCRETE/GRAVEL COLUMNS FOR POINT SUPPORT WITH INNOVATIVE CONCRETE PLACEMENT DEVICE

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This report covers three separate areas of abandoned mine subsidence abatement technology. The first is the development of a cost effective method of remote placement of point support columns in abandoned mines through boreholes. The second is the design and test of a twin screw pneumatic pipefeeder for metering fill material into a pipeline for stowing in abandoned mine entries. The third area is a demonstration task to fill an abandoned mine tunnel using the high-efficiency pneumatic ejector and the pneumatic pipefeeder.

The first section is the design and development of a mechanical device to place very low slump concrete in an annular ring (doughnut) so as to build a cylindrical wall from floor to roof remotely through an 8-in diameter borehole is presented. The diameter of the annulus is about 8-ft at the roof and a 20-ft-diam base.

The second section of this report describes the design and testing of a co-rotating twin screw pneumatic pipefeeder to meter fill material into a pipeline for transport of distances over 400-ft.

The third section of this report discusses the successful demonstration of the High-Efficiency Ejector and Pneumatic Pipefeeder to pneumatically stow gravelintol an abandoned mine tunnel at Vandling, PA. The tunnel was approximately 600-ft in length, 14-ft in width and 5 to 8-ft high and was completely filled with 2400 tons of material.

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FOREWORD

This report was prepared by Burnett Associates, Inc. Framingham, MA under Contract number J0329004. The contract was initiated under the Abandoned Mine Lands Research Program. It was administered under the technical direction of the U.S. Bureau of Mines with Robert C. Dyni acting as Technical Project Officer. Michael L. Nowicki was the contract administrator for the Bureau of Mines. This report is a summary of the work recently completed as part of this contract during the period 26 March, 1992 to 26 November, 1993. This report was submitted by the author on 26 November, 1993.

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

cfm	cubic foot per minute
ft/min	foot per minute
ft	foot
ft/s	feet per second
gal	gallons
gpm	gallon per minute
h	hour
hp	horsepower
in	inch
lbf/h	pound (force) per hour
lbf	pound (force)
pct	percent
psig	pound (force) per square inch (gauge)
r/min	revolution per minute
ton/h	ton (short) per hour
V ac	volt, alternating current

INTRODUCTION

This report covers three separate areas of abandoned mine subsidence abatement technology. The first is the development of a cost effective method of remote placement of point support columns in abandoned mines through boreholes. The second is the design and test of a twin-screw pneumatic pipefeeder for metering fill material into a pipeline for stowing in abandoned mine entries. The third area is a demonstration task to fill an abandoned mine tunnel using the high-efficiency pneumatic ejector and the pneumatic pipefeeder.

The first section of this report describes research involving the development and testing of an effective method of remote placement of point support columns through boreholes. The technology is needed to provide local support in subsidence prone areas especially under surface structures. The design and development of a mechanical device to place very low slump concrete in an annular ring so as to build a cylindrical wall from floor to roof remotely through a borehole is presented. The void in the center of the cylinder can be filled with dry fill to provide a stable support column. The concrete placement system accesses the mine through an 8-in-diam borehole and builds an annulus of concrete with sloped sides. The diameter of the annulus is about 8-ft at the roof and has a slope of about 1, which results in a 20-ft-diam base. The system consists of a frame to hold and raise and lower the pipe in the borehole, a motor driven rotating mechanism to rotate the pipe and a flexible trunk on the down-hole end of the pipe which is controllably bent into a curve of approximately 18-in radius after it is in the mine void.

The second section of this report describes research involving the design and testing of a co-rotating twin screw pneumatic pipefeeder to meter fill material into a pipeline for transport of distances over 400-ft. The twin screw pipefeeder is needed to provide an economical means of blowing material distances beyond the 400-ft limit of the original pneumatic pipefeeder(1)[1] The pipefeeder has two motor driven co-rotating helical screws which feed the material to be transported from a small hopper into the mixing chamber where it is mixed with air from a compressor for

[1]Underlined numbers in parentheses refer to items in the list of references at the end of this report.

transport through the pipeline. The screws act as a seal which keeps air from escaping and allows sufficient pressure to be attained in the mixing chamber for transport of material beyond 400-ft. The feeder is sized for a 6-in-diam pipeline.

The third section of this report discusses the demonstration of two previously developed pneumatic stowing technologies to fill an abandoned mine tunnel. In October 1991, the Office of Surface Mining (OSM) investigated a subsidence complaint on the surface of an undeveloped road in Vandling, Lackawanna County, PA.. The subsidence event was the result of the collapse of a portion of the Hillside Coal and Iron Slope. The slope was originally used to transport coal from a mine to a railroad siding and is situated directly underneath several of Vandling's residential streets and underneath a state highway.

The tunnel was approximately 600-ft in length, 14-ft in width and 5 to 8-ft high. The tunnel was positioned very near to the ground surface. The walls, roof and floor of the slope are constructed of reinforced concrete. In April 1992, a video survey of the slope was conducted. The survey showed several areas of the slope contained roof spalls while other areas were deteriorated and cracked. After a thorough review of the available information, federal officials determined that the slope presented a potential hazard to the safety and welfare of the residents and vehicular and pedestrian traffic in the area.

This abandoned mine site offered a unique opportunity to demonstrate the usefulness and applicability of two recently developed subsidence abatement technologies.

CONCRETE PLACEMENT SYSTEM

The concrete point support system consists of a series of mechanical elements which allow remote placement of low slump concrete in a mine void through a borehole. The system is made up of a support frame, a rotation device, a trunk bending mechanism, a series of modified 4-in-diam pipes and couplings, a flexible trunk, and a concrete delivery nozzle. The system also requires a low slump pumpable concrete. Each of the system elements are discussed in the following sections.

Support Frame Design

The support frame serves several functions. The frame system makes it possible to install the pipe in sections as it is lowered into the borehole. It provides a means to hold the pipe and trunk in the borehole. It contains the mechanism to rotate the pipe and trunk and to bend the trunk into the desired

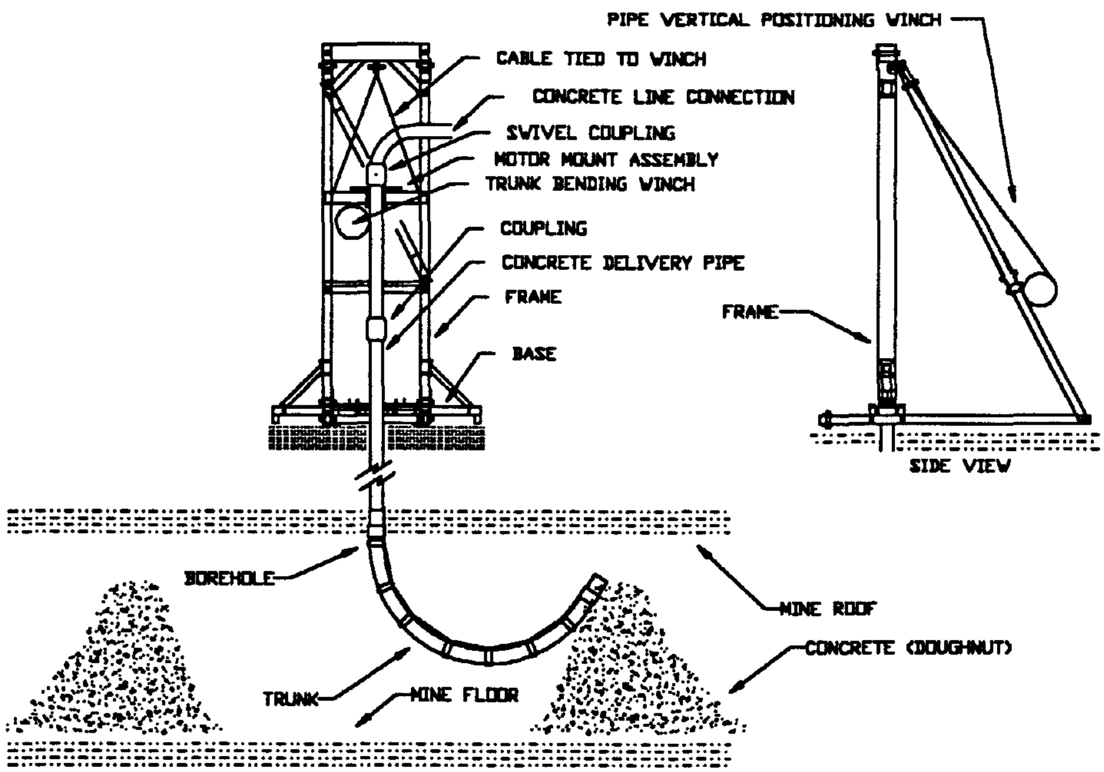

Figure 1 - Concrete placement system support frame and drive.

shape for concrete placement.

Figure 1 is a drawing of the complete system installed over a borehole. The frame was designed to be easily assembled on a job site. The various beams and other components are all separate items. This allows an efficient means of transporting the system from site to site. The beams are pinned together at the ends and to brackets on the base of the frame. The motor mount assembly fits between the vertical frame members and is winched up and down by a winch and cable system. The motor mount assembly consists of a hydraulic motor which rotates the pipe and trunk by driving through a chain and sprocket system. A pipe installation bracket (not shown in the figure), similar to the motor mount assembly without the motor and bearings, is used to install and remove the pipe and trunk in the borehole, one pipe section at a time. Plates in the base of the frame support the pipe in the borehole while a section of pipe is added or removed. The frame is sized to handle 7-ft long pipe sections, but can be easily modified to accommodate longer sections.

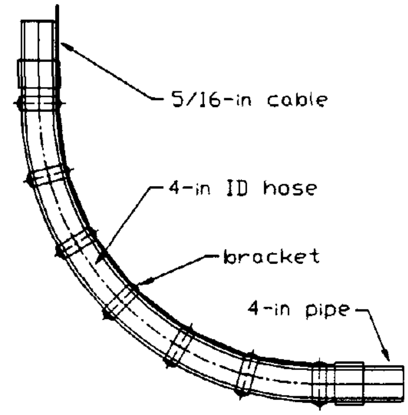

Figure 2 Trunk assembly

Pipe Installation System

The pipe hoist fixture consists of cable mounts and a centered hole just large enough for the 4-in-diam schedule 40 pipe to slip through. This allows the operator to slide the pipe into the hole and place a quick mount coupling over the end of the pipe. After the coupling has been clamped on the pipe end, the hoist is used to lift the pipe. The pipe hoist fixture is used only during pipe installation and removal and is removed from the headframe during concrete pumping.

Low profile pipe clamps

Special, low profile pipe clamps were designed for clearance in an 8 in. borehole and to carry two cables alongside the pipe sections. One cable supplies the tension for trunk bending and the other will be used in the future for the borehole camera. The coupling is designed to grip a groove turned in the outside diameter of the pipe.

Pipe and Trunk Rotation System

This rotation system holds the motor and the pipe support bushing, and has slide grooves at the ends for torque resistance. The rotation system is used during operation only and is out of the way during pipe installation. It consists of a hydraulic motor, a section of 4 in pipe welded to a chain sprocket, a bronze bushing designed to fit over the pipe and slide in a fitting on the motor mount. The pipe winch, not shown, is clamped to the pipe just underneath the motor mount. This allows the cable that supports the bent trunk at the bottom of the borehole to rotate with the pipe assembly. A swivel pipe coupling is mounted on the top of the rotating feed pipe. A pipe elbow is mounted on top of the coupling. The pipe from he concrete pumper then connects to the elbow which allows rotation of the trunk while connected to the pumper.

System Operation

At the Subsidence Abatement Investigation Laboratory (SAIL) the concrete pumper was parked at the top of the highwall near the feed conveyor location. Concrete was pumped in a pipeline provided by the concrete pumping contractor to the top of the borehole on the tower. The end of the pipeline was connected to the concrete placement system with a flexible hose. In an actual backfilling situation the pumping can be accomplished without the long pipeline in most cases since their would be access for a concrete delivery truck.

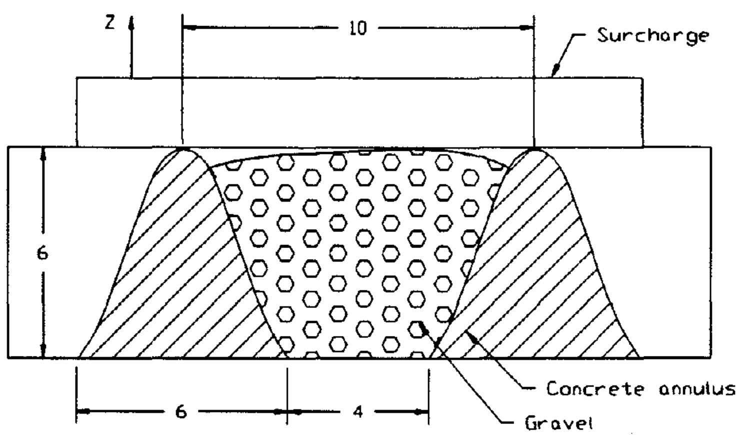

Figure 3. - Cross-section of the conical concrete support in the entry showing gravel backfill and overburden surcharge.

The trunk and pipe is installed in the borehole using the pipe hoist fixture. First, a 3 ft section of pipe is mounted on the hose. This trunk, short-pipe combination is hoisted up and the end of the trunk is placed in the borehole and lowered by the winch until the top end is near the frame base. At this point a quick release clamp is installed on two circumferential grooves on the pipe. The assembly is then lowered onto two sliding plates in the base of the frame. When there is slack in the supporting cable the pipe hoist fixture is removed from the assembly and it is attached in like manner to the next 7-ft section of pipe. This process is repeated until the trunk is in the mine void. At this point the pipe hoist fixture is removed and the motor mount assembly is installed in the support frame and attached to the top of the last section of pipe installed with a low profile clamp.

The pipe is then vertically positioned with the winch and the trunk is bent with the trunk bending winch until it is in the proper position for concrete placement. Figure 2 shows a typical position of the trunk.

Compressive Resistance of Concrete After Subsidence

Assume the conical concrete support will be supporting an effective cylinder of soil 15 ft in diameter and z = 150 ft high, see figure 3. The required compressive strength of the concrete is approximately 1000 psi. The specified concrete to be supplied by the local plant should gain at least a compressive strength f'_c = 1000 psi at 3 days. Therefore the compressive strength requirements are not a critical factor.

Backfilling Requirements

The tensile strength requirement for the concrete structure should be checked during backfilling especially at the early ages of concrete of concrete curing. This is due to the fact that the tensile strength of concrete is only 10% of its compressive strength. The conical concrete support is modeled as a thick walled cylinder and the radial and tangential stress σ_r and σ_t respectfully are given as shown by equations 1 and 2. The maximum stresses occur at $r = r_j$ and are both conservatively less than 10 psi compression and 10 psi tension for σ_r and σ_t respectively. Therefore the conical concrete supports may be backfilled as early as 1 day after the concrete has set.

$$\sigma_r=\frac{p_j r_j^2}{r_0^2-r_j^2}\left(1-\frac{r_0^2}{r^2}\right) \tag{1}$$

$$\sigma_t=\frac{p_j r_j^2}{r_0^2-r_j^2}\left(1+\frac{r_0^2}{r^2}\right) \tag{2}$$

where:

p_j = the internal pressure simulated by the active earth pressure and may be conservatively assumed as 1 psi.

r_i and r_o = inside and outside radius respectively

Concrete Support Under Water

Abandoned mines are sometimes inundated in water. The rise in water table should not affect the stability of conical concrete support except for the bearing capacity of the floor which we are assuming to be adequate. Concrete strength gain and setting characteristics will not be affected drastically as long as the concrete is placed adequately and not discharged freely into the water, but the discharge nozzle is inserted in the concrete cone.

Table 1 - Cost comparison of concrete cylinder vs. solid grouting

Radius ft	height ft	concrete volume ft^3	gravel volume ft^3	concrete/ gravel cost $	all concrete cost $
5	4	242	179	493	749
8	4	392	574	899	1720
5	5	378	197	742	025
8	5	613	669	1325	2280
5	6	545	208	1043	1341
8	6	884	746	1833	2898
5	7	742	214	1395	1702
8	7	1203	808	2423	3577
5	8	970	215	1800	2108
8	8	1572	856	3095	4318
5	9	1228	214	2259	2565
8	9	1990	892	3851	5126
5	10	1516	212	2770	3073
8	10	2457	918	4691	6003

System Economics

Table 1 shows volumes of concrete and gravel required for various radii of the concrete support. Based on approximate costs for the materials, a material cost is estimated for each of the examples for both a concrete support filled with gravel and a equal volume of grout with the same outside dimensions, but solid grout with no gravel filled void. This comparison is made to compare the relative cost of the concrete system with the practice of pumping grout down a borehole to obtain

Table 2 Concrete mix design experiments

Mix Id	Mix No 1	Mix No 2	Mix No 3	Mix No 4
Mix Type	26 Gal Mix	28 Gal Mix	Repeat Mix No 2	Mix 3 (Air & Ret)
Cement	611.6 lbs/yd^3	611.6 lbs/yd^3	611.6 lbs/yd^3	611.6 lbs/yd^3
Gravel No 67	1666 lbs/yd^3	1666 lbs/yd^3	1666 lbs/yd^3	1666 lbs/yd^3
Sand	1771 lbs/yd^3	1725.3 lbs/yd^3	1628 lbs/yd^3	1361 lbs/yd^3
Water	159.3 gal/ yd^3	175.5 gal/ yd^3	216 gal/ yd^3	235 gal/ yd^3
Water Reducer	2.8 oz/100 lbs Cement	2.8 oz/100 lbs Cement	2.8 oz/100 lbs Cement	2.8 oz/100 lbs Cement
Air Admixture	0 oz/100 lbs Cement	0 oz/100 lbs Cement	0 oz/100 lbs Cement	0.8 oz/100 lbs Cement
Retarder Admxture	0 oz/100 lbs Cement	0 oz/100 lbs Cement	0 oz/100 lbs Cement	1.4 oz/100 lbs Cement
Moisture Content Sand	2.7 %	2.7 %	2.7 %	2.7 %
Moisture Content Gravel	0.7%	0.7%	0.7%	0.7%
Initial Slump	0 in	0 in	1/2 in	3¾ in
Final Slump	-	-	-	2 ½ in
Air	-	-	-	6 1/2 %
Observations	Dry, crumbles	Dry, crumbles	-	-
Water added	6.5 lbs	3 lbs	-	-
New slump	2 in	2 in	-	-
Compressive Strength (29 day)				Cylinder A 5300 psi Cylinder B 5200 psi Cylinder C 5240 psi Strength=510 Tensile Avg Calc psi

support by building a solid grout cone. The cost is close to double for an all grout support versus a concrete support in the 4 ft mine and 1.6 times as much in a 6 ft mine.

At a 10 ft^3/min concrete pumping rate the time to complete the support ranges from 24 minutes in a 4 ft high mine to 55 minutes in a 6 ft mine and to 152 minutes in a 10 ft high mine entry.

Concrete Mix Design

Concrete mix designs were conducted at a local concrete supplier near the test facility. Several trial mixes were conducted to obtain a low slump pumpable concrete using local materials. The trial mixes are presented in Table 2. The final mix was selected based on stiffness (low slump) and pumpability at the job site. The retarding admixture was used to prevent any premature setting of the concrete in the pipe system and to allow sufficient time for cleanup in the mine roof simulator. The final specifications for the mix design are presented in Table 3.

Development and Testing of the Concrete Placement System

Four Concrete Mine Point Support Structures were constructed at SAIL. The concrete mix design was selected based on the initial mix design experiments conducted at the local concrete supplier.

Table 3 Specifications for Low Slump Pumpable Concrete

Max Coarse Aggregate	0.75 in
Max Slump	2.5 in
Min Slump	1.75 in
Min Mortar Fraction	17 cu ft/yd^3
Min Cement Factor	6.5 bags
Max air content	5 % vol
Min compressive strength	4000 psi (28 days)

Note: Chemical and mineral admixtures may be used depending on specific site and construction conditions

The test procedure for each of the concrete columns was as follows. The trunk was positioned at the desired vertical position, but was not put into the bent position until after initiation of pumping was started. The pumper first sends a wet slurry of grout or mortar through the system to wet the pump and pipe. This is followed by the concrete. As the concrete starts through the system the trunk is winched into its bent delivery position and rotation of the trunk is started. This continues until the concrete support is completed, which typically takes about one hour.

Mine Support Structure Number 1.

The first concrete point support structure was constructed in the simulated mine with the roof height set at 4-ft. The mechanical design of the system functioned as designed. The motorized pipe and trunk rotated at a rate of 1 to 3 rpm.

The concrete flow was highly dependent on the rheology and consistency of the concrete mix

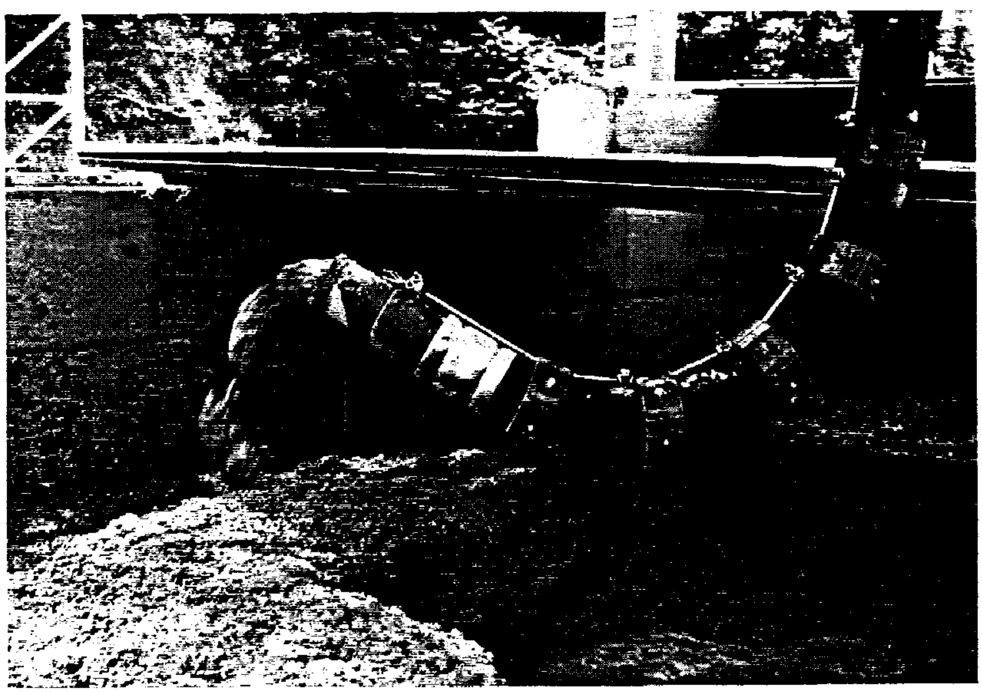

Figure 4 Concrete support number 1 being constructed.

and required minor additions of water to achieve the correct pumpability. In general the flow varied between 4 and 6.5 cfm. During this test, there was a large flow variation at the outlet of the hose. The concrete appeared to be moving faster down the pipe than it was in the horizontal line on the catwalk. It is likely that cavitation was occurring in the 35-ft drop in the vertical line. This erratic flow caused a large portion of the concrete to be thrown too far from the base of the borehole as it ejected from the hose. As a result, the base of the structure was larger in diameter than desired.

Figure 4 shows the construction of the concrete mine

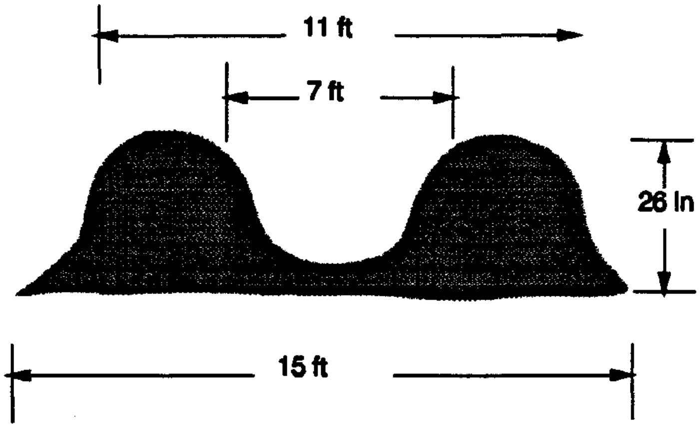

Figure 5 - Final geometric dimensions of concrete mine support structure number 1

Table 4 Concrete Mix Design For Concrete Mine Support Structure Number 1

Test 1 Truck 1	Quantity 10 yd^3
Cement	611 lbs/yd^3
Gravel No 67	1669 lbs/yd^3
Sand	1384 lbs/yd^3
Water	23.2 gal/yd^3
Water Reducer	12 oz/yd^3
Air Admixture	5 oz/yd^3
Moisture Content	Sand 4.4% Gravel 0.9%
Water Added	20 Gal/Truck
Initial Slump	0 in
Final Slump	2½ in
Air	4%
Compressive Strength (Age: 30 days)	Avg Calc Tensile Strength= 500 psi
Cylinder A	4840 psi
Cylinder B	5400 psi
Cylinder C	5370 psi
Cylinder D	5100 psi

support structure. Figure 5 shows the final geometric dimensions of mine support structure number 1. A total of 10 yd^3 of concrete was used during the construction of this point support structure.

Table 4 shows the results of the concrete mix data and the test results obtained on the fresh and hardened concrete.

Mine Support Structure Number 2

The problem of the concrete projecting too far from the hose was addressed during test 2 by the addition of a street elbow to the end of the hose. The elbow was screwed into the end fitting on the trunk, and the outlet end of the elbow was cut back so the outlet face was flush with the pipe wall. This made the elbow small enough for clearance in an 8-in diam borehole. The elbow was very effective at eliminating the spurting and keeping the concrete confined to the area below the outlet since

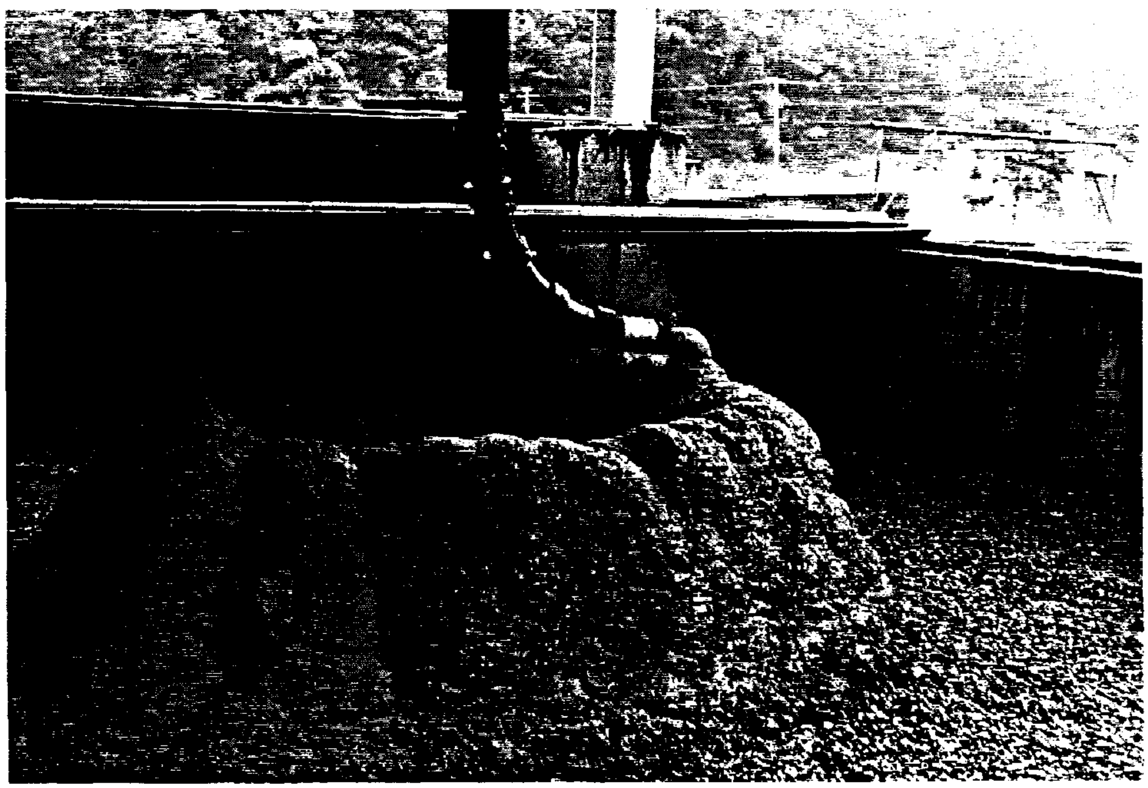

Figure 6 Concrete support number 2 being constructed.

the elbow was pointing directly into the ground. There were two major problems with the elbow. First, when the concrete was too stiff and the flexible hose was bent at a 90 degree radius, a plug

Table 5 Concrete Mix Design For Concrete Mine Support Structure Number 2

Test Date 29-Jun-93		
	Truck 1 Qty 7.5 yd^3	Truck 2 Qty 7.5 yd^3
Cement	611 lbs/yd^3	611 lbs/yd^3
Gravel No 67	1671 lbs/yd^3	1671 lbs/yd^3
Sand	1390 lbs/yd^3	1390 lbs/yd^3
Water	22.4 gal/yd^3	22.4 gal/yd^3
Water Reducer	12 oz/yd^3	12 oz/yd^3
Air Admixture	0 oz/yd^3	0 oz/yd^3
Retarder Admixture	12 oz/yd^3	12 oz/yd^3
Moisture Content	Sand 4.8% Gravel 1.0%	Sand 4.8% Gravel 1.0%
Water Added	9 Gal/Truck	7 Gal/Truck
Initial Slump	0 in	1/2 in
Final Slump	2 in	2 in
Air	2.50%	3.00%
Compressive Strength	(Age: 30 days)	(Age: 29 days)
Cylinder A	6370 psi	
Cylinder B		6510 psi
Avg Calc Tensile Strength		
Cylinder A	560 psi	
Cylinder B		565 psi

would form at the elbow and prevent flow. This problem was especially pronounced when the slump was less than 2 inches. With the restricter elbow at the end of the flexible hose, there was difficulty in making contact with the mine roof. This was due to the elbow scraping off the top layer of the concrete structure. Second, when attempting to seal the concrete to the roof of the simulated mine entry, the elbow interfered with the seal and had to be pushed through the concrete as the pipe turned.

The resulting torque was too much for trunk and it kinked at the pipe end. The elbow was abandoned for the remainder of the tests due to the plugging problem. The elbow restrictor did eliminate spraying of the concrete by providing enough resistance in the system to essentially eliminate the cavitation problem in the vertical pipe.

The simulated mine roof was 4-ft high during this experiment. Figure 6 shows the construction process of mine support structure number 2. Table 5 shows the results of the concrete mix design data and the test results obtained on the fresh and hardened concrete.

Table 6 Concrete Mix Design For Concrete Mine Support Concrete Structure Number 3

Test 3	Truck 1 Quantity 7.5 yd^3	Truck 2 Quantity 7 .5 yd^3
Cement	611 lbs/yd^3	611 lbs/yd^3
Gravel No 67	1671 lbs/yd^3	1671 lbs/yd^3
Sand	1390 lbs/yd^3	1390 lbs/yd^3
Water	22.4 gal/yd^3	22.4 gal/yd^3
Water Reducer	12 oz/yd^3	12 oz/yd^3
Air Admixture	0 oz/yd^3	0 oz/yd^3
Retarder Admixture	12 oz/yd^3	12 oz/yd^3
Moisture Content Sand 4.8% Gravel 1.0%	Sand 4.8% Gravel 1.0%	Sand 4.8% Gravel 1.0%
Water Added	7 Gal/Truck	2 Gal/Truck
Initial Slump	0 in	N/A*
Final Slump	2 in	3 in
Air	2.50%	3.00%
Compressive Strength	(Age: 30 days)	(Age: 29 days)
Cylinder A	6370 psi	
Cylinder B		5220 psi
Avg Calc Tensile Strength	560 psi	505 psi

*Operator added water prior to initial slump

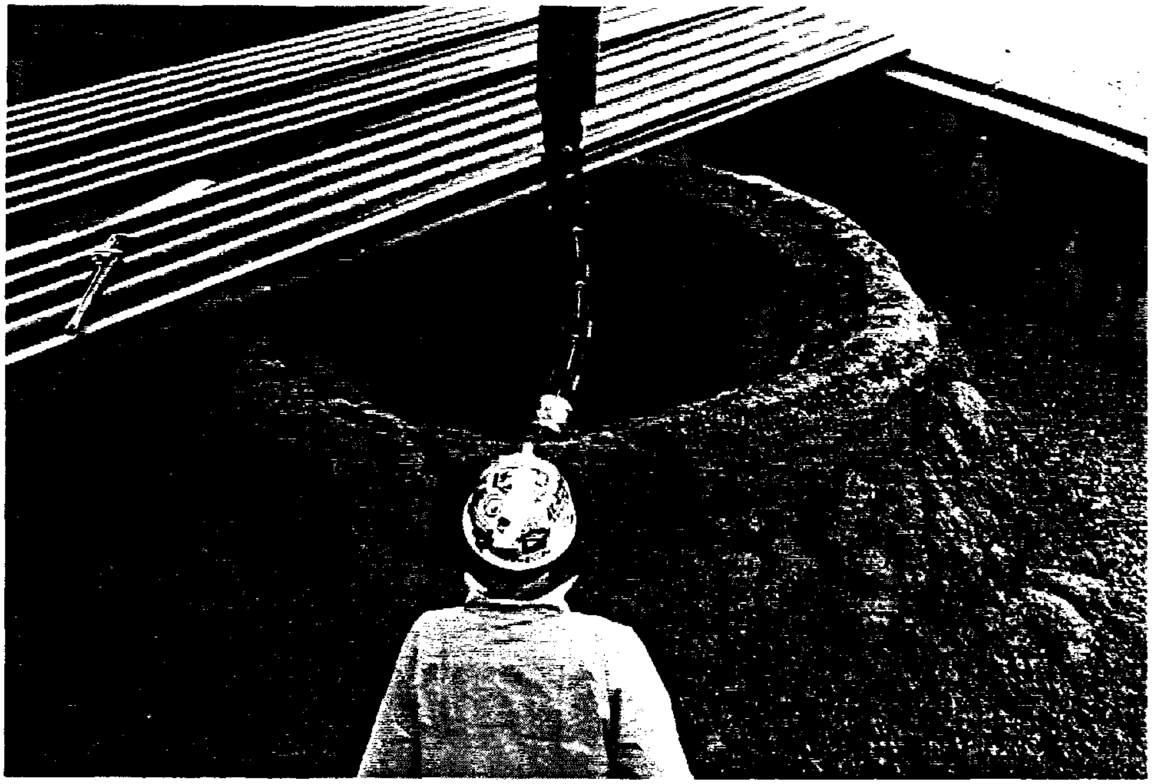

Figure 7 - Concrete support number 3 being constructed.

Mine Support Structure Number 3

The goal of minimizing concrete use while achieving roof contact was addressed in the third test by removing the 6-in long extension from the end of the trunk. In addition, the trunk was bent only slightly at the beginning of the pour. This kept the hose pointing down toward the ground so the spurts of high flow did not increase the base diameter. As the pour proceeded, the hose was bent further and the pipe was raised. This created a concrete structure that was satisfactory and neared the 4-ft high roof. A complete seal could not be made however

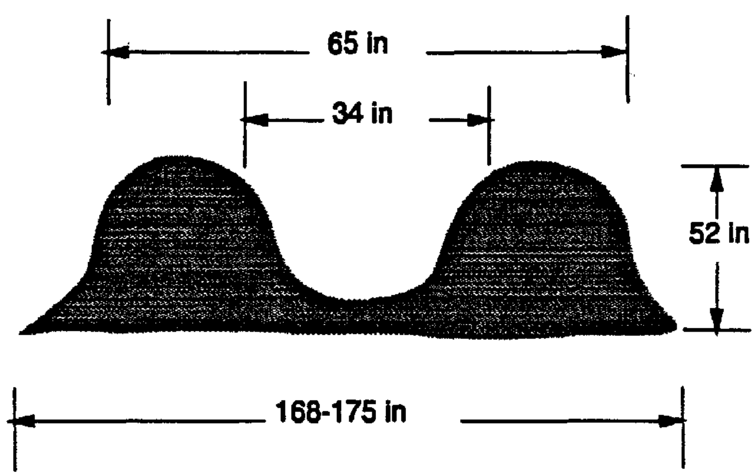

Figure 8 - Final geometric dimensions for concrete structure no 3.

since the trunk end would need to be buried in the top of the concrete in order to complete the seal. The trunk did not have the torsional resistance to resist the drag of pushing through the concrete.

Table 6 shows the results of the concrete mix design data and the test results obtained on the fresh and hardened concrete. Figure 7 show the construction phase of mine support structure no 3. The final approximate geometric dimensions of the concrete structure are shown in figure 8. A total of 15 yd^3 of concrete was used during this construction experiment.

Mine Support Structure Number 4

The simulated mine roof was placed at the 6-ft height. A reducing nozzle was used during this test to help eliminate the spurting problem encountered in tests 1 and 3. This was done to generate enough backpressure to stop the apparent cavitation in the down pipe. The reducer nozzle was 6-in long with an outlet diameter of 3.5-in. It was mounted at the trunk outlet. Observation of the outlet flow during this test indicated that the diameter reduction was not great enough to effect a change in the flow pattern in the concrete.

In an effort to make a small diameter concrete structure the intermediate cable clamps on the outlet hose were readjusted to create a tighter bend in the

Table 7 Concrete data and test results for test 4

Test 4	Truck 1 Quantity 10 yds	Truck 2 Quantity 10 yds
Cement	611 lbs/yd^3	611 lbs/yd^3
Gravel No 67	1672 lbs/yd^3	1672 lbs/yd^3
Sand	1392 lbs/yd^3	1392 lbs/yd^3
Water	21.9 gal/yd^3	21.9 gal/yd^3
Water Reducer	12 oz/yd^3	12 oz/yd^3
Air Admixture	0 oz/yd^3	0 oz/yd^3
Retarder Admixture	12 oz/yd^3	12 oz/yd^3
Moisture Content	Sand 4.8% Gravel 1.0%	Sand 4.8% Gravel 1.0%
Water Added	6 Gal/Truck	4 Gal/Truck
Initial Slump	1 in	1 in
Final Slump	2 in	3 in
Air	2.50%	3.00%
Compressive Strength	(Age: 29 days)	
Cylinder A	6670 psi	
Cylinder B	6370 psi	
Cylinder F	5920 psi	
Cylinder G	4950 psi	
Avg Calc Tensile Strength	565 psi	520 psi
Compressive Strength (Age: 29 days) Samples Obtained From the Formed Concrete Mine Support Structure		
Cylinder C	5060 psi	
Cylinder D	4950 psi	
Cylinder E	5590 psi	

trunk. This resulted in the trunk bending closer to the outlet, decreasing the cylinder diameter. This was effective in fabricating a small diameter concrete structure. In order to make roof contact the hose had to be bent further to bring the nozzle close to the roof. When the nozzle was bent close to the roof the hose kinked. The intermediate cable clamps on the outlet hose were replaced and the kinking problem was eliminated. The diameter of the concrete structure was small which allowed the center to fill with concrete due to its angle of repose. This prevented bending the trunk enough for the nozzle to make roof contact. A total of 20 yd^3 of concrete were used during this construction experiment. Table 7 shows the results of the concrete mix design data and the test results obtained on the fresh and hardened concrete.

Recommended Concrete Design

Table 8 is a list of recommendations for low slump pumpable concrete. This mix will provide the proper characteristics for concrete to construct point support structures in underground mine through boreholes.

Equipment operation

Equipment operation includes headframe setup, pipe installation, equipment operation, pipe removal and headframe takedown. Setup time for the headframe was 4 hours with 3 persons. With some experience and the possible change to taper pins for the joints, the setup time could be reduced to 4 hours with 2 persons. The pipe installation was safe, straightforward, and easily accomplished with two persons in less than 1 hour. Takedown time was less than the setup.

Table 8 Specifications for Low Slump Pumpable Concrete

Max Coarse Aggregate	0.75 in
Max Slump	2.5 in
Min Slump	1.75 in
Min Mortar Fraction	17 cu ft/yd^3
Min Cement Factor	6.5 bags
Max air content	5 % vol
Min compressive strength	4000 psi (28 days)

Note: Chemical and mineral admixtures may be used depending on specific site and construction conditions

This translates into a turnaround of two days per doughnut in the field with a new setup required for each borehole location. With some experience and close proximity between boreholes it may be possible to fabricate one or two concrete support structures per day.

The rotation of the pipeline was easily varied down to 3 rpm. The pipeline couplings prevented twisting of the line during rotation and held the cable as designed. The pipe raising and lowering mechanism provided the control of pipe height needed to make the cylinders. The pipeline swivel functioned well and did not become a source of leakage. The trunk bending system and trunk tension cable provided the control over trunk bend radius that was required. It was necessary to use the intermediate cable clamps on the trunk to control the bend radius. It may be necessary to construct a metal enclosure capable of bending so that the hose is not required to take the torque of the nozzle dragging through the concrete at the roof line seal. This can be easily done through a modification of the existing hose clamps.

Summary

The project demonstrated that an 8-ft diam concrete cylinder can be made through a borehole of 8-in diam. Improvements to the trunk design are needed to insure that it can be dragged through the concrete during the process. This is easily accomplished by surrounding the trunk hose with hinged sleeves. An improved concrete nozzle on the end of the trunk is needed to increase resistance in the delivery pipe to eliminate cavitation. A shield is required to protect the cable used for control; of the trunk bending. These are all technically possible change and should be considered for future work.

CO-ROTATING TWIN SCREW PIPEFEEDER

When transport distances for pneumatic backfilling are over 400 ft the only device that is capable of performing is the rotary air-lock feeder. This machine can be expensive and is often subject to severe wear problems. A low wear co-rotating twin screw pneumatic pipefeeder has been designed and demonstrated which provides an alternative to the rotary air-lock feeder for blowing backfill material distances over 400-ft. The design and results of the demonstration tests are presented in this

Screw feeder Design

The co-rotating twin screw feeder design is shown in figure 9. The design concept of the twin-screw feeder is to allow a higher line pressure than is possible with the pipefeeder. This will result in longer transport distances in a pipeline. The screws allow a means of metering the material to be transported into the line while minimizing the leakage of air at the hopper. This design has the two screws mounted in a vertical position. The material is transported downward from the hopper by the screws to a mixing zone at the outlet of the screws. The screws are overlapped so that the outside diameter of one screw just clears the shaft of the other screw. The screws both rotate in the same direction and are geared together by a pair of sprockets and a chain. Each screw shaft is driven by a hydraulic motor. The motors normally carry the torque of just one screw and the sprocket and chain only serve to keep the two screws in synchronization. If one screw lags due to a high load the other motor picks up the load through the chain drive. Normally the chain carries no load. The two screws are mounted in a housing which fits around the outside diameter of the two screws. The clearance between the screw outside diameter and the inside diameter of the housing is about 0.010-in. The two screws are mounted in ball-bearings on the lower end and bushings on the top. The top bushings are mounted to the hopper by struts. The vertical orientation of the screws minimizes the possibility of having partially filled screws as long as material is kept in the hopper. This is important since partially filled screws will allow air to leak by the screws and limit the ability of the feeder to hold pressure. This in turn limits the distance that gravel can be transported.

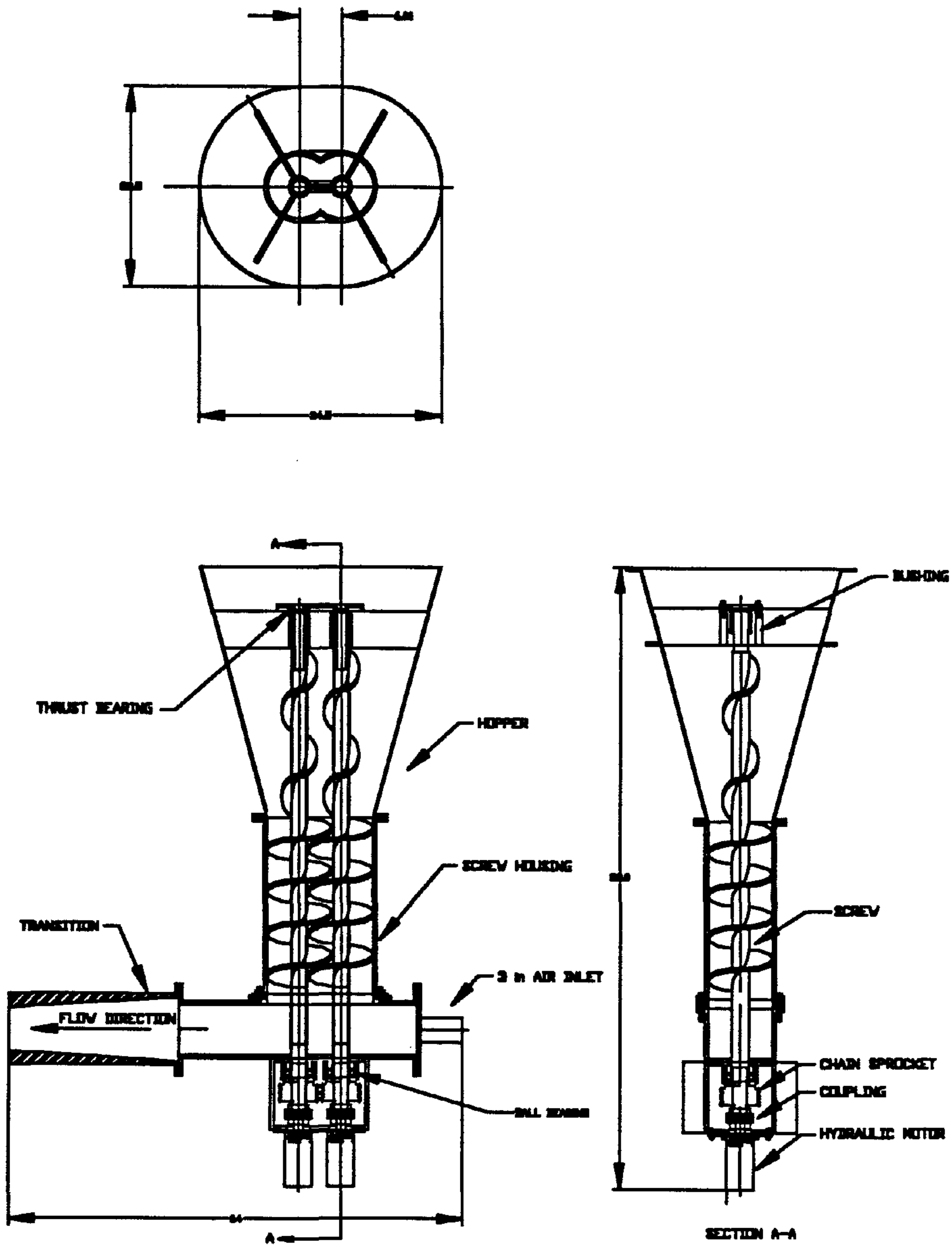

Figure 9 - Co-rotating twin screw pipefeeder

The material enters a mixing zone after it drops out of the screws where it is intercepted by the flow of air from the compressors. The entrained material passes through the transition section which changes the cross section of the flow path from rectangular to the 6-in diam pipe

Screw feeder Set-Up

The twin co-rotating screw pipefeeder was set up at SAIL as shown in figure 10. Gravel from a stock pile was loaded into the hopper by a front-end loader. The material passed through a gate in

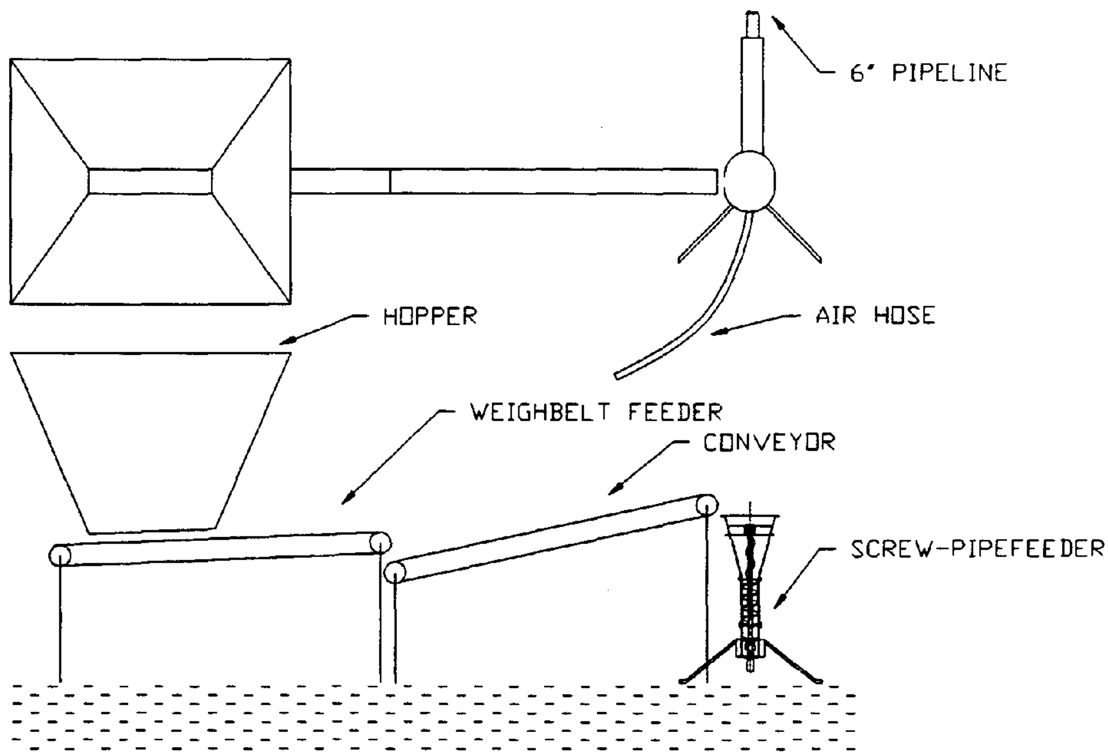

Figure 10 Screw-Pipefeeder Test Set-Up

the bottom of the hopper onto a weigh-belt feeder. The material then was conveyed by the weigh-belt feeder to the belt conveyor which delivered the metered gravel into the screw-pipefeeder hopper. A hydraulic power unit supplied hydraulic power to the two hydraulic motors on the bottom of the pipefeeder. The two hydraulic motors were synchronized to each other by a chain and sprocket system. Air from three air compressors connected in parallel provided up to 2800 cfm.

Six inch diameter poly vinyl chloride (PVC) pipe was set up for a total of 710 ft on the surface of the facility. The pipe sloped gradually upward about 30 ft from the feeder to the end of the pipe. The end of the pipe was inside the portal of the limestone mine.

Test Procedure

The feeder was first operated with 140 ft of pipe. Air and the screw power were turned on prior to starting the material flow. The material flow rate was set to maintain a level of gravel near the top of the feeder hopper, which assured that the screw flights were full of material. A full feeder hopper is essential for optimum sealing of the pneumatic transport air. The line was extended in approximately 200 ft increments to 710 ft. At each line length the same test procedures were used.

Each test was operated until a steady state condition was achieved. Measurements taken for each test were air flow rate, material flow rate and screw rotational speed. The average time for each test was 15 min.

Screw Pipefeeder-Feeder Test Results

The test results are presented in figures 11 and 12. The material flow rates are shown as a function of pipe length and line pressure. Measurement of pressure in the line was not possible due to a malfunction of the instrumentation. Therefore the pressure used in the analysis is calculated based on pneumatic transport parameters and is representative of past tests with the pipefeeder where the instrumentation was operable. Based on the calculations of line pressure, the maximum pressure achievable with this system was 7.05 psi as compared to 4 psi for the pipefeeder without screw feed. Figure 11 presents the maximum pipe length as a function of material flow rate for this system when using 2825 cfm air. Figure 12 shows the test data as a function of material flow rate and the required system pressure at 2825 cfm air flow rate.

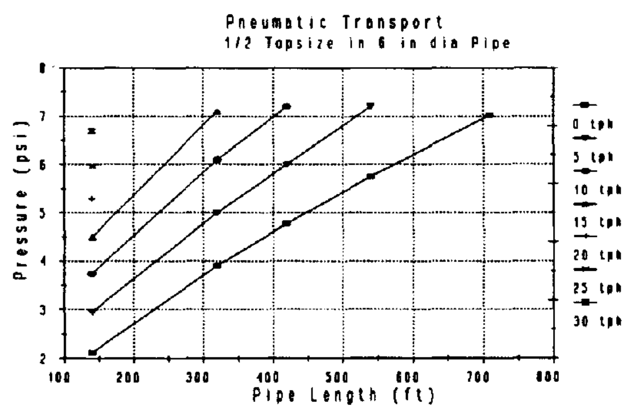

Figure 11 - Pipeline pressure as a function of material flow rate and line length

The initial two tests were conducted using 2000 cfm air flow with a 140 ft long pipeline. The screw-pipefeeder, operating at 60 rpm was limited to 16 tph material flow. When the speed was doubled to 120 rpm the material flow rate increased to 25 tph. No further increase in screw speed was possible. The screw-pipefeeder was limiting the material flow rate to 25 tph. Projecting these data, using pneumatic transport headloss equations gives a possible improved capability as shown in figure 13. The graph shows that it may be possible to transport 25 tph, 400 ft, in a 6 in diameter pipeline or 10 tph, 760 ft.

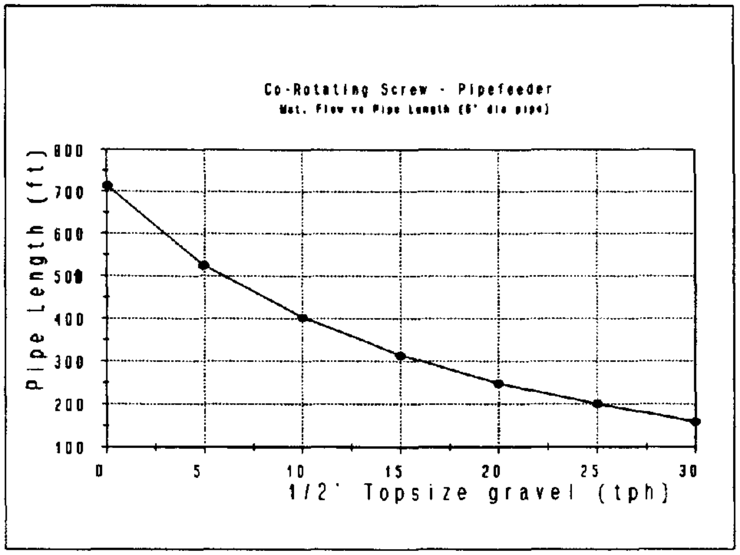

Figure 12 - Material flow rate as a function of pipe length

<u>Screw-Pipefeeder Conclusions</u>

The screw-pipefeeder showed improvement over the original pipefeeder which did not use screws to feed the material. The screw-pipefeeder allows higher pipeline pressures which equate to longer transport distances.

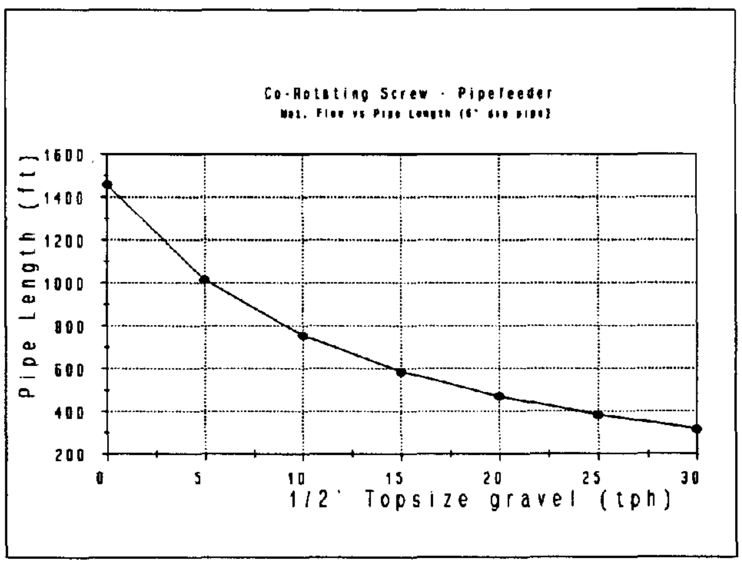

Figure 13 - Projected system performance at 2000 cfm air flow

<u>Summary</u>

The tests have shown

that the air flow rate has a large effect on the system performance. This variable needs to be further evaluated to optimize the system. The co-rotating screws were set at assembly so that the flights of one screw were midway between the flights of the second screw. This minimizes the tendency of large rocks jamming at the expense of poorer sealing. If the screws were adjusted so that the flights of one just cleared the flights of the second an improvement in the ability to seal air pressure should be achieved. The screw-pipefeeder reliably fed gravel without choking and no jamming of material occurred. Tests were not conducted long enough to evaluate if a wear problem may exist with this design.

PNEUMATIC STOWING DEMONSTRATION

A field demonstration of two previously developed pneumatic backfilling devices was conducted in the fall of 1992. These two devices, known as the High-Efficiency Ejector and the Pneumatic Pipefeeder, were developed under the Bureau's Abandoned Mine Land Research Program.

An abandoned coal mine haulage tunnel was identified by the Office of Surface Mining (OSM) as a potential danger to the community proximal to the tunnel. OSM and the Bureau entered into a cooperative agreement to both remediate the dangerous conditions existing at the site, and to provide an opportunity to demonstrate the two pneumatic backfilling devices developed under the Bureau's research program.

The Bureau of Mines, and OSM cooperated to ensure that this demonstration project was a success. Included in this report are the descriptions of both the pipefeeder and the ejector, and the operating theory behind these two devices. A description of the field demonstration is given, including overall project objectives, site conditions, remediation plan, and sequence of operations. Finally, the results of the demonstration are provided, including a limited cost analysis of the demonstration.

The results obtained from this field demonstration of the two pneumatic devices demonstrates their usefulness in pneumatically backfilling underground mines and entries. The pipefeeder and the ejector both proved to perform as expected, and lessons learned from this field demonstration can be applied to future applications to further improve their overall capabilities. The success of these two devices should be of interest to the AML community in particular and the mining industry in general

Introduction

In October 1991, personnel from the OSM in Wilkes-Barre, PA investigated a subsidence complaint on the surface of an undeveloped road in Vandling, Lackawanna County, PA. The subsidence event was the result of the collapse of a portion of an abandoned coal mine haulage tunnel, known as the Hillside Coal and Iron Slope. The tunnel is approximately 600 ft long, 14 ft in width, and 5 to 8 ft high. The tunnel is straight throughout its entire length, and dips such that the north end

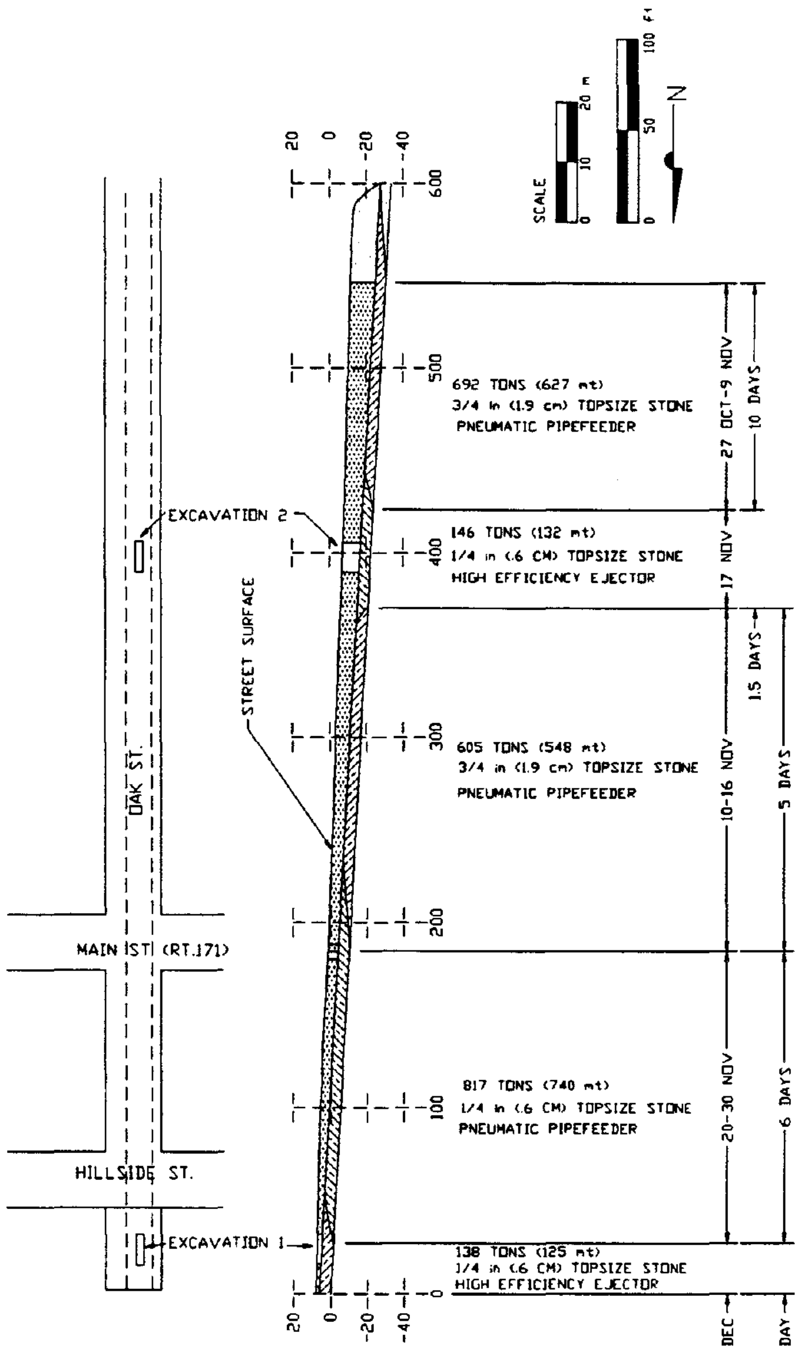

Figure 14. - Map View of the Tunnel

of the tunnel is approximately 40 ft higher than the south end. The ribs, roof and floor of the tunnel are constructed of reinforced concrete. The tunnel was originally used to transport coal from a mine to a railroad siding and is situated directly underneath several of Vandling's residential streets and underneath Pennsylvania State Highway 171. Figure 14 is a plan view of the tunnel showing its location relative to the surface roads.

In April 1992, OSM officials conducted a structural assessment of the tunnel. This survey indicated that several areas of the tunnel had significant roof and wall deterioration. OSM officials subsequently determined that the tunnel presented a potential hazard to the safety and welfare of the residents and vehicular and pedestrian traffic in the area. OSM officials contacted the U.S. Bureau of Mines to provide the Bureau with a field site in which to demonstrate two recently-developed pneumatic backfilling devices. This abandoned mine site offered a unique opportunity to demonstrate the usefulness and applicability of the two subsidence abatement technologies for actual abandoned mine remediation. The two pneumatic devices were developed for stowing backfill material into abandoned mines to provide roof support; the development of these two devices was funded through the Bureau's Abandoned Mine Land Research Program. The two devices are known as the Pneumatic Pipefeeder and the High-Efficiency Ejector.

High-Efficiency Ejector

Prior to the development of the ejector, conventional pneumatic stowing through boreholes into underground mines was conducted by blowing fill material through a vertical pipe placed in a borehole.

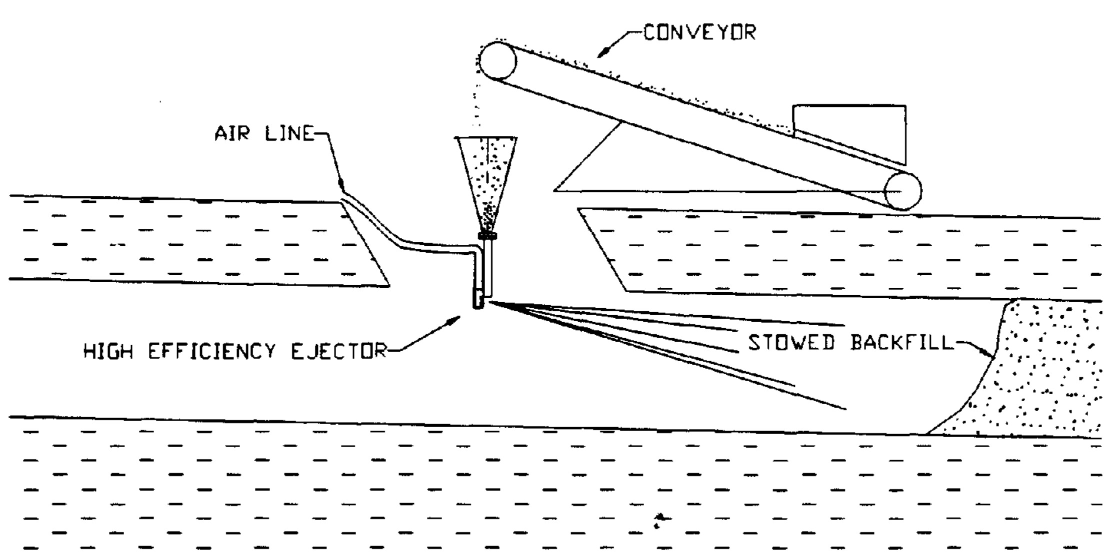

Figure 15. - Ejector installed in opening number 2

The system typically used to blow the fill is a pocket-feeder device and a low-pressure blower. Pocket feeders are typically subject to extreme wear. Backfill material is blown straight down the pipe and quickly builds a conical pile around the borehole which extends only a short distance radially from the borehole before choking off the flow. Attempts have been made to orient the flow of backfill by means of elbows or elaborate devices that unfold after they enter the mine void. These devices can work, but, because they mechanically redirect the high velocity gravel horizontally as it enters the mine void, they are subjected to wear which limits there usefulness. These problems have been solved with the High-Efficiency Ejector.

The ejector solves the abrasion problems associated with other remote pneumatic stowing systems. This system is also capable of producing cemented fills. The ejector is capable of moving the fill material over 70 ft from the injection point and filling a mine opening to the roof level for a distance of about 30 ft in all directions from the borehole.(2) This system operates by the transfer of momentum from a supersonic air stream to the fill material. The momentum transfer results in a horizontal stream of fill material and air at a velocity of about 100 ft/s. The ejector fits on the bottom end of a 3-in diameter vertical pipe. Fill material is fed into the ejector at a controlled rate through a conical hopper mounted on the top end of the same 3-in diameter pipe. High pressure air used to operate the ejector is fed through a second 3-in diameter pipe. Where necessary, cement can be added to the gravel and water added to the air supply to create a cemented fill. A drawing of the ejector system set up at the site is shown in Figure 15. The ejector is designed to fit in an 8-in diameter cased borehole.

To improve the mixing of the fill material with the airstream, the feed pipe diameter is made as small as possible so that the ratio of the width of the nozzle array to the drop feed pipe diameter approaches one. The small feed pipe concentrates the material so that the entire backfill supply passes in front of the nozzles. Figure 16 shows the 5-nozzle configuration of the ejector. The staggered arrangement of the five nozzles allows closer packing of the nozzles and avoids plugging of the feed pipe by maintaining a clear path for the material to exit the pipe. Because the nozzles cover virtually

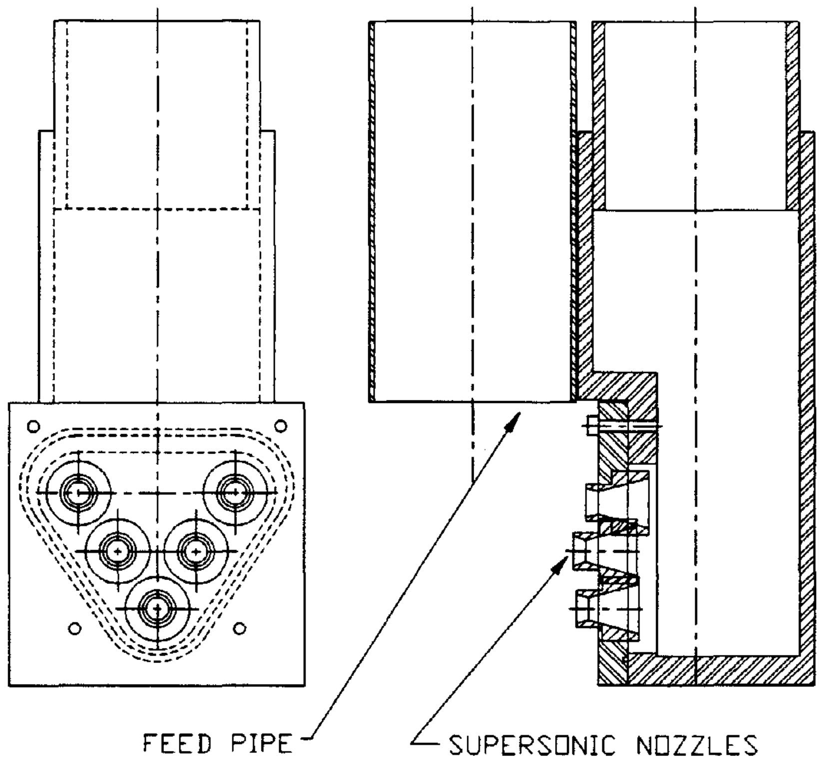

Figure 16. - Five nozzle design ejector drawing

all of the path of the falling material, none of the material can drop straight through to the mine floor.

The velocity of the airstream is also of critical importance. Each nozzle is dimensioned to the available air flow and pressure to ensure that the airstream attains a supersonic, 1,600 ft/s velocity. At this velocity the airstream has sufficient momentum to drive the fill material a great distance from the ejector.

Supersonic velocity is achieved when a gas is expanded through a properly designed converging/diverging (CD) nozzle. For a pressure ratio across a CD nozzle greater than approximately 1.89 for air at standard atmospheric conditions, the exit flow will be supersonic. The following relationships for velocities and areas as a function of pressure can be derived from thermodynamics.

When the slow moving fill material enters the high-velocity airstream generated by the nozzles, it is accelerated and the mixture of air and solids reaches a velocity of approximately 100 to 200 ft/s depending on the solids flow rate. This velocity can be calculated by the following momentum exchange equations.

$$m_s u_s + m_a u_a = m_s v_s + m_a v_a \tag{3}$$

where:

m_s = mass flow rate of solids (kg/s) (lb/s) ,

m_a = mass flow rate of air (kg/s) (lb/s),

u_s = initial velocity of solids (m/s) (ft/s) ,

u_a = initial velocity of air before mixing m/s) (ft/s),

v_s = final velocity of solids after mixing m/s) (ft/s),

v_a = final velocity of air after mixing m/s) (ft/s)

If we assume 100 pct mixing and momentum transfer then:

$$v_s = v_a \tag{4}$$

But:

Testing has shown that $v_s \approx 0.6 \text{ to } 0.7 v_a$ for up to -¾ material

Solving for v_s when $u_s = 0$ we get:

$$v_s = \frac{m_a u_a}{m_s + 1.67 \times m_a} \tag{5}$$

Using this equation, it follows that a feed rate of 25 short tons (st) per hour and an air flow of 2000 ft^3/min compressed to 100 psig from standard atmospheric conditions and expanded through a CD nozzle to a velocity of 1,600 ft/s has sufficient momentum to accelerate the fill material to a velocity of over 100 ft/s. Assuming no air drag on the fill material, this velocity relates to a projection distance of 100 ft from the ejector in a 6-ft high mine entry . As mentioned earlier, actual projection distances about 70 ft due to air drag.

Pneumatic Pipefeeder

The pipefeeder design concept uses an eductor as the means for mixing the solids with the air and for accelerating the solids to transport velocity. (Burnett, 1990).(3) An eductor works on the theory of momentum exchange, where high velocity air impacts low velocity solids, resulting in a velocity of the air/solids mix which is sufficient to pneumatically transport the solids through a pipeline. The Pneumatic Pipefeeder setup used at the Vandling site is shown in figure 17. Backfill material is loaded into a hopper by means of a belt conveyor. The belt conveyor meters the material to the pipefeeder and drops it into the hopper at a controlled rate. The material falls through the pipefeeder and is intercepted by the air flow of a supersonic nozzle. Air is supplied to the nozzle at 100 psig. The air passes through the supersonic converging/diverging nozzle and expands to pipeline pressure. During this expansion the air velocity is accelerated to approximately 1,600 ft/s. When the slow moving fill material passes by the high-velocity air stream generated by a single nozzle it is accelerated and the air and solids reach a velocity of approximately 100 ft/s depending on the solids flow rate. The pipefeeder can supply enough motive power to move the material up to 400 ft through a 6-in diameter pipe.

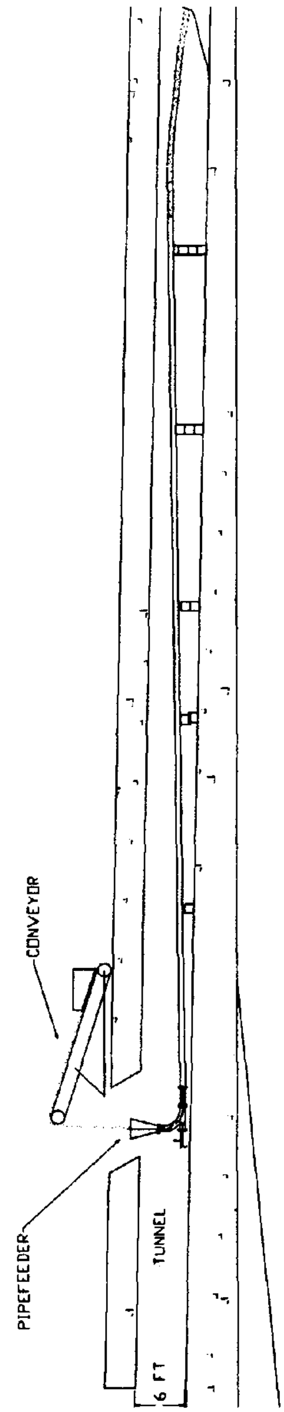

Figure 17. - Pipefeeder setup

Belt Conveyor

A belt conveyor metered the material to both the Pneumatic Pipefeeder and the ejector and dropped it into the respective hoppers at a controlled rate. The belt conveyor, powered by a small air cooled diesel engine, had a large hopper capable of holding several yards of gravel which allowed the front end loader with a 1 yd^3 bucket to stay ahead of the stowing process.

Air Compressors

Two diesel-powered air compressors were used to provide compressed air for the two pneumatic backfilling devices. A 1,200 ft^3/min unit and a 750 ft^3/min unit were plumbed in parallel to provide a total flow rate of 1950 ft^3/min at 100 psig. The conduit used to connect the compressors to the pneumatic devices was a 3-in inside diameter hose.

Pipe

Three different types of 6-in diam pipe were used on the project.

- High density polyethylene
- Schedule 10 steel pipe
- Schedule 40 steel pipe

Backfill

Two types of backfill material were used on this project. Initially, ¾-in topsize gravel was used. During the project it was decided to change to ¼-in topsize gravel to determine if there was any advantage using the smaller material.

Backfilling Sequence

Both pneumatic stowing methods were used to fill the tunnel at Vandling in several stages. Figure 15 shows the location and timing of the various phases of the project in the cross section view of the tunnel. Three pipefeeder setups and two ejector setups were used. Access was gained to the tunnel through two excavation sites.

The pipefeeder was used to fill the majority of the tunnel. The first opening (Opening 1) in the street was made to access the tunnel about 200 ft back from the lower end of the tunnel. Opening 2, shown in figures 14 and 15, was located at the upper end of the tunnel. The openings were made with a backhoe, and cutting torches were used to cut through 4 layers of 1-in diameter rebar embedded in the concrete roof. This allowed access to the tunnel and kept the maximum pipefeeder pipeline length to less than 200 ft.

The pipefeeder was first used to fill the tunnel from the lower end to within 30 ft of Opening 1 with 692 short tons (st)) of ¾-in topsize stone. Pipe was set up in the tunnel on wooden supports. Periodically the pipe and supports were moved to direct the fill left, center or right to achieve complete filling across the face. At the start of the job high density polyethylene pipe was used for the pipefeeder pipeline but proved to wear rapidly. It was tried because of its proven success in slurry transport and because of its light weight. After about 600 st were blown through the high-density polyethylene pipe, it was abandoned in favor of steel pipe. Gravel was blown into the tunnel starting at the pinched off lower end of the tunnel. As the tunnel filled, the pipe was shortened one section at a time until the tunnel was filled to within 30 ft of Opening 1. The pipefeeder was then turned 180°, and 105 ft of steel pipe was set going up towards the upper end of the tunnel. Fill material was again stowed until it came to within 30 ft of the opening.

After the pipefeeder was removed, the ejector was lowered through the excavation and into the

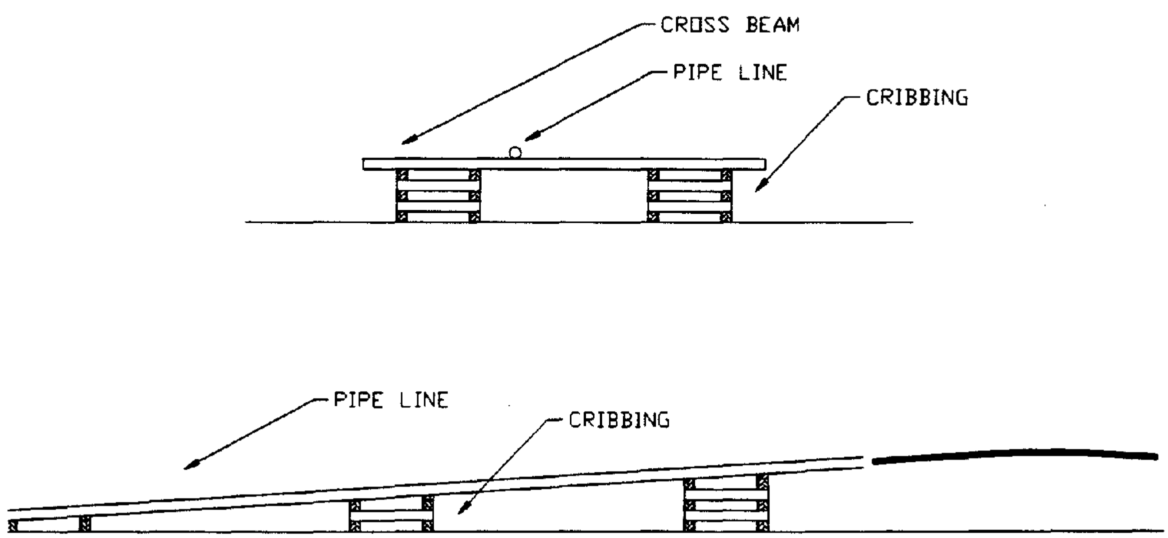

Figure 18. Method of supporting the Pneumatic Pipefeeder pipeline using wood blocks.

tunnel as shown in figure 15 and used to backfill the remaining 60 ft of the opening. When the ejector was first used, the gravel type was switched from ¾-in topsize to ¼-in topsize stone. This smaller stone was used for the remainder of the project. The ejector stowed 146 st into the 60-ft opening remaining at Opening 1 after removal of the pipefeeder. The pipefeeder was then set up at opening 2.

It was again lowered into the tunnel and supported with an improved method of pipe support. A series of wood supports, shown in figure 18, were installed so that the pipe could slide from rib to rib without rebuilding the wood supports each time a pipeline orientation move was necessary.

When the gravel pile reached within 30 ft of the pipefeeder, it was removed from the tunnel and the ejector was used to finish the backfilling.

Results

The pneumatic stowing process filled 100 pct of the tunnel void. To achieve this, it was necessary to insure that the material was aimed near the roof. It typically took three directional positions of the pipeline when using the Pneumatic Pipefeeder and two positions of the High-Efficiency Ejector to get complete filling across the 14 ft wide tunnel. When a section of pipeline was removed as the backfill pile advanced in the tunnel opening, the new pipeline length was redirected at the backfill pile three times. The packed material density was not measured, but it was observed that the material was tightly packed and could be walked on without causing the face of the pile to sluff off as happens on a pile of freshly dumped gravel. The stowing rate during operation was typically 25 st/h with a peak day of 35 st/h .

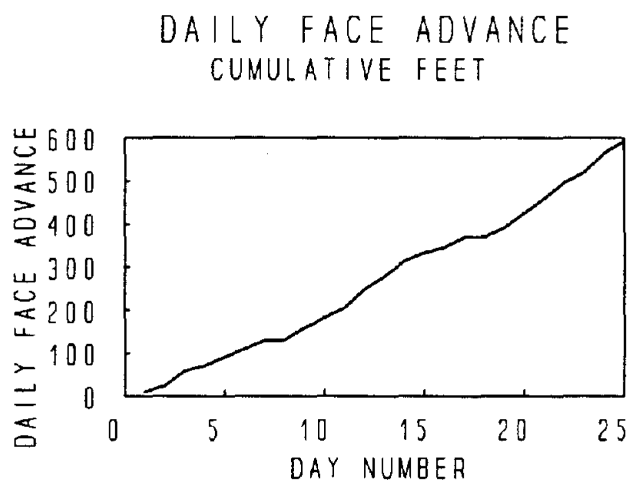

Figure 19. Face advance per day

A water flow of 5 gal/min on the floor of the tunnel running through the tunnel prior to the backfilling project appeared to be undisturbed by the presence of the stowed gravel.

The steel pipe was used for the majority or the project. The Schedule 10 steel pipe was cut into 10 ft sections, which resulted in 93 lb sections of pipe. The schedule 40 pipe was cut into 7 ft sections which resulted in 132 lb pipe sections. Hugger couplings were used for the high-density polyethylene pipe and, in some cases, for the steel pipe as well. Grooved couplings were used on the steel pipe with grooved ends. The heavier Schedule 40 pipe was used on the long runs of the pipefeeder pipeline, which minimized the amount of handling of the heavy pipe.

Wear of the steel pipe was not excessive on this project, but is a factor in the cost of the job. The selection of the pipe for future jobs will need to consider the ease of setting up the pipe, the total tonnage to be stowed and the number of times the pipe must be moved during the stowing project.

Although the pipe was ordered with grooves for use with the grooved couplings, it proved to be quicker to use the hugger-type couplings which do not require special pipe grooves, but depend rather on friction of the tightly clamped coupling to hold the sections of pipe together. The grooves on the schedule 10 pipe grooves rapidly wore through, which ultimately turned the grooved-end pipes into plain-end pipes. The pipe was set up in the tunnel and kept straight by careful building of wood supports made of 6-in by 6-in rough cut timber. Keeping the pipe as straight as possible ensured that excessive wear did not occur at sharp bends between pipe sections.

Total tonnage of backfill placed in the tunnel using ¾-in topsize material was 1,450 st, or about 60 pct of the total void of the tunnel. The remaining void was filled with ¼-in topsize material. No observable differences were determined regarding the advantages of one material over the other.

The project took 26 working days to complete. The face advance of the backfill pile in the tunnel is plotted in daily advance versus working day in figure 19. The face advance was measured on the center line of the tunnel at 3 ft up from the floor on a daily basis. The plot shows a gradual improvement in face advance rate or the total tons of material stowed per day. Changing material size had no effect on the stowing rate. Improvement can be attributed almost entirely to solving the wear problems and improving the method of using the wood supports. On the final setup from Opening 2 at the upper end of the tunnel, the wood supports were arranged to allow the pipe to slide from one rib to the other on a horizontal beam supported at each end by wood supports as shown in figure 18. This allowed very fast and simple adjustments of the pipe during stowing, and meant that two workers could very quickly and easily move the pipeline to a desired orientation.

Project Cost

This section presents the actual and projected costs for this job and future similar stowing jobs. This information may provide a guideline for bidding future projects that use the tested equipment. As with all field projects there are site specific elements that cannot be predicted. This includes primarily site setup and excavation prior to stowing equipment installation. The actual costs are higher than the projected costs. The projected costs are based on ideal conditions with an experienced operator operating the system.

The stowing rate was measured by the summing of the weigh tickets from the trucks delivering the fill and dividing by the actual stowing time logged. The actual stowing rate for the equipment during the demonstration was 21 st/h. During the later phases of the program however, the rate increased to 25 st/h. This figure is more typical of the performance that can be expected from the equipment when experienced operators are running the system.

The cost on a per ton basis for the project based on idealized operating conditions was $9.88 per st. Actual cost for this project which includes the down time for measurements and for inefficiency due to learning how to optimize the operation was $19.09 per st. Not included in the costs are the cost of the backfill gravel and the costs associated with the two pneumatic stowing devices which do not effect the comparison.

Conclusions and Recommendations

This demonstration project showed that the two pneumatic devices could cost effectively be used to stow material in abandoned mines. The quality of the stowed material was very dense which should mean that resistance to subsidence is good and very little, if any, movement of the roof will occur before the backfill material begins to take load.

The ejector showed no signs of wear at the end of the project. The pipefeeder needed repairs due to wear after 1,200 st of gravel had been processed. The worn areas were repaired and the unit lasted for the rest of the job.

It should be noted, however, that the Pneumatic Pipefeeder used for this project was a unit that had been previously used in extensive testing for a prior project, so much of its useful serviceability had been previously utilized. The Pneumatic Pipefeeder does not have an infinite life capability, and should be considered an expendable item. The low fabrication cost of the Pneumatic Pipefeeder should not make this an unacceptable feature.

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